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JOSEPH A. HOLMES, DIRECTOR

THE SMELTING OF COPPER ORES IN THE ELECTRIC FURNACE

BY

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AND

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THE SMELTING OF COPPER ORES IN THE ELECTRIC FURNACE.

By Dorsey A. Lyon and Robert M. Keeney.

INTRODUCTION.

This bulletin is one of a series dealing with the application of the electric furnace to the smelting of ores and the manufacture of alloys, and is published by the Bureau of Mines in the endeavor to increase efficiency in metallurgical processes.

The bulletin presents, first, a critical discussion of the possibility of smelting copper ores in the electric furnace; second, the results of the experimental work of other investigators on the electric smelting of copper; third, the results of experiments by the authors on the electric smelting of native copper concentrates and sulphide copper ores; and, fourth, a comparison of the electric furnace with the blast furnace and reverberatory furnace for copper smelting.

Before considering the electric smelting of copper ores the reader should understand that the electric furnace was not developed as a competitor of the combustion furnace, but for the purpose of doing high-temperature work that can not be done in the combustion furnace, and for the treatment of ores from deposits that are in regions where fuel is scarce and costly but hydroelectric power is comparatively cheap, as in Chile, and in certain parts of Canada, the United States, and Mexico.

Furthermore, it is not the object of this report to try to prove that the electric furnace should replace the reverberatory or the blast furnace, as used at present in smelting copper ores, but to show that the electric furnace may be used with advantage in localities where conditions are not favorable to the use of the reverberatory or the blast furnace.

For example, otherwise valuable ore deposits may be situated in a district so remote from, or inaccessible by, railway that the cost of getting coke for smelting, or of transporting the ore to a smelter, is prohibitive. If there is sufficient water power at hand, from which electric power can be developed at a reasonable cost, and this electric power is used for smelting the ore in an electric furnace, with subsequent bessemerizing, if necessary, the concentrated product may be valuable enough to render treatment of the ore possible. Or, the deposits may be accessible by railroad and yet the cost of fuel may be too great for smelting by the ordinary methods, whereas, on the other hand, hydroelectric power can be developed in the district at a low cost. By the use of the electric furnace such ores may be smelted cheaply enough to make their treatment feasible.

The possibilities being as stated, a brief comparative study of the problem is presented for the purpose of aiding those interested in the subject in determining whether it would be possible metallurgically, and feasible commercially, to use the electric furnace in those localities where, by reason of costly fuel, the cost of operating a reverberatory or blast furnace for smelting copper ores is either excessive or prohibitive.

The main points to be considered regarding the electric smelting of copper ores are whether the ores can be treated as successfully in an electric furnace as in a combustion furnace, and if so, whether they can be smelted at a reasonable cost.

CHEMISTRY OF COPPER SMELTING.

The objects sought in copper smelting may be summarized as follows: (1) To concentrate the copper into metal carrying any precious metals that may be present, if a native copper or an oxide ore is to be smelted, or into a matte, if a sulphide ore is to be treated, and (2) to remove, as slag, the excess of iron and gangue materials.

In general, there are three classes of ores that have to be considered in copper smelting, namely, native copper, the oxides, silicates, and carbonates, and sulphide ores.

SMELTING OF NATIVE COPPER ORES.

From the manner of occurrence of native copper, the smelting of native copper, or a native copper concentrate, resolves itself into the melting of the ore, or concentrate, and the separation of the metallic copper from the gangue through the difference in their specific gravities. After removal of the slag a further refining of the black copper obtained from the melting is necessary. The smelting of native ore is the simplest form of copper smelting. The electric furnace is especially adapted to simple melting in a neutral atmosphere, and appears to have particular application in this phase of copper smelting, especially the smelting of fine concentrates, as shown by recent experiments of the writers.

SMELTING OF OXIDE, CARBONATE, OR SILICATE ORES.

The process of smelting oxide, carbonate, or silicate ores of copper consists simply in melting the ores in the blast furnace with coke, thus reducing them to metallic copper. The separation of the slag and metal is an important feature of the process and the supply of coke must be carefully regulated to prevent reduction of iron by any excess carbon. The furnace atmosphere is reducing and not neutral. Such an atmosphere is easily obtainable in the electric furnace, and it is used in the electric smelting of iron. The experiments of Stephan, to be described later, also show the possibility of using a reducing atmosphere in the electric furnace.

SMELTING OF SULPHIDE ORES.

The smelting of sulphide ores presents additional complications. In order to concentrate the copper and any precious metals into a matte, it is first necessary to eliminate from the ore the sulphur, or as much of it as may be necessary, in order to obtain the desired results, and to cause the resultant oxides to unite with silica, which may either be present in the ore or may be added, preferably in the form of siliceous ore, to make a slag.

REMOVAL OF SULPHUR.

As pointed out by Peters,[a] when fused in a reducing atmosphere certain of the sulphides, such as Cu_2S, FeS, PbS, etc., melt down without change, while the very common and important mineral, pyrite, loses about one half of its sulphur, which is driven off as sulphur in the shape of yellowish fumes, which will deposit a coating of brimstone on a cool surface.

Peters also states that—

> The only two sulphides of much interest to the metallurgist, which lose a portion of their sulphur by heat alone in the manner just described, are (1) pyrite (FeS_2), which is assumed to lose one-half of its contents in sulphur by mere heating without air; and (2) chalcopyrite ($Cu_2S.Fe_2S_3$) which (as generally assumed by metallurgists) is composed of one-third copper, one-third iron, and one-third sulphur, and loses (by melting without air) one-third of its sulphur, or one-ninth of its entire weight.

This quotation is given in order to show clearly and concisely that there would be no object in simply melting sulphide ores in an electric furnace. That is, if this were done, only an altered form of the ore would result, somewhat concentrated, to be sure, through the loss of the elemental sulphur, but not concentrated enough.

RESULTS OF SMELTING RAW SULPHIDE ORE IN A BLAST FURNACE.

Peters gives the following example in order to illustrate the concentration effected by smelting in a blast furnace:

> Assume that you wish to smelt 100 pounds of a sulphide ore containing no earthy gangue rock, and consisting solely of a mixture of pyrite, chalcopyrite, and a little galena. It assays 35 ounces silver per ton and carries 6 per cent copper and 4 per cent lead. Neglect for the moment the inevitable metal losses in smelting, and assume (as the practical metallurgist generally does) that the pyrite consists of one-half

---

[a] Peters, E. D., Principles of copper smelting, 1907, p. 109.

iron and one-half sulphur, and that chalcopyrite consists of one-third each of copper, iron, and sulphur.

He then makes the necessary calculations to determine the weight of the matte that would be produced if this ore were to be smelted in a blast furnace, and from the results obtained constructs the following table:

Theoretical weight of matte produced by smelting 100 pounds of mixed sulphides.

Mineralogical composition of ore.	Weight of ore.	Chemical composition of ore.				Loss by smelting (S).	Remaining to form matte.				Total weight of matte.
		Fe.	Cu.	S.	Pb.		Fe.	Cu.	S.	Pb.	
	Lbs.	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*
Pyrite	77.36	38.68	......	38.68	......	19.34	38.68	......	19.34	......	58.02
Chalcopyrite	18.00	6.00	6.00	6.00	......	2.00	6.00	6.00	4.00	......	16.00
Galena	4.64	......	......	.64	4.00	......	......	......	.64	4.00	4.64
Total	100.00	......	......	......	......	21.34	44.68	6.00	23.98	4.00	78.66

As stated by Peters, this calculation shows that if 100 pounds of mixed sulphides were to be smelted in the blast furnace the matte would weigh 78.66 pounds, or a concentration of $100 \div 78.66 = 1.27$ would be obtained, and only 1.27 tons of ore would be smelted into 1 ton of matte, which is an unsatisfactory and unprofitable degree of concentration.

METHODS OF SMELTING SULPHIDE ORES.

Such being the case, the smelting of sulphide copper ores resolves itself into one of the three following methods, or a combination of them, in order to effect the removal of sulphur and a desirable concentration:

(1) Oxides and sulphides are mixed in such proportions that the reactions necessary to effect the desired degree of concentration will take place.

(2) Sulphide ores are roasted, thus converting them into oxides, or partly so, depending on the degree to which the roasting is carried out, and are then melted in a suitable furnace. By the preliminary removal of sulphur to the desired degree, the excess iron and the gangue materials present may then be eliminated in the furnace as slag, for it is to be remembered that sulphur which is present in the charge during the melting will combine with the copper, and that any sulphur in excess of the amount necessary to satisfy the copper will unite with the iron; the remainder of the iron present as iron oxide (FeO) will unite with silica to form slag. The concentration of the copper and precious metals into a matte is thus brought about.

(3) As roasting the ore is an additional step in the process of concentration it is to be avoided if possible, and the desired oxidation of the sulphides accomplished in the blast furnace by means of the

oxygen of the air that is forced into the furnace through the tuyères. This is what the modern copper blast furnace does; it combines the functions of roasting and smelting. In other words, it oxidizes and smelts simultaneously, and also utilizes the heat that is generated by the oxidation of the sulphur and iron. This utilization of the heat generated by the oxidation of the iron and sulphur is termed pyritic smelting, although, as will be shown later, true pyritic smelting involves the smelting of sulphide ores solely by the heat arising from their own oxidation, without the addition of any carbonaceous fuel.

REACTIONS IN THE BLAST FURNACE.

In order to illustrate what takes place in a blast furnace smelting sulphide ores, assume that the charge is made up of silica, pyrite, and chalcopyrite. The shaft may be divided into two zones, namely, the preheating zone, and the oxidation and fusion zone, it being understood, of course, that there is no sharp line of demarcation between the two.

Reactions in preheating zone.—The behavior of the main constituents of the charge in passing through the preheating zone is as follows:

(1) Silica (SiO_2) remains unaltered.

(2) Chalcopyrite ($Cu_2S.Fe_2S_3$) loses about one-fourth of its sulphur as elemental sulphur and becomes a mixture of Cu_2S and FeS.

(3) Pyrite (FeS_2) loses about one-half of its sulphur and becomes FeS and then, finally, Fe.FeS.

Reactions in oxidation and fusion zone.—The reactions that take place in the oxidation and fusion zone are as follows:

(1) Chalcopyrite (Cu_2S and FeS compound) melts as soon as it has descended to a point where the temperature is high enough, a comparatively moderate temperature.

(2) The sulphur in the pyrite (Fe.FeS compound) is partly burned by the oxygen of the blast to SO_2. That which remains unburned from lack of oxygen passes through the oxidizing zone and enters the matte.

REVERBERATORY COPPER SMELTING.

The objects sought and the methods used in the smelting of copper ores have been outlined. The next point to be considered is the manner in which the smelting of copper ores is conducted.

Inasmuch as copper ores are smelted in one of two ways—that is, either in a reverberatory or in a blast furnace—it will first be necessary to ascertain if the conditions essential for the successful conduct of these two methods can be attained in the electric furnace.

As stated by Peters,[a] "the one fundamental and preeminent duty of the reverberatory type of furnace is to produce a smelting temperature so that the ore may become liquid, and thus be able to separate into slag and matte." As regards this discussion, the reverberatory furnace is used for the smelting of ores and concentrates that can not be successfully treated in a blast furnace. As Peters has shown:

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 170.

> The manner in which the heat from the fuel is applied in the reverberatory method of smelting, though much improved in modern practice, is a peculiarly wasteful one, as the mere rapid passing of a flame over the surface of a layer of ore resting upon a comparatively cool hearth is a particularly incomplete way of transferring heat from the flame to the ore particles, especially as the latter are usually poor conductors of heat.
>
> The very large surface of walls, arch, flue, stack lining, etc., is in just as favorable a position for being smelted as is the ore itself, and in spite of its refractory composition, needs frequent repairs and renewals.
>
> Such a method could only originate in a district possessing cheap flaming coal and inexpensive refractory materials, and Swansea (where the process originated) has these in abundance.
>
> The reverberatory process is but little complicated by the active extraneous agents which exert so marked an effect upon the chemistry of the blast-furnace smelting of roasted ores, and the pyrite smelting of sulphide ores, namely, carbon in the one case and oxygen in the other.
>
> Speaking in a broad general sense the ore in the reverberatory smelting furnace is subjected to neither of these influences to any marked degree, and it is thus enabled to work out its own salvation under the influence of heat alone. Such chemical reaction as takes place in the ore mass during the smelting results almost entirely from the behavior of substances already contained in the ore and is affected by the mutual action of the various constituents of the charge itself. This renders the chemical part of the operation unusually simple and enables the smelter to concentrate his efforts upon developing the high temperature essential to the complete fusion of the ore.

SUBSTITUTION OF THE ELECTRIC FURNACE FOR THE REVERBERATORY FURNACE.

If, then, as Peters states, "the proper function of the reverberatory furnace, in smelting copper ores, is to generate heat as rapidly as possible," there is no apparent reason why copper ores can not be smelted equally well in an electric furnace, and perhaps even more advantageously; for, as pointed out by Peters, and as is well known, the thermal efficiency of the average reverberatory furnace is only 5 to 8 per cent, although an efficiency of about 20 per cent is obtained in the large reverberatory furnaces at Anaconda, whereas an electric furnace for smelting copper ores that are suitable for reverberatory smelting would probably have an efficiency of 70 per cent. Chemically, the only possible disadvantage in the use of such a furnace would be the effect of the carbon of the electrodes upon the iron oxide in the charge with which the electrodes might come in contact, and which would result in (1) the loss of electrode; (2) the production of metallic iron, which, if metallic copper was desired, would cause a low-grade product, and increase the cost of refining, or if a matte was desired might necessitate reconcentration; and (3) the loss of electrical energy, due to the additional heat consumed if there were much reduction of iron by the carbon of the electrodes.

As the reverberatory furnace, whether used for smelting sulphide, oxidized, or native copper ores, is essentially a means of melting, for comparison with the reverberatory, the results obtained in the smelting of native copper ores in the electric furnace will serve to clear up for sulphide or oxide ores the three points above mentioned—namely, the loss of electrodes, the production of metallic iron, and the loss of electric energy.

The slags would be somewhat similar in each case. The atmosphere of an electric smelting furnace charged with sulphide ore would not be as neutral as that of an electric furnace charged with native copper ore, because the liberation of sulphur and the formation of sulphur dioxide tend to dilute the air and to reduce its oxygen content, making it sufficiently reducing to prevent the consumption of the electrodes by oxidation. If the electric furnace was not kept closed, considerable sulphur dioxide might be formed, which would possibly be reduced in part by the carbon of the electrodes. Even with any reduction of sulphur dioxide the electrode consumption would be probably less for sulphide ores than for native copper ores. The results of experiments by the authors confirm this statement. In a furnace charged with native copper concentrates of the grade given in Table 2, the maximum electrode consumption in 14 experiments was 48 pounds of graphitized carbon per ton of charge, the minimum was 22.1 pounds, and the average was 34.5 pounds. In the same furnace, with mixtures of sulphide, oxide, and siliceous ores, a matte being the product, the maximum electrode consumption for 20 runs was 60 pounds, the minimum was 10.8 pounds, and the average was 23.3 pounds.

SMELTING OF NATIVE COPPER ORES.

METALLURGICAL PRACTICE AT MICHIGAN SMELTERS.

Copper smelting practice in Michigan differs considerably from that of smelters in the Western States, in that instead of the production of matte and eventually blister copper from sulphide ore, copper [a] is produced directly by melting concentrates and masses containing native metallic copper. The black copper obtained is subsequently refined, and the slags obtained during the operation are resmelted.

CONCENTRATION METHODS.

The concentrates are obtained from the concentration of conglomerate or amygdaloid ores. Before concentration these ores contain from 1 to 3 per cent of copper and some native silver. The amygdaloid is high in silica, whereas the conglomerate contains more iron than silica. The milling treatment preliminary to smelting consists of crushing with breakers and steam stamps or Hardinge mills and concentration on jigs and tables until a coarse concentrate is obtained containing 60 to 80 per cent copper. Most of the coarse native silver is removed in concentration. Large masses of copper are cleaned of gangue and smelted directly. At one smelter these concentrates were separated into three grades:[a] No. 0, mass copper, containing from 65 to 70 per cent copper; No. 1, coarse concentrate and some mortar discharge, averaging about 75 per cent copper; and No. 2, fine concentrate, containing 10 to 35 per cent copper. This research was conducted on material resembling the No. 2 grade.

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[a] Conant, H. D., Copper smelting in Michigan: School of Mines Quart., vol. 32, 1911, p. 285; Canadian Engineer, vol. 21, 1911, p. 246.

SMELTING PROCEDURE.

There are two general methods of procedure in smelting these concentrates. In one method the mass copper and coarse concentrate are charged into a reverberatory furnace and melted down without any fluxes; the slag is skimmed off as fast as it is formed and the copper tapped into another reverberatory furnace for refining. In the second method melting and refining are carried on in the same furnace. Formerly the fine concentrate, No. 2, was mixed with lime and briquetted for smelting in the blast furnace, but now at some smelters this material is first agglomerated in a reverberatory furnace and later smelted in the blast furnace.

Up to 1904 the largest reverberatory melting furnaces in operation in the district had a hearth area of 17 by 28 feet and a capacity of 200,000 pounds per charge, which, because of the cycle of operation, is practically the capacity for 24 hours.[b] At present furnaces are in use having a hearth area of 16 feet by 35 feet and a capacity of 300,000 pounds per charge. The largest reverberatory refining furnaces have a hearth 13 feet wide by 17 feet long.

METHOD OF OPERATING REVERBERATORY FURNACE.

The cycle of operation in smelting native ore is as follows:

A reverberatory furnace is charged, say, at 4 p. m., with concentrates and mass copper. After slag begins to form the charge is skimmed every 2 hours for a period of 15 hours, after which the bath is oxidized by blowing air on it or into it with an iron pipe as a carrier or tuyère. This oxidation removes arsenic and other impurities and oxidizes some of the copper to suboxide of copper, Cu_2O. The molten bath at this time contains about 95 per cent of metallic copper and about 5 per cent of cuprous oxide. The oxide is then reduced by poling with green wood and charcoal. Casting is commenced about 1 p. m. and finished about 3 p. m. The furnace is repaired if necessary and is ready for the next charge at 4 p. m.

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[a] The western metallurgical field; copper smelting in Michigan: Met. and Chem. Eng., vol. 11, 1913, p. 12.

[b] Austin, L. S., Recent copper smelting at Lake Superior: Eng. and Min. Jour., vol. 81, Jan. 13, 1906, p. 83.

The completion of the process of oxidizing with air is determined by the electrical conductivity and fracture of a test piece. Copper refined from native copper concentrates must be coarsely crystalline to have a conductivity of 100 per cent, whereas electrolytic copper of 100 per cent conductivity may be only finely granular. Poling is complete when a test button of the metal sets with a level surface.

TREATMENT OF SLAGS FROM FURNACE.

The slags from the reverberatory furnace have a very high copper content, but if an attempt is made to reduce it the purity of the product is impaired. The slags have to be retreated in any case, so the percentage of copper in them makes little difference. Reverberatory slags contain from 10 to 30 per cent copper, the content being higher toward the end of the skimming process. Slag from amygdaloid ore is acid and contains 40 per cent SiO_2 and 20 per cent FeO, whereas slag from conglomerate ore is basic and contains 25 per cent SiO_2 and 45 per cent FeO.

These reverberatory slags are remelted in a low-pressure blast furnace with anthracite coal or coke as reducing agent and fuel, and limestone, iron oxide, or silica as a flux. The furnace is run with a slag containing 33 per cent SiO_2, 33 per cent FeO, and 20 per cent CaO, which is a fusible and liquid slag. The slag runs continuously from the furnace and contains 0.6 to 0.8 per cent copper. If the production of a cleaner slag is attempted the iron content of the product is increased, owing to the extra reducing material and higher temperature used. The metal produced contains 90 to 95 per cent copper and is retreated in a reverberatory furnace.

The blast furnace is rectangular in cross section, being 40 to 48 inches wide at the tuyères and 8 to 12 feet long. The crucible is lined with fire brick incased in a steel shell, and the shaft, which is 7 to 10 feet high, consists of steel water jackets. No forehearth is used. A blast pressure of 3 to 4 ounces is used with a production of about 72 tons of copper per 24 hours. With a higher pressure there is greater loss of copper in the slag, owing to more rapid driving of the furnace.

The refined product from the reverberatory furnace is cast into various sizes and shapes of ingots for marketing, but if it contains much silver it is cast into anodes for electrolytic refining.

EXPERIMENTAL SMELTING OF FINE MICHIGAN NATIVE COPPER CONCENTRATES IN THE ELECTRIC FURNACE.

For the purpose of investigating the feasibility of smelting fine Michigan native copper concentrates in the electric furnace instead of the reverberatory furnace, the following experiments were conducted by the writers.

The objects of these experiments were: (1) To note the effect of variation of the proportions of the three main slag constituents—silica, ferrous oxide, and lime—on the power consumption, the operation of the furnace, and the purity of the copper produced; (2) to ascertain the extent of iron oxide reduction caused by the graphite electrode; (3) to compare the results of intermittent smelting with those of continuous smelting; and (4) to study the feasibility of using the slag produced in copper smelting for the manufacture of ferrosilicon in the electric furnace.

GENERAL DESCRIPTION OF EXPERIMENTS.

The experiments were divided in six series, based on different variables in the slag-forming components of the charge. Fourteen experiments were made in all, grouped as follows: In series 1, the slag was formed from the natural gangue of the concentrate; in series 2, the acidity of the slag was varied by the addition of silica; in series 3, the basicity of the slag was varied by the addition of Lake Superior hematite; in series 4, lime was used instead of iron ore as a base; in series 5, the basicity of the slag was varied by the addition of lime and hematite; and in series 6, the furnace was charged at intervals and kept full during the run, and the metal tapped at intervals.

In all of the experiments the furnace was operated as a resistance furnace as far as possible in order to keep down the volatilization of the copper. There was arcking at the start of a run before the charge became molten, and in several runs arcking occurred at intervals during the experiment, but not for a long period of time. In all experiments, except those of series 6, the furnace was charged intermittently—that is, after starting the charge was added gradually—and all of it was completely melted before tapping. As far as possible, the furnace was tapped cleanly after each run. Any slag left in the furnace after tapping was removed by balling in the furnace in a manner similar to that used in the puddling process. Slag adhering to the walls was allowed to remain there.

In the operation of an experimental electric furnace, except on a very small scale, where tapping cannot be done, it is not advisable to go to extremes to obtain an absolutely clean discharge of the molten products. To remove entirely all metal and slag, especially slag, one of two procedures must be followed: Either the charge, after becoming molten, is superheated to remelt any material that has adhered to the cooler parts of the walls, or, when the furnace has cooled off, this material is chiseled off. Either procedure results in a partial or complete destruction of the furnace lining, making a new one necessary. If the charge is superheated, the high temperature results in the brick walls becoming pasty and gradual dripping of the softened brick into the slag. One brick may be more exposed than the others and gradually be completely melted, and, aside from causing injury to the lining, materially alter the composition of the slag.

The prolonged heating may also cause volatilization of the copper, although this is not so likely to occur after the copper has been smelted down and covered with a bath of slag, as during the melting period. The prolonged heating also caused a much higher power consumption than would occur in practice, as in practice the accretions and the hangings would not affect this item after the furnace was running regularly and continuously. Chiseling out the furnace is impractical, because of the time consumed and on account of the destruction of the lining. For these reasons the furnace was built with a high degree of slope on the bottom toward the tap hole, and tapped as cleanly as possible through this hole. As a result, some of the runs show a considerable deficiency in the quantity of metal tapped as compared with the calculated amount in the charge, and some an excess; although the greatest discrepancy is in the slag. In some runs the amount of material tapped checks very closely with that charged.

DESCRIPTION OF FURNACE USED.

An electric furnace of the Siemans type is the best suited for laboratory experimental smelting of ores. A furnace of this type, as shown in figures 1 and 2, was used in these experiments.

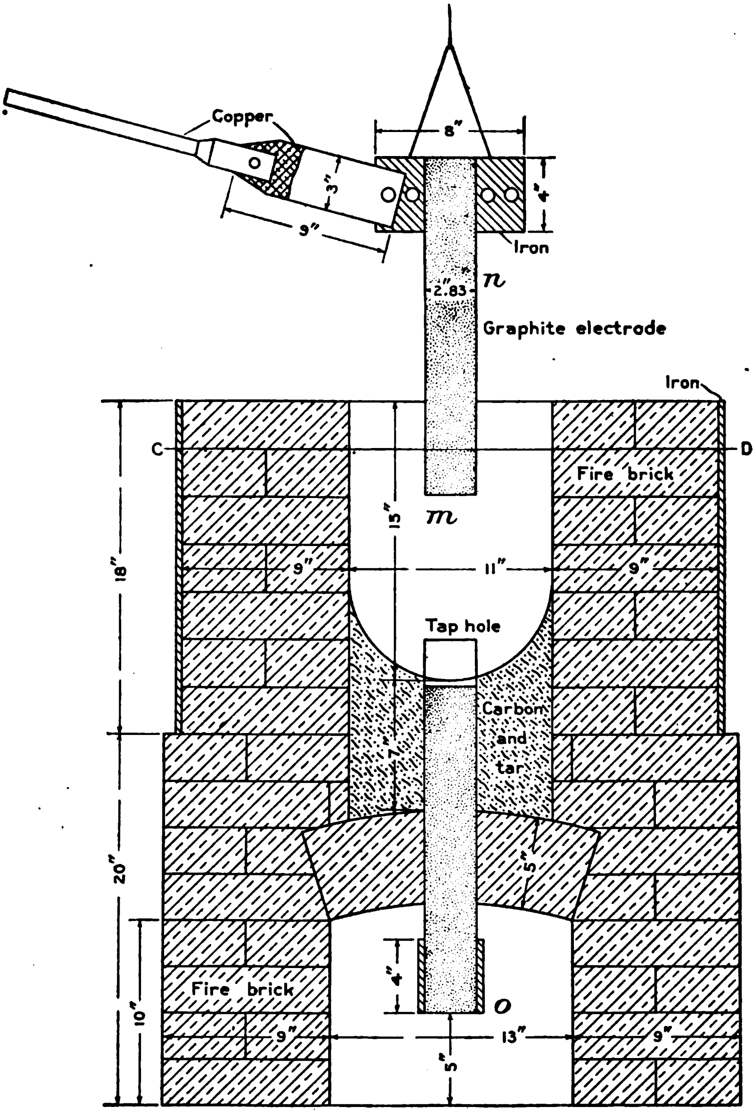
The furnace had a square crucible, *m*, lined with fire brick. The current passed into the crucible through an upper graphite electrode, *n*, then through the charge, and then through another graphite electrode, *o*, placed in the bottom of the furnace, which was made of carbon. The exterior shell of the furnace was of fire brick. The bricks were held together by iron bands, which could be adjusted for expansion. The crucible was roofed with fire-clay tiles during some experiments, in others the roof was discarded. The upper electrode was hung so as to extend vertically into the center of the crucible. No attempt was made to keep the joints in the roof tight, so that gas escaped around the electrode. Part of the tile roof was put on before charging and the opening was completely closed when all of the charge had been added. There was one tap hole at the bottom of the crucible, in the middle of one side.

THE CRUCIBLE.

The crucible was 11 inches square and its walls were vertical. The carbon bottom was tapered up about 4 inches around the walls, so that the lowest part of the bottom was about 4 inches square. The depth of the crucible was 15 inches on the average, but varied a little with different linings. The total volume of the crucible was approximately 1,600 cubic inches, but this was varied slightly from time to time as convenient.

WALLS AND CRUCIBLE SUPPORT.

The crucible was supported on an arch of fire brick set on two brick bases, each 2 feet 7 inches by 9 inches in section, and 10 inches high to the base of the arch. The arch was of 1 foot 1 inch span. The outside dimensions of the furnace at the bottom were 2 feet 7 inches by 2 feet 7 inches. These dimensions extended up to a height 1 foot 8 inches above the floor, on a level with the end of the tap hole,



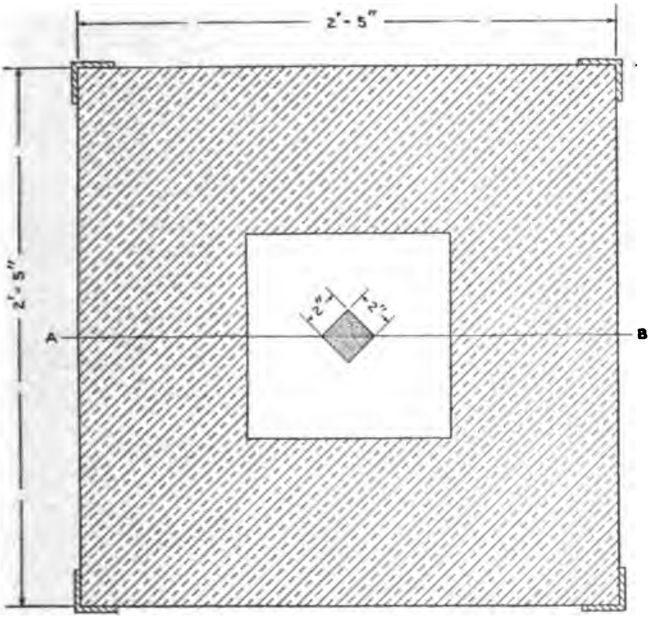
SECTION A-B

FIGURE 1.—Elevation of experimental furnace.

which was 4 inches lower than the bottom of the crucible. The brick walls surrounding the crucible were 2 feet 5 inches square and 1 foot 6 inches high, making the total height of the furnace 3 feet 2 inches. An iron plate would have been better than the arch of brick, but to have procured one would have involved considerable expense and loss of time.

ELECTRODES AND HOLDERS.

The upper electrode was of Acheson graphite and was 2 inches square and 36 inches long. The lower electrode was of the same cross section and material but was 18 inches long. The part above the top of the arch was embedded in a mixture of ground electrode stumps, or coke, made plastic with tar. This material covered the upper end of the electrode to a depth of 1 inch and extended up on the walls of the furnace. The lower end projected below the arch to within 5 inches of the floor. The electrode was supported at first by an iron clamp embedded in the bricks but later simply by pressure of the bricks and the carbon mass around it. The cross-sectional area of the electrode was, of course, 4 square inches. Current was conducted to the electrode by two insulated copper cables 1 inch in diameter. The cables extended 30 feet to four copper bus bars 4 inches by 1 inch in cross section, which extended 30 feet to the switchboard where all electrical readings were taken.



SECTION C-D

FIGURE 2.—Plan of experimental furnace.

The upper electrode holder consisted of two iron plates 4 inches wide by 8 inches long bolted together around the electrode. During some of the experiments this holder was connected to the circuit by a plate which held the loose wire of the connecting cable against the holder. This connection did not work well with high amperages, owing to short circuits burning out the wires and on account of the heating of the holder from the heat of the crucible. Later a copper plate 3 inches wide and 9 inches long was bolted to the iron holder and connected at the other end with a lug soldered in the cable. This arrangement gave complete satisfaction, as the lug was far enough from the crucible to prevent melting of the solder. The bottom connection was made by clamping the loose ends of the cable between the holder and a plate bolted to the holder, as was first used for the upper holder.

REGULATING DEVICE.

The upper electrode was suspended by flexible copper wire, running over a pulley attached to a steel crosspiece, which was set on a wooden framework 7 feet 6 inches high. The wire was then wound around a loose horizontal pipe, set at the rear of the furnace, and the electrode counterbalanced by attaching weights to the wire. This makes a simple and satisfactory device for hand regulation of electrodes in small furnaces, very close adjustment being possible.

For holding tight the upper section of the furnace, 2-inch channel irons were set on the corners and around them were placed two 1-inch strips of sheet iron.

APPARATUS AND INSTRUMENTS.

Single-phase current was received in the laboratory at a voltage which could be varied on the primary side from 440 to 550 volts with a frequency of 60 cycles. The voltage was stepped down by a 75 kilovolt ampere, single-phase transformer. The primary windings of the transformers were so adjusted and connected with switches and a voltage regulation box that the voltage could be regulated within three ranges, as follows: High voltages, ranging from 120 to 300 volts, by 12-volt steps; medium voltages, 60 to 150 volts, by 6-volt steps; and low voltages, 30 to 75 volts, by 3-volt steps. On the primary side there was a 300-ampere rheostat and a main switch fitted with a spring. On the secondary side there were two main switches and numerous voltage regulation switches.

The instruments used in taking readings were a Thomson high-torque, watt-hour meter set in the primary circuit; a Keystone indicating wattmeter, with a range of 100 kw., connected in the primary circuit; two Keystone alternating-current ammeters, with a range of 200 amperes, connected in the primary circuit, and a Keystone alternating-current voltmeter, the lower scale having a range of 100 volts and the upper scale a range of 500 volts, connected for both primary and secondary circuits. These instruments were set on a switchboard so situated that the current was conducted from 60 to 80 feet by bus bars and cables to the furnace.

Tappings were made in cast-iron molds, 19 inches long by 5 inches wide at the top, 17 inches long by 3½ inches wide at the bottom, and 4½ inches deep, with sloping sides 1 inch thick.

CONCENTRATES SMELTED.

The concentrates treated were practically subdivisions of grade No. 2, previously mentioned, and will be designated as No. 3 and No. 4. The coarser of the two grades, No. 3, is obtained by concentrating ore as follows: The rock is crushed with Blake breakers, stamped under a steam stamp; passed through a trommel with ¼-inch openings to a hydraulic classifier and to two jigs in series, the tailings from the last jig passing over a Wilfley table, the product of the table being grade No. 3. The overflow from the hydraulic classifier is conducted to a settling tank, and the slimes obtained are then treated upon a Card table, which gives grade No. 4. The copper in grade No. 3 was rather coarse and plainly visible; in No. 4 it was so fine that it could be detected only with difficulty by the naked eye, although by panning it was very easily concentrated to an appreciable extent. There was, however, considerable loss by sliming.

The sieve analyses of these concentrates are as follows:

Sieve analyses of Michigan copper concentrates.

Fineness.	No. 3.	No. 4.
	Per cent.	*Per cent.*
Less than 10-mesh	0.80	0.03
10 to 20 mesh	5.33	1.38
20 to 40 mesh	14.20	8.32
40 to 60 mesh	9.90	3.13
60 to 80 mesh	8.37	4.05
80 to 100 mesh	29.14	19.88
Over 100-mesh	32.22	63.19
	99.96	99.98

The great difference in the fineness of the two grades is apparent from these analyses, 87.12 per cent of No. 4 passing through a 60-mesh sieve, whereas only 69.73 per cent of No. 3 passed 60 mesh. The fineness of the material, especially that of the copper, seemed to affect the results obtained in the experiments.

The silica used was a ground quartzite containing 96.82 per cent SiO_2. The lime was from a limestone containing 63.2 per cent CaO and 5.7 per cent MgO. A hematite ore of the analysis given below was used as a flux.

The chemical analyses of the concentrates and of the hematite were as follows:

Chemical analyses of concentrates and hematite.

Constituent.	Concentrate No. 3.	Concentrate No. 4.	Hematite.
	Per cent.	*Per cent.*	*Per cent.*
Cu	37.35	25.35	
SiO_2	33.33	25.92	5.19
FeO	15.40	21.60	
Al_2O_3	8.78	12.40	1.22
CaO	5.18	5.98	0.20
MgO	1.09	1.09	0.16
S	0.056	0.096	
P			0.61
Fe	0.30	0.50	51.68
Mn			0.49

As may be seen from the above analyses, both grades of concentrates were rather high in alumina. The iron, silica, and alumina were so proportioned in each as to give a rather infusible slag. The concentrates were not uniform in composition, the analyses given being the average from several samples. This variation in composition of the concentrate affected the results in some experiments in that the calculated amount of copper in the charge and that tapped do not check well.

PROBLEMS OF ELECTRIC SMELTING OF NATIVE COPPER CONCENTRATES.

Before describing the experiments, some of the problems that arise in connection with the smelting of native copper concentrates in the electric furnace will be discussed.

As previously stated, the treatment of these concentrates consists of a melting operation and the separation of the metallic copper and gangue by reason of the difference in their specific gravities.

LOSS OF COPPER.

The loss of copper by slagging and volatilization depends upon the melting point, fluidity, and specific gravity of the slag. The slag should have as low a melting point and be as fluid as possible in order to prevent volatilization of the copper. In the electric furnace there is always a tendency for the furnace to develop a higher temperature as the melting point of the charge is raised, so that charges which could not be treated in an ordinary metallurgical furnace are easily smelted. For this reason the furnace must be watched to keep the temperature low in treating easily volatilized metals. As long as there is slag covering the metal in an electrode furnace the loss is not excessive, but with an arc and no slag the copper is rapidly volatilized or oxidized.

The loss of copper in the slag depends, as stated above, on the fluidity and specific gravity of the slag. The loss in treating native copper concentrates in the electric furnace is practically entirely due to mechanical loss of minute metallic globules that are carried off because of imperfect settling. A fluid and liquid slag of low specific gravity prevents such loss.

REDUCTION OF IRON OXIDE.

The only reducing material present in the electric furnace during a simple melting operation would be the carbon of the electrode. As a result, the reducing action need not be very strong in operating an electric furnace for this purpose. If the slag has a low melting point, there is no reason why the reduction of FeO to metallic iron should be excessive. This factor would have greater influence in a small furnace than in a large furnace, where the electrode area exposed to the charge would not be so great in proportion to the amount of material charged. In order to lower the reducing action, the temperature should be kept as low as possible and still give a fluid slag. Also in the electric furnace the effect of the base used to flux the silica may result in a decreased reduction of FeO.

DESCRIPTION OF EXPERIMENTS.

SMELTING CONCENTRATE NO. 4 WITHOUT FLUX.

Two experiments were made in which No. 4 concentrate was smelted without the addition of anything to flux the gangue, the results of which are shown in Table 2, experiments Nos. 1 and 2.

EXPERIMENT NO. 1.

Before experiment No. 1 was begun the furnace was heated with gas for an hour and a half to burn the tar out of the lining. The furnace was then filled with concentrate, and the current turned on.

Throughout the experiment the furnace did not run well and it was difficult to keep it from arcking at times. A green flame burned at the top, due to the oxidation of some of the copper. Carbon monoxide gas was also given off, due to the partial oxidation of the electrode.

In about one hour the charge was ready for tapping. Instead of tapping into an iron mold, as was done in later experiments, the charge was tapped into a mold made of bricks. The furnace did not tap well, because the bottom had settled, so that some metal was left in the furnace, and was later removed by chiseling.

No copper could be seen in the slag except at the bottom of the mold, where small pellets had concentrated. The slag flowed out of the furnace readily, but was a little thick, and had a glassy appearance when cold. Some of the carbon bottom came up and mixed with the slag in a semifused mass. There was not enough tar mixed with the carbon to make it set properly.

The metal contained so much iron that it could be seen with the naked eye. Under the microscope, iron, copper, and blowholes were visible, but no cuprous oxide.

EXPERIMENT NO. 2.

The furnace was hot when run No. 2 was started. The furnace tapped cleanly, the slag was glassy and rather thick, but flowed from the furnace. It was free from shots of metal. One brick in the furnace was eaten out by the slag. The metal showed the same characteristics as that from run No. 1.

VARYING THE ACIDITY OF THE SLAG WITH SILICA.

Five experiments were made, using varying amounts of silica for the purpose of noting the general effect of a very acid slag. Slags Nos. 3, 4, 5, 6, and 7 (see Table 2) were made in this manner. In all of these experiments No. 4 concentrate was used with silica as a flux.

EXPERIMENT NO. 3.

In experiment No. 3, the furnace was started cold. The furnace ran well, almost entirely as a resistance furnace. The green flame noted in the two previous experiments was present. The lining was in such poor condition that the metal could not be tapped, but was removed after cooling. The slag was viscous and very glassy, but flowed from the furnace. The metal showed iron and blowholes, but no cuprous oxide.

EXPERIMENT NO. 4.

In experiment No. 4, the furnace was hot from the previous run. The furnace ran well throughout the run, as a resistance furnace. The slag was viscous and very glassy, but flowed from the furnace. No iron or cuprous oxide were visible in the metal, but it contained some blowholes.

EXPERIMENT NO. 5.

Before beginning experiment No. 5 the furnace was relined and heated with the air to burn out the tar in the lining. The charge was slow in melting and shots of metal floated in the slag. On tapping, these seemed to settle toward the bottom of the mass. The slag was very thick and glassy, but flowed from the furnace. During the run, the furnace was operated entirely as a resistance furnace. The metal, on being tapped into two round ladles used for molds, froze and had to be melted out. This accounts for the abnormally high percentage of iron in the metal.

Under the microscope iron and blowholes were clearly visible, but no cuprous oxide.

EXPERIMENT NO. 6.

At the start of experiment No. 6 the furnace was cold. The power supply was very unsteady, being off for 18 minutes throughout the run. On tapping, the slag was thick and glassy, but flowed from the furnace, and no copper could be seen which did not settle out. There was a large amount of iron visible in the metal under the microscope, but it was present in a more dendritic form than in previous experiments. No cuprous oxide was formed.

EXPERIMENT NO. 7.

In experiment No. 7 the furnace was hot before charging. The slag was sticky and thick, but flowed from the furnace and was free from shots. Under the microscope very little iron could be seen in a section of the metal, and no cuprous oxide at all. The metal was also comparatively free from blowholes.

VARYING THE BASICITY OF SLAG WITH HEMATITE.

In experiments No. 8 and No. 9 (Table 2) hematite was used to note the effect of a more basic slag and the value of iron oxide as a flux in comparison with lime.

EXPERIMENT NO. 8.

In experiment No. 8 the furnace was hot before charging. The furnace acted better than in previous runs, probably because of the slag being more fusible. The slag was very fluid on tapping. It was of a dull basic appearance and free from shots of copper. Neither iron nor cuprous oxide was visible in the metal under the microscope.

EXPERIMENT NO. 9.

The furnace was cold at the beginning of experiment No. 9. The slag was dull, basic, and very fluid and was free from shots of metal. Neither iron nor cuprous oxide was visible in the metal.

VARYING THE BASICITY OF THE SLAG WITH LIMESTONE.

EXPERIMENT NO. 10.

In experiment No. 10 limestone was used as a flux. The furnace was hot at the start and operated well throughout the run. The slag was very fluid and basic and was free from shots of metal. A very little iron was visible under the microscope, but no cuprous oxide. The power consumption was very low.

VARYING THE BASICITY OF THE SLAG WITH LIMESTONE AND HEMATITE.

In experiments No. 11, No. 12, and No. 13 the basicity of the slag was changed by using both limestone and hematite and was increased as far as was considered practical.

EXPERIMENT NO. 11.

In experiment No. 11 the furnace was started cold. The slag was very fluid and boiled out of the top of the furnace considerably. Iron was distributed rather evenly throughout the metal as dendrites, but no cuprous oxide could be detected.

EXPERIMENT NO. 12.

The furnace was cold at the start of experiment No. 12. During the run fine globules of copper floated in the slag, which boiled out of the furnace considerably. There was so much of this boiling and slipping of the charge that some of the limestone dust was carried off by the gases. The slag was a little thick and sticky, but flowed readily from the furnace. Very fine dendrites of iron were visible in the metal, but there appeared to be no cuprous oxide.

EXPERIMENT NO. 13.

Experiment No. 13 was started with a hot furnace, but a rather sticky and thick slag was produced, which boiled and slipped so much that there was considerable loss of limestone and ore dust. There were shots of metal in the top of the slag in the furnace, but these settled out in the mold. Under the microscope only a few pieces of iron were visible in the metal and no cuprous oxide.

CONTINUOUS SMELTING.

EXPERIMENT NO. 14.

In experiment No. 14 concentrate No. 3, with lime as a flux, was charged at intervals so as to keep the furnace full, and three tappings were made during a run of 2 hours and 40 minutes. The furnace operated well and very little gas was given off, so long as the top of the charge was not fused.

When the bath was tapped, considerable material hung on the walls. The slags were very fluid, but the first two tapped contained considerable metal because of semifluid material running out of the furnace with the slag. No iron or cuprous oxide could be detected in the metal from any of the tappings.

RESULTS OF EXPERIMENTS.

The results of each experiment were tabulated as shown in Table 1, which gives the results of experiment No. 10 in detail.

TABLE 1.—*Results of smelting concentrate No. 3 with limestone flux, experiment No. 10.*

Time.	Amperes.	Volts.	Kilowatts.	Power factor.	Remarks.
P. m.					
12.35	500	70	33	94.3	Furnace started hot after experiment No. 9.
12.40	630	72	45	98.8	Charge all added at 12.37 p. m.
12.45	680	71	48	99.2	All fused at 12.46 p. m.
12.50	600	78	46	98.0	Shots of metal in upper part of slag.
12.55	580	80	46	99.0	Tapped. Slag fluid and free from shots.

Charge:	
Michigan concentrate No. 3, pounds	22.0
Limestone, pounds	2.2
Length of run, minutes	20
Mean volts in the furnace	74
Mean amperes on the furnace	598
Mean kilowatts on the furnace	44
Mean power factor of circuit	.97
Kilowatt-hours	14.4
Electrode consumption per pound of metal tapped, pounds	.052
Electrode consumption per ton of charge smelted, pounds	28.2
Calculated weight of copper, pounds	8.21
Weight of metal tapped, pounds	8.13
Weight of slag tapped, pounds	16.09
Loss of copper in the slag, per cent	.33
Copper unaccounted for, per cent	3.17
Total loss of copper, per cent	3.50
Kilowatt-hours per pound of metal tapped	1.77
Kilowatt-years per ton of ore smelted	.15
Kilowatt-years per ton of charge smelted	.135
Analysis of metal:	
Copper, per cent	97.13
Iron, per cent	2.50
Sulphur, per cent	.35
Arsenic, per cent	.02
Analysis of slag:	
Cu, per cent	.17
SiO_2, per cent	35.82
CaO, per cent	15.45
MgO, per cent	7.09
FeO, per cent	25.20
Al_2O_3, per cent	17.10

REMARKS.—On examining a microscopic section of the metal, after etching with picric acid, a few spots of dendritic iron could be seen, but no cuprous oxide.

The complete results of experiments 1 to 14 are tabulated in Table 2.

TABLE 2.—*Results of smelting of fine Michigan native copper concentrates in the electric furnace.*

Item.	Series 1.		Series 2.					Series 3.		Series 4.	Series 5.			Series 6.			Average.
	1	2	3	4	5	6	7	8	9	10	11	12	13		14		
Composition of charge:																	
Concentrate No. 4, pounds	22.0	22.0	22.0	22.0	22.0	22.0	22.0	......	22.0	......	......	......	......		.....		......
Concentrate No. 3, pounds	......	......	......	......	......	......	......	22.0	......	22.0	19.6	19.6	22.0	5.5	97.1		......
Quartzite, pounds	......	......	2.2	2.2	4.4	4.4	4.4	......	......	......	......	......	......		.....		......
Hematite, pounds	......	......	......	......	......	......	......	1.1	2.2	......	1.98	4.18	6.6		.....		......
Limestone, pounds	......	......	......	......	......	......	......	......	......	2.2	1.98	4.18	6.6		9.71		......
Composition of metal:																	
Cu, per cent[a]	87.29	88.62	96.24	97.96	83.82	90.95	95.95	98.59	97.96	97.13	93.73	93.51	96.64	[b]99.49	[c]98.69	[d]97.77	......
Fe, per cent	12.50	11.10	3.50	1.75	15.90	9.25	3.75	1.10	1.90	2.50	6.20	6.30	3.10	[b].30	[c]1.04	[d]2.00	......
S, per cent	.19	.26	.24	.27	.26	.21	.30	.29	.12	.35	.15	.17	.24	[b].19	[c].25	[d].21	......
As, per cent	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02	[b].02	[c].02	[d].02	......
Composition of slag:																	
Cu, per cent	.40	.23	.31	.23	.15	.15	.10	.15	.02	.17	.54	.24	.46	[b]1.91	[c]1.60	[d].62	......
SiO_2, per cent	49.80	40.60	53.22	51.10	49.96	42.58	48.00	44.88	31.71	35.82	33.15	33.96	29.37	[b]34.01	[c]34.33	[d]34.62	......
CaO, per cent	13.25	9.07	12.40	8.78	10.10	8.80	9.00	6.92	10.10	15.45	17.60	20.04	19.20	[b]16.80	[c]17.50	[d]20.32	......
MgO, per cent	1.93	3.00	3.12	2.03	4.60	2.71	1.84	2.32	2.45	7.09	3.61	4.46	4.17	[b]2.95	[c]3.14	[d]2.86	......
FeO, per cent	15.25	21.15	11.58	25.50	19.90	29.00	26.25	29.80	37.95	25.20	29.00	27.60	31.25	[b]26.70	[c]27.30	[d]24.70	......
Al_2O_3, per cent	13.70	[e]23.82	20.33	10.45	16.10	16.48	14.99	15.34	14.89	17.10	16.28	11.75	14.10	[b]17.09	[c]14.83	[d]13.35	......
Length of run, minutes	50	30	30	24	25	23	22	23	23	20	25	30	26		100		......
Mean volts on the furnace	59	86	90	79	80	75	82	64	57	74	66	77	70		66		73
Mean amperes on the furnace	......	......	......	532	......	542	444	710	808	598	683	663	670		557		621
Mean kilowatts on the furnace	32	33	40	40	42	38	34	43	45	44	44	49	45		34		40
Mean power factor of circuit	......	......	......	.95	......	.95	.93	.94	.97	.97	.96	.97	.96		.94		.95
Kilowatt-hours	26.7	16.4	20	16	17 6	14.6	12.5	16.4	17.2	14.4	18.4	20.3	19.5		57.3		......
Electrode consumption per pound of metal tapped, pounds	.08	.16	.043	.12	.16	.12	.12	.052	.052	.052	.071	.071	.071		.036		.086
Electrode consumption per ton of charge smelted, pounds	48	41.9	21.8	33.25	41.9	33.25	33.25	28.20	28.20	28.20	34.20	34.20	34.20		22.10		34.5
Calculated weight of copper, pounds	5.58	5.58	5.58	5.58	5.58	5.58	5.58	8.21	5.58	8.21	7.31	7.31	8.20		36.30		......
Weight of metal tapped, pounds	[f]6.60	3.97	5.45	4.52	2.86	3.07	2.97	7.60	3.63	8.13	6.81	6.81	7.04	[b]11.0	[c]6.28	[g]15.28	......
Weight of slag tapped, pounds	13.58	21.10	16.22	21.30	17.95	15.82	22.20	14.42	16.40	16.09	16.20	16.60	15.03	[b]15.83	[c]16.55	[h]34.30	......
Loss of copper in slag, per cent	.97	.87	.90	.88	.48	.42	.40	.26	.59	.33	.54	.22	.85		2.12		.70
Copper unaccounted for, per cent	......	36.90	5.03	17.65	56.40	49.20	48.30	7.22	35.25	3.17	11.62	12.03	16.41		10.85		23.10
Total loss of copper, per cent	......	37.77	5.93	18.53	56.88	49.62	48.70	7.48	35.84	3.50	12.16	12.25	17.26		12.97		23.80
Kilowatt-hours per pound of metal tapped	4.05	4.13	3.67	3.53	6.30	4.75	4.19	2.13	4.73	1.77	2.68	2.96	2.77		1.76		3.51
Kilowatt-hours per ton of ore smelted	2,425	1,500	1,820	1,450	1,600	1,320	1,130	1,490	1,565	1,310	1,880	2,070	1,780		1,190		1,430
Kilowatt-hours per ton of charge	2,425	1,500	1,740	1,320	1,330	1,075	1,015	1,420	1,425	1,280	1,565	1,455	1,100		1,075		1,380

[a] Copper by difference.
[b] First tapping.
[c] Second tapping.
[d] Third tapping.
[e] Fire bricks melted into the slag.
[f] Copper left in furnace from previous run.
[g] Third tapping. Total metal tapped in all three tappings, 32.56 pounds.
[h] Third tapping. Total slag tapped in all three tappings, 66.80 pounds.

DISCUSSION OF RESULTS.

As previously stated, the three most important points for consideration are: (1) The loss of copper by slagging, volatilization, or by dust being carried off in the gases; (2) the reduction of the iron oxide to metallic iron which enters the copper; and (3) the power consumption and general operation of the furnace.

All of these factors are largely affected by the melting point of the slag and the temperature necessary to make it liquid enough to flow readily from the furnace. The results obtained from the experiments just described can be used as a basis for discussion in considering the commercial application of the electric furnace to the smelting of these concentrates.

COPPER LOSSES.

In order to have a uniform basis for comparing the results obtained by varying the silica, lime, and ferrous oxide contents of the slag, the oxygen ratios of the various slags were computed and tabulated. In doing this alumina was considered as a base. Of course the results, considered on this basis, do not show conclusively that a slag of a certain oxygen ratio gives the best results, because the high percentage of alumina in the slag varied considerably, owing to melting of the brick in the furnace lining. That is, the effect of alumina, when substituted for another base, on the fluidity of the slag is not shown completely.

The oxygen ratios of the different slags and the copper losses in the different tests were as follows:

Losses of copper with different slags.

Experiment No.	Ore smelted.	Flux used.	Oxygen ratio of slag. $\left(\frac{\text{Acid O}}{\text{Basic O}}\right)$	Proportion of copper in slag.	Proportion of copper lost.[a]		
					Losses in slag.	Losses by volatilization, in hanging pieces, and in dust.	Total.
				Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
4	Concentrate No. 4.	Silica	1.97	0.23	0.88	17.65	18.53
1	do	None	1.85	.40	.97	....	....
5	do	Silica	1.60	.15	.48	56.40	56.88
7	do	do	1.59	.10	.40	48.30	48.70
6	do	do	1.28	.15	.42	49.20	49.62
2	do	None	1.10	.23	.87	36.90	37.77
9	do	Hematite	.87	.20	.59	35.25	35.84
8	Concentrate No. 3.	do	1.44	.15	.26	7.22	7.48
12	do	Limestone and hematite	.93	.24	.22	12.03	12.25
10	do	Limestone	.91	.17	.33	3.17	3.50
13	do	Limestone and hematite	.82	.46	.85	16.41	17.26
11	do	do	.70	.54	.54	11.62	12.16

[a] Computed on basis of total copper in ore.

EXPERIMENTS WITH CONCENTRATE NO. 4.

From the table it is evident that as the acidity of the slag is decreased for concentrate No. 4 there is a gradual decrease in the percentage of copper in the slag until a slag having an oxygen ratio of 1.59 is reached, when the loss rises slightly and remains uniform with further increase of basicity. Concentrates No. 3 and No. 4 are considered separately because of the higher alumina and lower copper content in No. 4. The oxygen ratio does not vary directly with composition of the charge, because some of the slag from previous runs and some of the lining entered the slag. The ratio 1.59, at which the percentage of copper in the slag reaches the lowest point (0.10 per cent), corresponds to that of a slag containing 48 per cent SiO_2, 9 per cent CaO, 1.84 per cent MgO, 26.25 per cent FeO, and 14.99 per cent Al_2O_3. With this ratio the lowest percentage of losses in the slag from No. 4 concentrate, on the basis of total copper in the ore charged, occurred. The percentage of copper unaccounted for was high, however, though in all experiments, except where noted, the furnace was tapped practically clean, except for what slag stuck to the wall in spots. This unaccounted for loss may be considered as largely due to volatilization, dust loss, and hanging of the charge. To be on the safe side, all unaccounted for loss will be considered as volatilization and dust loss, although in a small furnace, without tearing out the lining, 10 per cent of hanging charge would not be excessive. The total loss of copper in experiment No. 7, including copper unaccounted for—that is, copper lost by volatilization, as dust, and by hanging of the charge, and the copper loss in the slag—was 48.7 per cent.

A further study of the results of lowering the acidity of the slag shows that although there is an increase in the percentage of copper in the slag it is not excessive, and that there is a considerable decrease in the percentage of copper lost in dust or by volatilization. With an oxygen ratio of 0.87, the percentage of copper in the slag is 0.20 per cent, the percentage of total copper charged that was lost in slag is 0.59 per cent, the copper unaccounted for is 35.25 per cent, and the total loss of copper 35.84 per cent. The slag resulting from experiment No. 4 shows the effect of a low percentage of alumina in a marked manner. This slag has an oxygen ratio of 1.97, greater than any of the others, but the loss by volatilization and in dust is less than in any other experiment with No. 4 concentrate, being 17.65. The slag contained 0.23 per cent copper, the total loss being 18.53 per cent of the copper contained in the quantity of ore charged. This slag contained 51.1 per cent SiO_2, 8.78 per cent CaO, 2.03 per cent MgO, 25.5 per cent FeO, and only 10.45 per cent of Al_2O_3. The cause of the decrease in the alumina content was not apparent. This evidently lowered the melting point considerably, so that less copper was volatilized.

The slag loss of copper was low in all of the experiments with No. 4 concentrate and the loss by volatilization and in dust was high. These two losses, the writers believe, can be eliminated. The material used was much finer than No. 3 concentrate, as shown by the analysis on p. 21. This fineness of subdivision accounts in a measure for the oxidization of copper dust, which was plainly visible during the experiments, as shown by the marked green flame. Also, more dust was carried off by the gases from the oxidation of the electrode than in the later experiments with concentrate No. 3. The results of the experiments with concentrate No. 3, in which a more basic slag was used, show that the volatilization loss can be kept low by using a slag with a low melting point. As the oxygen ratio was decreased in the tests with concentrate No. 4 the decrease of this loss was marked. The dust loss would be small in a large furnace. It probably would be necessary to use a slag higher in ferrous oxide and having a lower melting point for concentrate No. 4 than for No. 3, in order to keep down the oxidation and volatilization of fine copper.

EXPERIMENTS WITH CONCENTRATE NO. 3.

The experiments with concentrate No. 3 showed a much less total loss than did those with No. 4. This was undoubtedly due to the fact that the slags used for No. 3 had a lower oxygen ratio, resulting in a much more fluid slag with a lower melting point. Also bases were added to dilute the alumina, so that this constituent had less effect than in the previous experiments. The best results were obtained in experiment No. 10, in which the oxygen ratio of the slag was 0.91, the percentage of copper in the slag being 0.17. The total loss of copper in the slag was 0.33 per cent of the amount of copper in the ore charged. The loss by volatilization, in dust, and in hangings was 3.17 per cent, making a total loss of 3.5 per cent. These results were obtained by increasing the lime content by charging limestone, giving a slag containing 35.82 per cent SiO_2, 15.45 per cent CaO, 7.09 per cent MgO, 25.20 per cent FeO, and 17.10 per cent Al_2O_3. That this slag gives good results was confirmed in the continuous experiment No. 14, in which the same proportions of charge were used as was used in No. 10.

As the oxygen ratio became lower than about 0.91 there was a tendency for increased losses in the slag and a higher loss from other causes. It seems from the results that the most satisfactory slag for these particular ores is one with an oxygen ratio of about 1.0, that is, a monosilicate, and containing 35 per cent SiO_2, 22 per cent CaO and MgO, 25 per cent FeO, and 17 per cent Al_2O_3. The oxygen ratio of this slag corresponds to that of an ordinary copper blast furnace slag, but owing to the nature of the ore the alumina content is very high. Also the use of lime as a base instead of iron oxide seems to give better results with ore of high alumina and iron oxide content as regards copper losses, due probably to the formation of a slag of lower melting point. Increasing the lime content also lowers the specific gravity of the slag, thus permitting easier settling of the metal.

There is no reason why the results obtained with the coarser material No. 3 should not apply to the finer concentrate No. 4. The high losses in experiments with the latter were due largely to the high melting point of the slag. As no more ore was at hand, the experiments could not be continued with this concentrate, and No. 3 was used. The percentage of copper in the slag need not exceed 0.25 per cent with a slag of proper composition. The percentage loss in the slag of total copper in the ore charged should not exceed 0. 5 per cent. The other losses in the best run (No. 10) amounted to 3.17 per cent, but in practice with continuous operation of a larger furnace this item should not exceed 1 per cent, and the total loss should not be over 1.5 per cent of the total amount of copper in the ore charged. The use of limestone is preferable in the case of the material treated, simply because of the production of a slag of easier fusibility and lower specific gravity, which allows better mechanical separation of metallic copper and slag and reduces the loss by volatilization. The dust loss would also be less if the very fine concentrate was mixed with a coarse concentrate.

IRON IN THE METALLIC COPPER.

In some of the runs in which a slag of high oxygen ratio and high melting point was used the percentage of iron in the copper produced was very high. The effect of length of run and of the ratio of acid to basic constituents in the slag is shown in the following table:

Metallic iron in the metal produced, experiments Nos. 1 to 13.

Experiment No.	Ore smelted.	Flux used.	Length of experiment.	Oxygen ratio. $\left(\frac{\text{Acid O}}{\text{Basic O}}\right)$	Proportion of iron in metal.	Weight of iron in metal.	Weight of metallic iron in charge.	Weight of iron reduced.	Percentage of electrode consumption used in reduction.
			Minutes.		*Per ct.*	*Pounds.*	*Pounds.*	*Pounds.*	
4	Concentrate No. 4.	Silica	24	1.97	1.75	0.079	0.110	0.0	0.0
1	do	None	50	1.85	12.50	.825	.110	.715	29.10
3	do	Silica	30	1.68	3.50	.191	.110	.081	7.41
5	do	do	25	1.60	15.90	.454	.110	.344	16.20
7	do	do	22	1.59	3.75	.111	.110	.001	.06
6	do	do	23	1.28	9.25	.283	.110	.173	10.35
2	do	None	30	1.10	11.10	.441	.110	.331	11.60
9	do	Hematite	23	.87	1.90	.069	.110	.0	.0
8	Concentrate No. 3.	do	23	1.44	1.10	.083	.066	.017	.92
12	do	Limestone and hematite.	30	.93	6.30	.429	.059	.370	16.40
10	do	Limestone	20	.91	2.50	.203	.066	.137	6.95
13	do	Limestone and hematite.	26	.82	3.10	.217	.066	.151	6.50
11	do	do	25	.70	6.20	.423	.059	.364	16.10

There was a small amount of free iron present in both concentrates No. 3 and No. 4, which entered the metal to an appreciable extent. This free iron was from the wear of breakers and stamps during milling of the ore. The influence exerted by the carbon electrode as a reducing agent is also shown in the table. Aside from iron little attention was paid in this research to other impurities in the copper, as it would be necessary to refine the copper in any case. The sulphur and arsenic ran about the same in all the samples. However, it was desirable to ascertain whether the ore could be melted and the gangue separated from the copper without reduction of iron by the carbon of the electrode to an extent that would contaminate the product and cause excessive expense in subsequent refining in a reverberatory furnace. In the reverberatory furnace the iron is removed from copper by oxidation and slagging, so that an excessive amount of iron, say over 5 per cent, would increase the cost of the process considerably.

FACTORS AFFECTING REDUCTION OF IRON BY THE ELECTRODE.

The chief factors which might have influenced the percentage of iron reduced by the carbon electrode are the length of time the ore was exposed to the electrode and the melting point of the slag formed. The time of exposure to the reducing action of the electrode did not appear to have a marked effect on the percentage of iron reduced, for in some tests a product high in iron was obtained after a run of the same length of time as an experiment in which the product contained little iron. For example, the copper from run No. 6, which lasted 23 minutes, contained 9.25 per cent of iron, whereas that from No. 9, which also lasted 23 minutes, contained only 1.90 per cent of iron.

A classification of the products of the experiments with both concentrates according to the oxygen ratios of the slags does not show a uniform change in the percentage of iron, but in general the percentage of iron reduced by the electrode decreases as the slag becomes more basic and approaches a monosilicate. This result is probably due entirely to the higher melting point of the more acid slag. The furnace, when operating as a resistance furnace, adjusts itself somewhat to the melting temperature of the charge so that with a refractory slag it gets hotter. This increase in temperature results in accelerating the reaction between the carbon electrode and the iron oxide of the gangue. With a monosilicate slag containing not over 15 per cent of alumina this reduction of iron need not result in a product containing over 2 per cent iron, including any metallic iron in the charge. This was shown by the results of the intermittent runs (see p. 28) and was further confirmed by experiment No. 14, in which the metals from three tappings during continuous operation of the furnace contained, respectively, 0.30 per cent, 1.04 per cent, and 2 per cent of iron.

The proportion of the total electrode consumption used in the reduction of iron varied from zero to 29.1 per cent, the average for 13 experiments being 9.35 per cent. From these results it may be seen that, with a furnace operating at a low temperature, the reduction of iron by the carbon electrode should not be excessive, or enough to result in a product containing less than 95 per cent of metallic copper. The writers believe that in practice the contents of metallic iron and other impurities could be kept as low as 2 per cent, giving a product containing 98 per cent copper.

EFFECT OF BASE USED.

The kind of base used to form a monosilicate slag did not seem to affect materially the reduction of iron oxide. The experiments indicate that there was slightly less reduction with hematite than when limestone was added, but there is so little difference that probably as good results could be obtained with one flux as with the other, the problem being simply one of economics as to which is the cheaper. The essential point is to have a fluid slag of low melting point, which can be obtained with either flux. It was thought in the beginning of the experiments that by the use of a slag high in silica it might be possible to eliminate the reducing effect of the electrode. This result was not accomplished in most of the tests because the slag formed was difficultly fusible, owing to the high alumina content of the ore. The high temperature increased the reducing effect so that any strong tendency of the FeO to combine with the SiO_2 was more than offset.

EFFECT OF ALUMINA.

Here, as well as in the discussion of copper losses, the effect of the alumina, as regards the reduction of iron oxide, should not be forgotten. In considering a series of very acid slags for copper losses and iron oxide reduction, it may be noticed that any diversion from high loss by volatilization was generally due to a decrease in the percentage of alumina in the slags. The adaptability of the electric furnace to smelting slags of high alumina content is clearly demonstrated by these experiments. As before stated, when the electric furnace is operated with the charge as the resistance and no arcs are formed, it adapts itself to the temperature required to melt the charge and produce a fluid slag. Of course there are many slags which could not be melted without undue power expenditure.

CONCLUSIONS.

These experiments show that the quantity of metallic iron reduced by the action of the carbon electrode is so small as to be of no practical importance as a drawback to the substitution of the electric furnace for the reverberatory furnace in the smelting of these concentrates. The above statement also applies to sulphide copper ore. The small amount of iron which is reduced alloys with the copper, or, if the ore is a sulphide, enters the matte and increases its collecting power for gold, especially if the matte consists mainly of FeS. In either case not enough iron could be reduced to form a product so impure that excessive refining or reconcentration would be necessary.

POWER CONSUMPTION.

With the exception of the continuous run, experiment No. 14, the lowest power consumption and in general the best results of all the experiments were obtained in experiment No. 10, in which concentrate No. 3 was fluxed with limestone. For comparing the theoretical power consumption with the actual consumption the charge used in this run is taken as a basis.

For convenience, the results of this run will be reduced to the basis of 1,000 kilograms of ore; that is, 1 metric ton (2,204 pounds). The quantity of limestone required to flux 1,000 kilograms of ore is 220 kilograms. The metal is assumed to contain 2.50 per cent iron, which includes that reduced from the FeO of the ore by the carbon electrode, and any metallic iron present. The gases, CO from the electrode and CO_2 from the limestone, are assumed to escape at an average temperature of 500° C. at the top of the furnace during the melting stage. This temperature is high, but there was no shaft on the furnace. The ore is assumed to be dry and of the composition given on page 22. The loss of copper in the slag is assumed to be 0.33 per cent of the total amount of copper in the ore charged.

The results, computed on this basis, are as follows:

Results of smelting concentrate No. 3 with limestone flux on basis of 1,000 kilograms of ore, experiment No. 10.

	Quantity charged or consumed.	Quantity in product formed.		
		Metal.	Slag.	Gases.
Ore No. 3:	*Kilos.*	*Kilos.*	*Kilos.*	*Kilos.*
Cu	373.5	372.27	1.23	
SiO_2	333.3		333.30	
Fe	122.7	9.53	113.17	
Al_2O_3	87.8		87.8	
CaO	51.8		51.8	
MgO	10.9		10.9	
O	34.3		32.5	1.8
	1,000.0			
Limestone flux:				
CaO	121.0		121.0	
MgO	19.09		19.09	
CO_2	79.2			79.2
Carbon of electrode	14.1			14.1
Oxygen from air, completely burned to CO, with oxygen from iron reduction	17.0			17.0
Total	1,264.69	381.80	770.79	112.1

Results of smelting concentrate No. 3 with limestone flux on basis of 1,000 kilograms of ore, experiment No. 10—Continued.

COMPOSITION OF SLAG.

Constituent.	Calculated.	Analyzed.
	Per cent.	*Per cent.*
Cu	0.16	0.17
SiO_2	43.20	35.82
CaO	22.40	15.45
MgO	3.88	7.09
FeO	18.81	25.20
Al_2O_3	11.37	17.10

The wide difference between the actual analysis and the calculated composition of the slag may be accounted for by several factors which must be taken into consideration. Possibly some of the fire-brick lining was fused and entered the charge, thus increasing the alumina content; or some of the previous charge, which contained hematite, may have been left in the furnace; also during the run considerable limestone was carried out of the furnace by the gases.

In the calculation of the theoretical power required it seems best to consider the melting point of the slag as that of the slag actually obtained. According to Hofman [a] the melting point of a slag containing 32.1 per cent SiO_2, 17.95 per cent FeO, 16 per cent CaO, and 11.37 per cent Al_2O_3 is 1,310° C. and that of a slag containing 32.1 per cent SiO_2, 24.69 per cent FeO, 22 per cent CaO, and 11.37 per cent Al_2O_3 is 1,145° C. The melting point of the slag actually obtained in run No. 10 is assumed to be 1,250° C. and the temperature necessary to make it flow to be 1,350° C. The quantity of heat necessary to melt and superheat the slag is assumed to be 600 calories per kilogram. In oxidizing C to CO, 8,100 calories are developed per kilogram of carbon. The amount of heat necessary to reduce 1 kilogram of Fe from FeO is assumed to be 1,173 calories. The quantity of heat necessary to expel 1 kilogram of CO_2 from $CaCO_3$ is 191 calories. The specific heat of dry carbon monoxide at 0° to 500° C. is 0.256 calorie per kilogram and that of carbon dioxide is 0.245 calorie per kilogram. It is assumed that 204 calories of heat are required to melt and superheat 1 kilogram of pure copper, and that 370 calories are required for 1 kilogram of iron. The metal contained some iron, but owing to insufficient data on the specific heats of alloys of iron and copper, the specific heat of pure copper is used. All of the heat constants are taken from Richards's "Metallurgical Calculations."

The theoretical amount of heat required to smelt 1,000 kg. of ore may now be computed, as follows:

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 395.

Theoretical amount of heat required to smelt 1,000 kg. of ore.

		Calories.
Melting and superheating 372.27 kg. of copper	372.27×204=	75,943
Melting and superheating 9.53 kg. of iron	9.53×370=	3,386
Reducing 9.53 kg. of Fe from FeO	9.53×1,170=	11,178
Melting and superheating 770.79 kg. of slag	770.79×600=	462,474
Raising the temperature of 32.8 kg. of CO to 500° C	500×0.256×32.8=	4,195
Raising the temperature of 79.2 kg. of CO_2 to 500° C	500×0.245×79.2=	9,650
Total		566,826
Amount of heat supplied by combustion of 14.1 kg. of carbon	14.1×8,100=	114,210
Amount of heat supplied by electric current		452,616
Total		566,826

The theoretical amount of electrical energy required for smelting No. 3 concentrate is $\frac{452,616}{857}=529$ kilowatt-hours per metric ton of concentrate charged, or 481 kilowatt-hours per short ton, or 0.055 kilowatt-year per ton. On the basis of total charge the energy consumption is 393 kilowatt-hours per short ton of charge or 0.045 kilowatt-year per ton. The theoretical energy consumption per pound of copper produced is 0.63 kilowatt-hour.

In experiment No. 10 the actual power consumption was 1,310 kilowatt-hours per ton of ore smelted, 1,280 kilowatt-hours per ton of charge smelted, and 1.77 kilowatt-hours per pound of metal tapped. As 1,310 kilowatt-hours per ton of ore smelted were actually used, whereas 481 kilowatt-hours per short ton are theoretically required, without allowing for heat losses or other inefficiency, the furnace used 2.72 times more power than was theoretically necessary, or it had a heat efficiency of 36.76 per cent.

The power consumption in experiments 1 to 13, tabulated according to the oxygen ratios of the slags, is as follows:

Power consumption in smelting tests of native copper concentrates.

Experiment No.	Ore used.	Flux.	Oxygen ratio. $\left(\frac{\text{Acid O}}{\text{Basic O}}\right)$	Power consumption— Per pound of metal tapped.	Per ton of ore smelted.	Per ton of charge smelted.
				Kw.-hours.	*Kw.-hours.*	*Kw.-hours.*
4	Concentrate No. 4	Silica	1.97	3.53	1,450	1,320
1	do	None	1.85	4.05	2,425	2,425
3	do	Silica	1.68	3.67	1,820	1,740
5	do	do	1.60	6.30	1,600	1,330
7	do	do	1.59	4.19	1,130	1,015
6	do	do	1.28	4.75	1,320	1,075
2	do	None	1.10	4.13	1,500	1,500
9	do	Hematite	.87	4.73	1,565	1,425
8	Concentrate No. 3	do	1.44	2.13	1,490	1,420
12	do	Limestone and hematite	.93	2.06	2,070	1,455
10	do	Limestone	.91	1.77	1,310	1,280
13	do	Limestone and hematite	.82	2.77	1,780	1,100
11	do	do	.70	2.68	1,880	1,565

As some of the charges were treated longer than was absolutely necessary, the power consumption varied and the results are not uniform. The power consumption per pound of metal tapped in some of the runs with No. 4 concentrate was high, owing to the low amount of metal tapped, but of course was less when concentrate No. 3, which contained a higher percentage of copper, was charged. A low power consumption per ton of charge was obtained with No. 4 ore in experiments 6 and 7, in which slags with oxygen ratios of 1.28 and 1.59, respectively, were produced, but the volatilization loss was so high as to prohibit the use of such charges. The lowest power consumption obtained in any of the experiments in which copper losses were not excessive was in runs 10 and 13, being 1,280 and 1,100 kilowatt-hours per ton of charge. Limestone flux was used for No. 10 and a mixture of limestone and hematite for No. 13, the oxygen ratios of the slags being 0.91 and 0.82, respectively.

In general it may be stated that the most satisfactory results, as regards both power consumption and copper losses, are obtained when the native copper concentrate is fluxed with limestone or hematite, or both, in such proportions that the slag formed approximates a monosilicate. With the small furnace used, which required 40 kilowatts—the average in 14 experiments—a heat efficiency of 36.76 per cent was obtained. A 750-kilowatt commercial furnace of modern design, equipped with feeding shafts and a covered top, when working on similar ore should have a heat efficiency of 70 per cent and possibly higher, as the process consists of simply melting the ore, but 70 per cent may be taken as a safe figure. In experiment No. 14, in which the furnace was run continuously on the same kind of charge as was used in No. 10, the power consumption was even lower than in No. 10, because of the continuous charging, being 1.76 kilowatt-hours per pound of metal, 1,190 kilowatt-hours per ton of ore, and 1,075 kilowatt-hours per ton of charge.

On the basis of 70 per cent efficiency a large 750-kilowatt furnace would consume only $\frac{36.76}{70}\times 100=52.5$ per cent as much energy as the small 40-kilowatt furnace in smelting the same amount of ore. Hence it may be stated that the following power-consumption figures should be obtained in smelting fine Michigan native copper concentrates, containing from 25 to 40 per cent copper, in a 750-kilowatt commercial electric furnace: Kilowatt-hours per pound of metal tapped, $0.525\times 1.76=0.92$; kilowatt-hour per ton of ore smelted, $0.525\times 1{,}190=625$; kilowatt-hours per ton of charge smelted, $0.525\times 1{,}075=564$.

OPERATION OF FURNACE.

No difficulty was experienced in operating the furnace on the resistance principle. The fineness of the material assisted in this considerably, as there was not so great a tendency for arcking with the particles of the charge packed closely. After the first few experiments it was found that the furnace worked best at 70 volts, and the voltage was kept at about 70 during later runs. The average power on the furnace was 40 kilowatts at a voltage of 73 volts and an amperage of 621 amperes. The power factor was high, averaging 0.95. As the furnace was over powered little difficulty was experienced in starting it. The fire-brick walls were not corroded appreciably by the charge, but when superheating was prolonged after the charge had been melted the fire brick melted in some places and mixed with the slag. This accounts for the variable nature of the slag as compared with that of the materials charged. The carbon bottom used in the first three experiments was not properly put in and was replaced by another rammed in harder, which lasted throughout the experiments, and was apparently not at all affected. Ground coke mixed with tar to a paste gives as good satisfaction as ground graphite electrode butts.

ELECTRODE CONSUMPTION.

The average electrode consumption throughout all of the experiments was 34.5 pounds of graphitized carbon per short ton of charge. The highest consumption, 48 pounds, was in experiment No. 1 and the lowest, 22.1 pounds, was in experiment No. 14. This last figure more nearly represents practical working conditions and could undoubtedly be attained, and even bettered, in practice. The atmosphere of the electric furnace is practically neutral during the melting of a material in which all of the iron oxide (FeO) is combined to form a silicate. For this reason the consumption of the electrode, although it would not be as low as in a reducing atmosphere, should not be excessive, because of the combined state of the ferrous oxide. In practice the electrode consumption should not exceed 10 pounds per ton of charge and probably would be much less.

From these results it is evident that the loss of electrodes through contact with the charge would not form a serious objection to the substitution of the electric furnace for the reverberatory furnace in smelting copper ores. If all of the electrode consumption were used in the reduction of oxides, the amount of energy consumed in supplying heat for reactions might be appreciable, but as an average of only 9.35 per cent of the total electrode loss was so used any loss of electrical energy in this way may be considered as negligible.

UTILIZATION OF SLAGS FOR PRODUCTION OF FERROSILICON.

In connection with the electric smelting of both oxide and sulphide ores of copper, the idea has always been prevalent that it is possible to utilize the slag for making ferrosilicon in the electric furnace, but not much actual experimental work has been done to determine the feasibility of the process.

To obtain information on the factors involved a series of experiments was undertaken.

Five experiments were made, using two lots of slag produced in smelting the native copper concentrates. The composition of these slags, as shown by analysis, was as follows:

Analyses of slags used.

Constituent.	No. 1.	No. 2.
	Per cent.	*Per cent.*
CuO	0.22	0.80
SiO_2	46.19	34.63
CaO	9.61	16.54
MgO	2.55	3.57
FeO	24.10	28.24
Al_2O_3	16.58	14.75

The coke that was used contained 84.38 per cent of fixed carbon, 0.34 per cent of moisture, 0.61 per cent of volatile and combustible matter, and 14.67 per cent of ash. The iron ore and silica were of the composition given on pages 21 and 22. The results of the tests are negative, but are given to show that the making of ferrosilicon from copper slags, which has been assumed by some writers to be possible without experimental confirmation, is a difficult matter.

EXPERIMENT WITH SLAG AND COKE.

In the first experiment, No. 15, 61.6 pounds of slag No. 1 and 17.6 pounds of coke were charged. There was some copper frozen in the furnace when it was charged, which partly accounts for the high percentage of copper in the product. An attempt was made to operate the furnace continuously, but both the metal and the slag were too thick to flow from the furnace, although several attempts were made to tap it during the run. At the end of the run 8.3 pounds of metal were tapped, but the slag would not flow. The slag was removed as far as possible by balling with iron rods, and what could not be removed in this way was chiseled out when cold. The total amount of slag obtained was 59.3 pounds. The metal was very brittle, looked like white iron, and had the appearance of ferrosilicon containing 25 per cent of silicon. Dense white fumes were given off during the run. The power consumption was 3.82 kilowatt-years per ton of metal tapped, or about 10 times the power required to produce 1 ton of "25 per cent" ferrosilicon from iron turnings, quartz, and coal in a commercial ferrosilicon furnace. Coke was charged in amount sufficient to reduce all iron and silicon charged. Possibly, however, some silicon carbide was formed, as this compound tends to form if an excess of coke be present. The effect of silicon carbide is to make the slag thick. The amount of copper in the metal was considerably greater than the amount charged, for the reason previously stated.

The product contained 21.23 per cent silicon, 58.01 per cent iron, 19.35 per cent copper, 0.02 per cent phosphorus, and a trace of sulphur. The slag contained 46.25 per cent SiO_2, 22.4 per cent Al_2O_3, 14.6 per cent FeO, 9.81 per cent CaO, and 2.70 per cent MgO.

EXPERIMENT WITH SLAG, QUARTZ, AND COKE.

In experiment No. 16 the charge consisted of 44 pounds of slag No. 2, 14.65 pounds of quartz, and 12.24 pounds of coke. The charge was calculated to give a product containing 50 per cent of silicon and a slag containing 40 per cent of SiO_2, on the assumption that all of the coke was used for the reduction. The results were similar to those obtained in the previous run; the slag fused, but would not flow from the furnace, the yield of metal was low, and the power consumption was high. The ferrosilicon contained 18.38 per cent silicon, 73.02 per cent iron, 7.01 per cent copper, 0.08 per cent sulphur, and 0.04 per cent phosphorus. The slag contained 49.63 per cent SiO_2, 27.24 per cent Al_2O_3, 9.76 per cent FeO, 9.85 per cent CaO, and 2.60 per cent MgO.

OTHER EXPERIMENTS.

Three other experiments were conducted in which even more negative results were obtained. In one, iron ore was added to ascertain whether the presence of iron in an easily reducible form would accelerate the silicate reduction, just as it lowers the reduction temperature of silica not combined as silicate, but no beneficial effect was obtained. In these runs part of the slag would flow from the furnace, but the product was a high-silicon pig iron. Fine grinding was tried, but it caused the material to pack so that the furnace could not be made to arc steadily, and the maintenance of a steady arc is necessary to attain a high enough temperature for the production of ferrosilicon.

DISCUSSION OF RESULTS.

In these particular experiments the production of ferrosilicon from slag in the electric furnace must be accounted a failure because of the low yield of ferrosilicon, due to the difficulty of reducing the silicate and to the tendency of the constituents of the slag other than iron and silica to rob the ferrosilicon of those elements. As a result of this, the power consumption and the difficulty of operating the furnace were too great for practical purposes. The remarks on the results of these experiments apply in general to the production of ferrosilicon from slags. The writers believe that it would be impractical to attempt the manufacture of ferrosilicon from a slag containing more than 10 per cent of oxide other than FeO or SiO_2 because of the clogging up of the furnace with the difficultly fusible slag resulting. With even 10 per cent of oxide present to form a slag with the FeO and SiO_2, it would probably not be possible to make a ferrosilicon containing over 30 per cent silicon. In the commercial production of ferrosilicon from iron turnings and silica in the electric furnace, "30 per cent" ferrosilicon can be made if some slag is present; a "50 per cent" product can be made only with difficulty, and higher grades not at all, if any slag is formed.

Even if the proper slag is available, the process, except under unusual circumstances, does not seem commercially attractive. A copper smelter is usually situated in a district remote from steel centers, where the greatest market for ferrosilicon is found. The slag, in order to compete with iron turnings, or iron ore, and quartz or sand as raw material for making ferrosilicon, must be cheap enough to offset the disadvantages of being possibly farther from the market, and requiring a higher power consumption.

CONCLUSIONS.

The conclusions that were drawn from the experiments described in the preceding pages are as follows:

1. Fine Michigan native copper concentrates can be smelted in the electric furnace with the production of a good grade of black copper and without excessive losses of copper in the slag or by other means.

2. The percentage of copper in slag of the proper composition need not be more than 0.25 per cent. The proportion of total copper charged that is lost in slag should not exceed 0.5 per cent, and all other losses should not exceed 1 per cent, or the total loss should be not more than 1.5 per cent.

3. The loss of copper by volatilization is high if the slag is much more acid than a monosilicate or if it contains much alumina. The lowest losses were obtained when limestone was added to form a slag containing 35 per cent of SiO_2, 22 per cent of CaO and MgO, and 25 per cent of FeO.

4. The difference between limestone and hematite as a flux is not marked, one being about as efficient for this purpose as the other if the slag is not of too high a specific gravity. No more iron was found in the black copper product when hematite was used than when limestone was used.

5. With a furnace operating at a low temperature on a monosilicate slag, the reduction of iron by the carbon electrode should not be excessive or sufficient to result in a product containing less than 95 per cent of metallic copper. With care a product containing 98 per cent of copper could probably be produced regularly.

6. The total electrode consumption greatly exceeded the amount of carbon necessary for reduction of the iron found in the metal, as an average of only 9.35 per cent of the total amount of electrode consumed was used in iron reduction. In practice the electrode consumption should not exceed 10 pounds per ton of charge.

7. The electric furnace operates satisfactorily with the highly siliceous-aluminous slag produced by the natural gangue without flux, but the losses by volatilization are high because of the high melting point of the slag. There was no copper oxide in the product, as was shown by microscopic examinations.

8. The power consumption for smelting fine Michigan native copper concentrates, carrying 25 to 40 per cent of copper, in an electric furnace of 750 kilowatts or greater capacity should not exceed 625 kilowatt-hours per ton of ore smelted.

9. The production of ferrosilicon from the copper slags can not be economically accomplished if the slag contains large proportions of components other than ferrous oxide and silica.

PROPOSED PROCESS FOR ELECTRIC SMELTING OF FINE NATIVE COPPER CONCENTRATE.

The commercial smelting of fine native copper concentrate in the electric furnace, in order to attain the greatest economy, would be a continuous operation. The process would consist of charging the mixture of ore and flux at intervals into the top of an electric furnace having a short stack above the smelting crucible for preheating the charge before it reached the smelting zone, and tapping the metal at intervals, the slag being permitted to run continuously from the furnace through a slag notch above the metal hole.

The possibility of operating a furnace in this manner was shown in experiment No. 14. There was an average of 1.10 per cent of iron in the product, and 1.37 per cent of copper in the slag. The total loss of copper was as low as in the intermittent runs and was largely due to the small scale of the operation. The loss of copper in the slag was high because of imperfect settling and because of the tapping of partly fluid material with the slags, a condition that would not occur in a large furnace properly operated. The power consumption was lower than in the other experiments.

DESCRIPTION OF FURNACE.

The furnace features essential to the success of this process are a crucible, in which the electrical energy is introduced, and in which the metal settles from the slag, and a short shaft above the crucible for preheating the charge before it reaches the smelting zone. As there is no reduction to be performed in the shaft it need not be higher than to give plenty of space between the top of the crucible and the charging floor for head room in moving the electrodes.

A furnace of the type shown in figures 3 and 4 satisfies these conditions. The furnace has a smelting chamber, *a*, lined with fire brick or silica brick; three carbon electrodes, *b*, suspended vertically through the roof; and a feeding shaft, *c*, which is lined with fire brick, over the crucible.

A 750-kilowatt furnace of this type would smelt in 24 hours about 23 tons of native copper concentrates, carrying 25 to 40 per cent copper. This furnace would be operated with three-phase current at from 50 to 100 volts. The internal dimensions of a 750-kilowatt furnace of this type are as follows: The crucible is 14 feet 6 inches in diameter and 9 feet high, the shaft is 1 foot 6 inches in diameter and 18 feet 6 inches high.

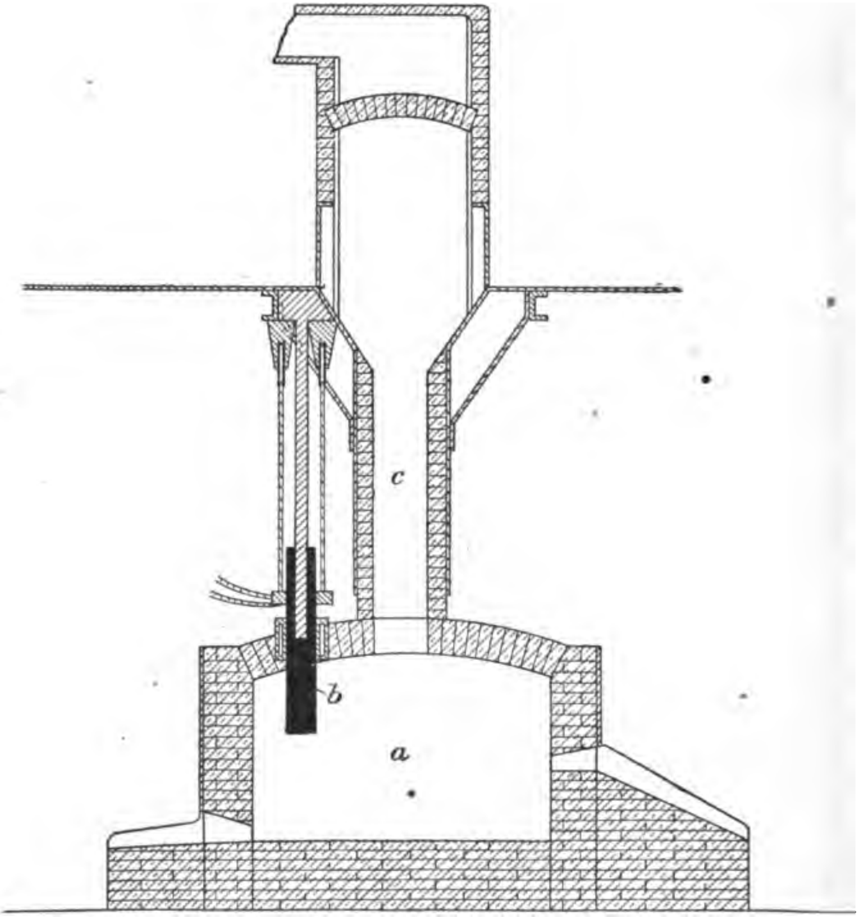

FIGURE 3.—Elevation of electric furnace for smelting native copper concentrates.

As regulation of electrodes would not be necessary in this type of furnace when in regular operation the electrical connections are made close to the roof of the furnace, and regulation is by hand. There is less difficulty in handling the electrodes if they are suspended vertically, and the vertical shaft eliminates the need of suspending the electrodes at an angle of about 55 degrees.

The slag from such a furnace should not contain more than 0.25 per cent copper, and could be discarded at once without further treatment.

It could be granulated with water as discharged from the furnace. The metal would require further refining for elimination of iron and other impurities, and could be charged directly from the electric furnace into a reverberatory furnace and refined as previously described.

COMPARISON OF PROPOSED AND PRESENT METHODS.

The present method of treating these concentrates consists of the following steps:

1. The coarse concentrate is melted and refined in either one or two reverberatory furnaces, from which a slag is produced containing from 10 to 30 per cent copper.
2. The fine concentrate is agglomerated in a reverberatory furnace, or briquetted with lime.
3. The slags from the reverberatory furnace and the agglomerated, or briquetted, fine material is smelted with coke in a blast furnace. The slags from the blast furnace contain 0.6 to 0.8 per cent copper and the metal 90 to 95 per cent copper.
4. The metal from the blast furnace is re-treated in a reverberatory furnace.

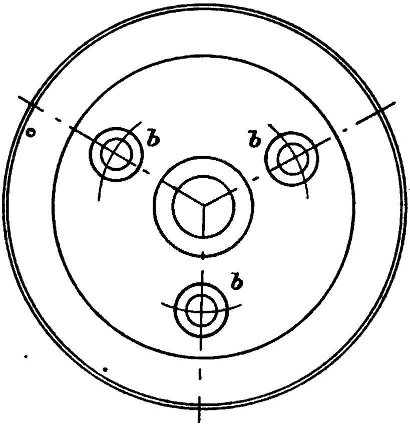

FIGURE 4.—Plan of electric furnace for smelting native copper concentrates.

There would be the following advantages in using the electric furnace for smelting these concentrates: (1) No agglomeration or briquetting would be necessary; (2) none of the slags produced would require re-treatment; (3) a large proportion of the higher grade concentrates, containing 75 per cent copper, could be mixed with the fine concentrates and treated directly in the electric furnace, thus reducing the amount of slag necessary for resmelting; and (4) if cheap power were available the electric furnace could be used for reducing the copper from the reverberatory slags. The resulting metal would probably be lower in iron, and the slag lower in copper because of the greater ease of regulating the reducing atmosphere in the electric furnace, as the amount of coke is sufficient only for the reduction of the combined copper.

The limit to the amount of higher grade concentrates that could be mixed with the fine material would be determined by the electrical conductivity of the mixture in the furnace. In the experiments performed, no short circuits or other difficulties were experienced in smelting a charge containing 37 per cent metallic copper. The point to which the proportion of copper in the charge could be increased would have to be ascertained by experiment.

Technically, the electric furnace does away with the necessity of briquetting or nodulizing the fine concentrates and in general facilitates handling of the materials. The commercial factors influencing the choice of the method to be adopted, as regards cost, would largely depend upon local cost of power and fuel. As regards present practice, the cost of fuel, the cost of briquetting or nodulizing, and possibly a higher labor cost are to be set against the cost of power for electric smelting. Even if the cost of power were equal to these three factors, the electric furnace might still have the advantage because of ease of operation.

In the case of a mine remote from smelting plants it might be cheaper to concentrate by smelting in a small electric furnace the low-grade product obtained by wet concentration and thus reduce the cost of freight. Also to save losses in wet concentration the ore might not be so highly concentrated by wet methods, a low-grade concentrate being made which could be smelted directly in the electric furnace, the high-grade black copper made being shipped to a custom smelter.

COST OF ELECTRIC SMELTING COMPARED WITH COST OF PRESENT METHODS.

Some reliable figures have been obtained by the bureau regarding the cost of smelting fine native copper concentrates in Michigan. These concentrates, containing 35 per cent copper, are agglomerated in a reverberatory furnace. One ton of concentrate yields 0.21 ton of metallic copper and 0.79 ton of slag containing 18 per cent copper. The slag is resmelted in a blast furnace and the copper is refined in a reverberatory and made into marketable shapes. On resmelting in the blast furnace, 0.79 ton of slag yields 0.152 ton of cupola block containing 88.9 per cent copper. The blast furnace slag averages 0.6 to 0.8 per cent copper. These blocks are refined in a reverberatory furnace, the yield from 0.152 ton being 0.129 ton of copper and 0.023 ton of slag. The copper is made into marketable forms and the slag is re-treated in the blast furnace.

Thus in treating one ton of concentrates a total of 0.813 ton of slag must be re-treated in the blast furnace.

The cost of smelting concentrates by the above method is given in the table following. The cost figures include overhead charges, all repairs, and maintenance of plant, but not amortization of investment.

Cost of smelting 1 ton of "35 per cent" native copper concentrate by reverberatory process.

	Per ton of concentrate.
Smelting 1 ton of concentrate in reverberatory furnace	$3.69
Making marketable shapes from 0.21 ton of copper, at $1.96 per ton	.41
Resmelting 0.79 ton of slag in blast furnace, at $4.71 per ton	3.72
Refining 0.152 ton of "cupola block" in reverberatory, at $2.77 per ton	.42
Making marketable shapes from 0.129 ton of copper, at $1.96 per ton	.25
Treatment of 0.023 ton of slag in blast furnace, about	.15
Total	8.64

For estimating the cost of treatment of these concentrates by the combined electric and reverberatory process, the electric furnace is assumed to be a 750-kilowatt furnace with a capacity of 28 tons per 24 hours. The cost of the plant, including cost of buildings, transformers, and furnace, is estimated at $20,000, which is a liberal estimate, as the furnace alone, including transformers, can be erected for $10,000. Reverberatory furnace smelting costs are taken as given in the preceding table. The furnace is assumed to be operating at a smelter where only a portion of the total superintendence, laboratory, and general costs would be charged against it.

The cost per ton of concentrate with power costs of 0.50, 0.625, 0.75, and 1 cent per kilowatt-hour is estimated to be as follows:

Estimated cost of smelting one ton of "35 per cent" native copper concentrate by the combined electric and reverberatory method.

Item.	Cost per ton of concentrate at a power cost per kilowatt-hour of—			
	0.5 cent.	0.625 cent.	0.75 cent.	1 cent.
Smelting in the electric furnace:				
Limestone, 0.10 ton at $1 per ton	$0.10	$0.10	$0.10	$0.10
Electrodes, 1 pound, at 4 cents per pound	.44	.44	.44	.44
Power, 640 kilowatt-hours	3.20	4.00	4.80	6.40
Labor, 3 men at $2 each and a boy at $1 per 8-hour shift	.75	.75	.75	.75
Superintendence, chemist, general expenses	.20	.20	.20	.20
Maintenance and repairs	.25	.25	.25	.25
Amortization and depreciation on $20,000 at 5 per cent each	.18	.18	.18	.18
Interest, on $20,000 at 6 per cent	.11	.11	.11	.11
Total	5.23	6.03	6.83	8.43
Smelting in the reverberatory furnace:				
Reverberatory refining of 0.352 ton of "cupola block"[a], at $2.77 per ton	.97	.97	.97	.97
Retreatment of 0.05 ton of slag from reverberatory	.25	.25	.25	.25
Cost of making marketable copper of 0.345 ton copper at $1.96 per ton	.67	.67	.67	.67
Total	1.89	1.89	1.89	1.89
Total cost of smelting ore per ton of concentrate	7.12	7.92	8.71	10.32

[a] Cupola blocks 98 per cent copper. Total loss of copper 1.5 per cent.

According to these figures the electric process with power at 0.5 cent per kilowatt-hour or $43.80 per kilowatt-year is $1.52 per ton cheaper, at 0.625 cent per kilowatt-hour is $0.72 per ton cheaper, and at 0.75 cent is $0.07 more expensive than the reverberatory process. The commercial operation of such a process does not require power of the great cheapness necessary in iron smelting to compete with existing processes. A rate of 0.5 cent per kilowatt-hour with a high load factor is possible in many districts. The load factor in this case would be above 90 per cent and the power factor not below 0.85. In practice the cost of refining the "cupola block" of the electric furnace would probably be considerably reduced over that of the blast furnace because of the higher percentage of copper in it.

Also by the electric process there would be only 0.023 tons of slag per ton of concentrate to be resmelted instead of 0.813 tons as with the present process.

SMELTING OF SULPHIDE COPPER ORES.

ORDINARY COPPER BLAST-FURNACE SMELTING.

Ordinary copper blast-furnace smelting—that is, smelting the ore with carbonaceous fuel—differs widely from pyritic smelting, which consists in smelting the ore with heat derived by the oxidation of the sulphur and iron contained in the ore itself.

As stated by Peters,[a] "The main characteristic of ordinary blast-furnace smelting is the employment of carbonaceous fuel mixed with ore as the principle source of heat." Such being the case, the problem that presents itself is to determine whether it is metallurgically feasible to substitute electricity for carbonaceous fuel as a source of heat and whether in so doing there will be obtained as favorable results chemically as it is possible to obtain in ordinary blast-furnace smelting.

PRINCIPAL FEATURES OF THE PROCESS.

The prerequisites and distinguishing features of ordinary copper blast-furnace smelting, as stated by Peters,[b] are as follows:

(*a*) The employment of moderately compressed air to effect combustion.

(*b*) The complete, or nearly complete, continuity of the smelting process, due to the fact that the ore and fuel are fed periodically on top of the charge column at the upper extremity of the furnace shaft and withdrawn continuously (or periodically) in a molten state from the lower portion of the furnace, so that the operation of blast-furnace smelting, as understood by copper metallurgists, is not only continuous but, theoretically speaking, perpetual.

(*c*) The actual contact of the ore and fuel, these substances being charged into the top of the furnace in alternate layers.

(*d*) The employment of carbonized or high-carbon fuel, such as coke, anthracite, or charcoal, to avoid the annoyance and danger arising from the presence in the ore column of the volatile hydrocarbons and various products of distillation which would result from the use of wood or the ordinary coals containing volatile matter.

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[a] Peters, E. D., Principles of copper smelting, 1907. [b] Peters, E. D., Op. cit., pp. 109, 167.

In ordinary blast-furnace smelting the atmosphere is distinctly reducing, owing to the presence of large quantities of C and CO, which have a greater affinity for O than have most of the constituents of the charge. Under normal conditions, however, this reducing action is not sufficiently powerful to reduce the oxides of iron to metallic iron, although it reduces Fe_2O_3 and Fe_3O_4 to FeO, providing there is SiO_2 present to combine with this base at the instant of its formation.

In the true pyrite blast furnace, however, where very little coke is used, the atmosphere may be regarded as neutral until the focus is reached. Here we first encounter the O of the blast, and the atmosphere is powerfully oxidizing, * * * but as the O has a greater affinity for the iron sulphide than it has for any other important substance that should ever reach the focus, and as there is—or should be—always more iron sulphide present than there is O to burn it, it follows that even the atmosphere of the focus of the pyrite furnace may be regarded as neutral toward a certain proportion of the iron sulphide, as well as toward various other substances that may come within its influence.

REMOVAL OF SULPHUR.

In the smelting of copper ores, sulphide ores in the main have to be dealt with and the problem consists in getting rid of the sulphides other than copper, and of concentrating the latter into a matte containing anywhere from 20 to 60 per cent copper. The first step, therefore, is to free the ore from its sulphur. Consider first how this is done in ordinary copper blast-furnace smelting. As pointed out by Peters,[a] owing to the constant presence of a large amount of glowing carbon in the furnace shaft, extending from the tuyères for several feet up the shaft of the furnace and often to the charging door itself, the oxygen of the air blown through the tuyères is consumed with extraordinary rapidity and thoroughness, and there is consequently but a poor chance for the oxidation of any substance in the ore that possesses less affinity for oxygen than carbon does.

Therefore, in ordinary blast-furnace smelting with carbonaceous fuel, the sulphides of iron and copper, and such small quantities of other metallic sulphides as may be present, stand little chance of obtaining any of the oxygen to combine with, to form sulphur dioxide and metal oxide. The atmosphere is almost always strongly reducing, and the sulphides tend to melt down with almost the same results that they would in a closed crucible, with complete exclusion of air.

It will thus be seen that, owing to the reducing atmosphere which prevails, we can, when smelting sulphide ores, look for the removal of but little sulphur in the blast furnace beyond the portion that will be driven off by heat alone without the presence of oxygen.

Of the two sulphides in the charge which have to be taken into consideration in this connection, namely, pyrite (FeS_2) and chalcopyrite ($Cu_2S.Fe_2S_3$), the former loses one-half of its sulphur content by mere heating without air, whereas the latter, when similarly heated, loses one-third of its sulphur content and one-ninth of its entire weight.

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 109.

Continuing, Peters states:

> It is obvious, then, that there would be little object in smelting unroasted ores carrying a high percentage of metallic sulphides in a blast furnace running on carbonaceous fuel.

From what has been stated, it would appear that electric heat could be used as well for the melting down of the ore as heat derived from the combustion of coke.

However, such melting would not, in either case, give the concentration desired. The concentration of the copper and accompanying precious metals into a matte must be by the removal of sulphur and the subsequent oxidation of the iron.

The removal of sulphur in ordinary blast-furnace smelting is brought about in two ways, namely, by heat alone, which drives off a part of the sulphur as elemental sulphur, and from the reaction of oxides and sulphates, present in the ore as such, on the sulphides. The presence of oxides in the charge may be due to the use of natural oxide ores, whereas if roasted ores are used, both oxides and sulphates may be present. These components react with the sulphides present, a partial oxidation of the latter takes place, and some of the sulphur is removed as sulphur dioxide (SO_2). It is to be remembered that the atmosphere of the ordinary blast furnace is reducing, and therefore in this instance these reactions have to take place in a reducing atmosphere, and for this reason do not take place as freely as they would in a neutral atmosphere. In general it may be stated that with no oxides present in the charge practically no sulphur (aside from that removed as elemental sulphur by heat, as above stated) would be driven off except, as stated by Peters,[a] "the small amount that would be oxidized by the blast, in spite of the generally reducing atmosphere of the furnace." He also states that, judging from his own experience, this "loss of extra sulphur in the blast furnace can not be placed higher than 5 per cent of the sulphur remaining in the ore after deducting the sulphur which the pyrite and chalcopyrite lose by direct volatilization."

POSSIBILITY OF SUBSTITUTING THE ELECTRIC FURNACE.

Ordinary blast-furnace smelting of copper ores consists simply of melting down the ore, and the sulphur removed is that which is removed when a sulphide is heated without air. Any concentration that takes place is due to the reactions which take place between the oxides and sulphates present, and which would take place better in a neutral atmosphere than they do in a blast furnace, the atmosphere of which is reducing. Such being the case, a blast furnace operating in this manner is only a melting-down furnace, and the same results would be obtained by melting such an ore in an open crucible. The only advantage which the blast furnace has over the reverberatory furnace in this instance is that it is more efficient as a melting furnace than is the reverberatory, because it affords intimate contact between the fuel of the charge and the resultant gases and permits of continuous smelting. On the other hand the reactions between the oxides and sulphates do not take place as readily as they would in the comparatively neutral atmosphere of the reverberatory furnace, or in the still more neutral atmosphere of the electric furnace. So far, then, aside from the question of costs, there is apparently no reason why the electric furnace could not be substituted for the ordinary blast furnace in smelting the class of ores above mentioned.

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 116.

LOSSES OF METALS.

According to Peters,[a] the losses which occur in ordinary blast-furnace work are (1) the loss of dust, which contains valuable metals, every time the ore is moved; (2) losses of dust in the flue gases, owing to the blast; (3) volatilization losses; and (4) losses in slag "due to particles of matte or metal which have failed to separate properly from the slag and, to a lesser degree, from the oxidation of the oxides of the valuable metals, especially lead and copper, that have entered into combination with the silica and have thus become chemically part of the slag."

The relative extent to which these losses would occur if an electric furnace (with no blast) were used instead of a blast furnace operated in the manner previously described will now be discussed.

Loss of dust in handling the ore would be common, of course, both to the blast furnace and the electric furnace.

Although there would be more or less fine dust carried away by the gases escaping from the electric furnace, the amount, of course, would not be as great as with the blast furnace.

As regards loss by volatilization, an electric furnace, used for smelting sulphide ore, would be operated as a resistance furnace (the charge acting as the resistor) rather than as an arc furnace, and hence the heat in the vicinity of the electrodes would not be as intense as if an arc were used, and probably not greatly in excess of that which is generated at the tuyères of such a blast furnace as has been considered.

If an electric blast furnace of the type shown in figures 5 and 6 be used, the volatilization would take place at the base of the shaft of the furnace, and hence the volatilized metals would have a chance to be deposited out on the cold part of the charge in the upper part of the stack, and as it would not be necessary to use a blast when operating

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 113.

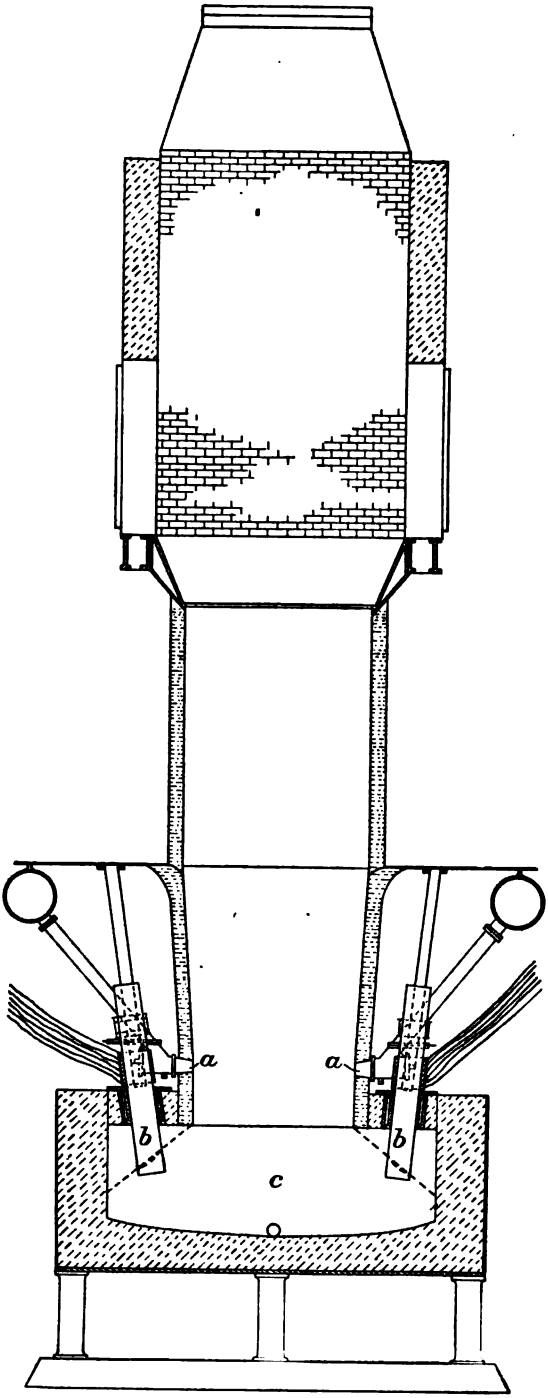

FIGURE 5.—Elevation of electric blast furnace. *a*, tuyères; *b*, electrodes; *c*, crucible.

the electric furnace for the purpose under discussion—the blast of the blast furnace in this particular instance is used only for furnishing air for burning the coke to produce the heat necessary for the process—the volatilization loss should not, and probably would not, be as great as in the blast furnace.

As regards the losses in the slag owing to particles of matte or metal failing to separate properly from the slag, such losses would occur in the electric furnace as well as in the blast furnace, but there is no reason to believe that they would be any greater and on the other hand they should not be as large, because the electric furnace could be regulated to impart a sufficiently high temperature to the bath before it leaves the furnace to insure its being fluid enough to cause perfect separation of matte and slag in the settler, which would thus obviate this loss. As to the oxidation of valuable metals, this loss should be reduced to a minimum in a furnace operating without a blast, for the loss by oxidation is doubtless due to the action of the blast upon the products of the furnace as they pass the region of the tuyères.

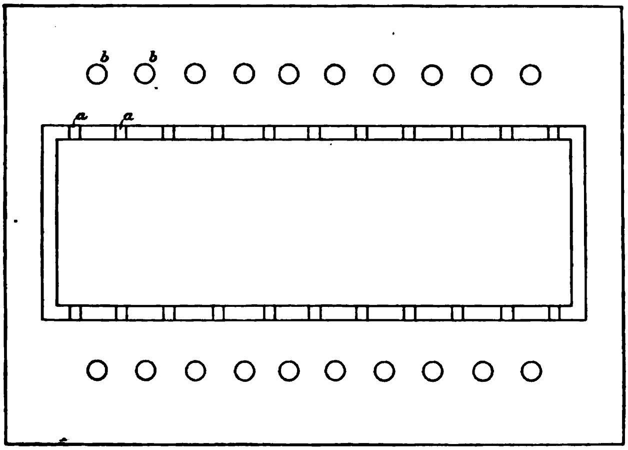

FIGURE 6.—Plan of electric blast furnace. *a*, tuyères; *b*, electrodes.

DATA ON EXPERIMENTAL SMELTING OF RAW SULPHIDE, OXIDIZED, AND CARBONATE ORES.

Having thus made a brief comparative study of the possibility of using an electric furnace for the smelting of mixed sulphide and oxide ores—it being understood, of course, that the object of this paper is not to prove that the electric furnace should replace the combustion furnace, but to determine, if possible, whether it can be substituted for the combustion furnace in those localities which are not favorable to the latter—it may be well at this point to consider some of the experimental work which has been done along the lines above indicated.

EXPERIMENT BY VATTIER.

One of the best known experiments on the smelting of copper ores in the electric furnace was made on April 23, 1903, by Vattier[a] at the works of the Compagnie Electrothermique Keller et Leleux, at Livet, France. The test was made by Vattier for the Chilean Government, which wished to procure such data as it would be possible to obtain in making experiments on a commercial scale. The test was made before a commission which was composed of some of the best known metallurgists of England and continental Europe.

As stated by Vattier, the main object of the experiment was to determine, if possible, whether in smelting copper electric heat could be substituted either wholly or in part for the heat derived from coal. This is especially important to Chile, for it has plenty of valuable copper ores, so it is stated, and an abundance of water power, but coal is costly.

METHOD AND RESULTS OF TEST.

The experiment was conducted with two kinds of ore, namely—

(1) An ore carrying approximately 7 per cent of copper which was present principally as copper pyrites, 8 to 9 per cent of sulphur, the gangue consisting mainly of micaceous copper oxide, together with some silicates and a little carbonate of lime; and

(2) A low-grade copper ore from the mining regions in the vicinity of Santiago, Chile, mixed with a small proportion of manganese and lime.

In the experiment a shaft furnace was used which had a smelting chamber 70 inches long, 35 inches wide, and 35 inches high. The melted material was run into a forehearth 47 inches long and 23 inches high. Two carbon electrodes, 11¾ inches square by 66 inches long, were used in the smelting chamber, and two electrodes 9¾ inches square by 39 inches long were put in the forehearth to reheat the bath for tapping.

The furnace used, so Vattier states, "Had a capacity of 25 tons of ore per 24 hours. The current used for the experiments was 4,750 amperes, 119 volts, cosine ϕ equals 0.9, and the energy corresponded to 500 kilowatts, or 680 horsepower."

The charge used was a mixture of the two ores. About 8,000 kilograms of ore was smelted in eight hours. The composition of the charge was as follows:

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[a] Haanel, E., Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in operation in Europe, Mines Branch, Department of the Interior, Canada, 1909, pp. 215-223.

Composition of charge.

	Per cent.
Carbonic acid	4.310
Silica	23.700
Alumina	4.0
Lime	7.300
Magnesia	0.33
Sulphur	4.125
Iron	28.500
Manganese	7.640
Phosphorus	0.046
Copper	5.100
Arsenic	Trace.

Matte and slag of the following composition were obtained:

Analysis of matte and slag.

Constituent.	Composition of matte.	Composition of slag.
	Per ct.	*Per ct.*
Silica	0.80	27.20
Alumina	.50	5.20
Lime	....	9.90
Iron	24.30	32.50
Manganese	1.40	8.23
Sulphur	22.96	.57
Phosphorus	.005	.06
Copper	47.90	.10

DISCUSSION OF RESULTS.

Vattier remarks that the slags at both the beginning and end of the run usually contained a slightly higher proportion of copper. He does not say why this was so, or whether the copper present was free or combined. Uncombined copper would probably be present in larger amounts at the beginning of the run, because of the viscosity of the slag, due to lack of heat, preventing proper settling; also a slightly higher proportion of copper at the end of the run would probably be due to the same cause—that is, improper settling.

Vattier also states that the slag, being high in iron and silica, could be used to manufacture ferrosilicon, but, as has been previously pointed out in this bulletin, the results of experiments by the Bureau of Mines indicate that such utilization of slag is impractical.

Vattier also states that, "For good results it is desirable to use a voltage sufficient to cause the arc or electric current * * * to just clear the surface of the bath, in order to avoid as much as possible their (the electrodes) coming in contact with the bath."

His reason for wishing to keep the electrodes out of contact with the bath is explained by his statement that "Carbon at such high temperature has a tendency to reduce the iron oxide into metallic iron, which gives rise to the following inconveniences: (1) A more rapid wear of electrodes; (2) loss of electrical energy; and (3) decrease of the copper contents of the matte."

It is of course true that, if the electrodes extend down into the charge, as they do when the furnace is operated as a resistance furnace (the charge acting as the resistor) instead of an arc furnace, any iron oxide coming in contact with the carbon of the electrodes will be reduced by the carbon and thus cause a wearing away of the electrodes. On the other hand, in tests which have been conducted by members of the bureau and others, it has been found that more satisfactory results are obtained if the charge be smelted by resistance heating, using the charge as the resistor, than if it be smelted by heat from an arc. With the arc a very high local temperature is obtained and hence volatilization losses are higher than if resistance heating, which heats the charge more uniformly, is employed. The loss of electrical energy and the production of a low-grade matte resulting from the reduction of metallic iron has been previously discussed under experiments by the writers.

EXPERIMENTS BY SCHILOWSKI.

SMELTING MIXTURES OF RAW AND ROASTED ORE.

More recently some experiments were conducted by J. Schilowski [a] for the purpose of determining whether it would be possible to obtain in one operation a reaction between raw ore and roasted ore that would give a usable copper and a clean slag, Schilowski states that in these experiments he used a furnace which "had an upper carbon electrode and a water-cooled lower electrode," but he does not give much information in regard to the furnace beyond this. It was probably a small furnace, holding, perhaps, 25 pounds of ore, of the Siemens vertical type, shown in figures 1 and 2.

Analyses of the raw and roasted ore used in the tests are as follows:

Analyses of raw and roasted ore used in experiment.

	Raw ore.	Roasted ore.
	Per cent.	*Per cent.*
SiO_2	20.30	22.5
Cu	10.10	10.5
Fe	27.10	34.6
Al_2O_3	9.45	11.5
As	9.25	Trace.
Mn	.48	.5
S	15.09	1.5
Bi	Trace.	....
Co		....
Au		....
Ag	([b])	....

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[a] Schilowski, J., La fusion electriques des minerais de cuivre et des produits intermediares de fonderies de cuivre: Revue Met., vol. 9, 1912, p. 205.

[b] 0.116 ounces per ton.

The amount of copper unaccounted for was very large in most of the tests, especially when only roasted ore was used. The unexplained losses are probably due to volatilization and mechanical losses, especially the latter, for the gases arising from a small electric furnace carry with them a large amount of fine particles.

A second group of experiments was made to determine what proportions of raw ore should be used in the charge in order to obtain the largest percentage of copper in the matte with the minimum difficulty of fusion.

Schilowski states that in the experiments he used "two electrodes of carbon projecting into the slag." From this the authors judge that the fusion was done much the same as it would be if a charge were fused in a single-phase Héroult steel furnace.

DISCUSSION OF RESULTS.

As a result of his experiments, Schilowski states that the proportion of raw ore that should be used in making up a charge depends on the consumption of energy and the loss of copper in the slag.

He obtained these conditions with a charge containing 60 to 70 per cent of roasted ore. With such a charge, the loss of copper in the slag was 5 per cent of the total copper present in the charge, and the weight of the copper in the speiss was one-ninth of the weight of the copper in the matte.

He also found that the maximum energy consumption was when the charge was made up entirely of crude ore, and the minimum energy consumption was when the charge contained 64 per cent of roasted ore. He states that in treating a mixture of ores of this nature that 17,400 kilowatt-hours would be required per ton of copper produced.

He also states that the high losses of copper were due to the violence of the furnace reaction. The writers have not noticed that the reactions were especially violent in any of the experiments of this nature which they have performed. It is true that after the charge becomes melted it bubbles or boils considerably, due to the gases which are being given off, and it is also true that, in order to get a satisfactory separation of matte and slag, the fused product must be kept at a melting temperature for some little time, just as is done in a settler in commercial smelting. That this is not always done in experimental work may account for some of the high losses of copper in the slag.

RECOVERY OF ARSENIC FROM RAW ORE.

Another experiment was made by Schilowski in order to determine whether a recovery of arsenic could be made by fusing the raw ore.

METHOD OF TEST.

The fusion was made in a carbon crucible with a bottom carbon electrode. A carbon resistance of carbon block was placed in the crucible between the electrodes. A covering was placed around the crucible in order to prevent loss of heat by radiation. The gases were led off through a tube and then through several condensing chambers.

RESULTS OF TEST.

No speiss was obtained in this experiment. The matte contained 21.37 per cent of copper, 50.27 per cent of iron, and 28.39 per cent of sulphur. The slag contained only 0.15 per cent copper. In treating a quantity of ore corresponding to a ton of copper 27,500 kilowatt-hours were used per ton of copper produced.

An analysis of the fine material obtained as a result of the volatilization was as follows:

Composition of volatilized material.	Per cent.
As	60
Fe	2
Cu	1
S	7
Miscellaneous	16

Schilowski states that in spite of the imperfections of the condensing apparatus, 65 per cent of the arsenic was recovered, and that it would be feasible to attempt the recovery of arsenic from the raw ore in this manner.

EXPERIMENTS BY WOLKOFF.

Wolkoff [a] made some experiments on the smelting of a copper sulphide ore, containing 8.20 per cent of copper, with an acid gangue. Several experimental runs were made with this ore. The results from one or two of them will be briefly discussed.

SMELTING ORE AND IRON OXIDE TO OBTAIN MATTE.

In one experiment, by smelting 12 kilograms (25.5 pounds) of ore, with 6.2 per cent of hammer scale (iron oxide) added to furnish iron oxide for the siliceous gangue, he obtained a thoroughly fused product, a matte containing practically the whole of the copper, and a slag retaining only 0.15 per cent of the copper. He also states that the volatilization losses were very small.

SMELTING ORE AND MATTE TO OBTAIN METALLIC COPPER.

In another experiment he smelted 10 kilograms (22 pounds) of ore with 1.25 kilograms (2.74 pounds) of roasted matte, containing 75 per cent CuO, 8 per cent Cu_2S, and 15 per cent Fe_2O_3, the object being to determine the extent to which the reaction $Cu_2S + 2\ CuO = 2\ Cu_2 + SO_2$, resulting in the production of metallic copper, would take place. On smelting this charge with a current of 400 amperes at 75 volts for half an hour, Wolkoff obtained 1.78 kilograms (3.91 pounds) of metal which was 92 per cent copper, 3 per cent iron, and 1 per cent sulphur. He also obtained 10.9 kilograms (2.39 pounds) of slag with a copper content of 0.10 per cent. In this experiment 96 per cent of the copper was extracted; the slag contained 0.6 per cent of the total quantity of copper; therefore, as stated by Wolkoff, the copper balance is as follows:

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[a] Wolkoff, W., Electric smelting of copper sulphide ore: Metallurgie, vol. 7, 1910, p. 99.

	Kilograms of copper.
1.78 kilograms of crude copper (92 per cent copper)	1.638
10.9 kilograms of slag with 0.1 per cent copper	0.111
Losses	0.057
Total copper in charge	1.706

EXPERIMENTS BY STEPHAN.

At the first general meeting of the recently formed society of German Metallurgical Engineers, M. Stephan, superintendent of the Girod electric steel works, at Ugine, France, gave an account of some experiments made by him on the electric smelting of nonferrous ores, one of which was a copper ore. The following is an abstract of Stephan's paper:[a]

SMELTING NONFERROUS ORE IN THE ELECTRIC FURNACE.

The ore came from the Belgian Congo, being mined by a Belgian-English concern. In five different analyses given the CuO varies from 21.01 to 5.73 per cent; SiO_2 from 28.48 to 78.55 per cent; Al_2O_3 from 4 to 13 per cent; Fe_2O_3 from 4 to 16 per cent; and, besides smaller amounts of other impurities, 2 to 7 per cent CoO, and no nickel.

The moisture in the ore varied from 7 to 32 per cent. It was not removed, in order to meet the conditions of practical operation and to carry the experiments out under most unfavorable conditions.

Charcoal, coke, and anthracite were used successfully as reducing agents. Charcoal would probably be the cheapest in this particular instance.

Electric furnaces similar to the Girod ferro-alloy furnace were used, and the dimensions changed in several runs within wide limits in order to secure enough data for the construction of a large furnace for the same work. Electrodes were suspended from the top and inserted in the bottom, also in the sides for heating by radiation, and a system was adopted of heating with a smothered arc, mainly by the resistance of a thick layer of slag over the conducting metallic charge. The temperatures were measured with Le Chatelier and Fèry pyrometers.

A slag of the composition 51.9 per cent SiO_2, 11.31 per cent Al_2O_3, 16.83 per cent CaO, 13.71 per cent MgO, 3.55 per cent Fe_2O_3, 0.94 per cent MnO, 0.46 per cent CuO, and 0.87 CoO begins to melt at 1,250° C. and is liquid enough at 1,400° C. to allow the little copper balls to settle completely. At 1,550° C. the slag is liquid enough to flow freely; 1,920° C. were necessary to render the slag liquid when the highly acid charge was smelted without any fluxes.

The pig copper (Schwarzkupfer) produced in six different runs analyzed from 65 to 95 per cent copper, from 1 to 21 per cent iron, and from 1 to 11 per cent cobalt. The lower the smelting temperature the higher will be the purity of the product. The lower the temperature the smaller will be the chance for reducing any of its impurities, but at the same time more copper will also be retained in the slag, so that the output is decreased.

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[a] Stephan, M. Einiges über die Erzeugung von Metallen im elektrischen Ofen: Metall und Erz, vol. 1, 1913, p. 11; abstracted in Met. and Chem. Eng., vol. 11, 1913, p. 22.

CONCLUSIONS OF STEPHAN.

Stephan's conclusions as to the results of the experiments were as follows:

> It will be a matter of commercial calculation, according to the composition of the ore and other special conditions, whether a high quality product or a maximum output will be desirable.
>
> A continuous run for several days with the same ore, aiming at slags of about the above-mentioned composition, required 1,000 to 1,200 kilowatt-hours per ton (2,205 pounds) of ore. These figures are high as a result of the amount of heat required to keep the very viscous and abnormal slags in liquid condition.
>
> With an easily fluxible ore the power consumption was only 500 kilowatt-hours per ton.
>
> The electrode consumption averaged 8 kilograms, or 17.6 pounds per ton of ore. It (the electrode) was operated with 4 amperes per square centimeter (25.8 amperes per square inch).
>
> Coal for reduction was used at the rate of 25 per cent of the copper in the charge.
>
> The best lining to withstand the severe conditions of the furnace was tamped from fire clay with 80 per cent SiO_2 and 15 per cent Al_2O_3.

EXPERIMENTAL SMELTING OF SULPHIDE ORES IN THE ELECTRIC FURNACE.

A series of experiments on the smelting of sulphide ores in the electric furnace was recently conducted at the Bureau of Mines laboratory by the writers. The objects in view were: (1) To determine if there are any conditions arising which make electric smelting without air anything but a simple melting operation; (2) to note the percentage of concentration and the sulphur removal; (3) to study the possibility of condensation of the elemental sulphur as such; (4) to get general figures on power consumption with varying charges; (5) to determine losses of gold, silver, and copper, and (6) to study the use of a low-grade copper matte as a collecting agent for gold and silver.

ORES TREATED.

The ores used in the experiments were a low-grade sulphide copper ore high in sulphur, a gold and silver bearing siliceous ore, and some roasted ore; the analyses of the ores are given below. Limestone containing 63.2 per cent CaO and 5.7 per cent MgO was used to flux the ore.

Analyses of ores used in electric smelting of sulphides.

Constituent.	Raw pyritic ore.	Nodule.	Siliceous ore.
	Per cent.	*Per cent.*	*Per cent.*
SiO_2	2.50	4.00	74.53
Fe	44.07	65.60	10.71
S	48.20		8.36
Al_2O_3	.24	1.76	.40
CaO	.08	.45	
MgO	.73	.46	.26
P	.02		
As	.20		
Cu	1.30	.07	
	Ounce per ton.	*Ounce per ton.*	*Ounces per ton.*
Au	0.01	0.03	1.28
Ag	.14	.05	3.20

METHOD USED AND RESULTS OF TESTS.

The ores were smelted in the electric furnace that was used in smelting the native copper concentrates. The furnace is described on pages 17 to 21. The top was roofed and kept closed tightly to prevent escape of sulphur and admission of air. A condenser, which consisted simply of three rectangular horizontal chambers with several baffles in them, was attached to the furnace to condense the sulphur and catch any escaping dust. Twenty experiments were made in all. The results of five are given in Table 3.

TABLE 3.—*Results of smelting copper sulphide ores in the electric furnace, experiments 1 to 5.*

	1	2	3	4	5	Average.
Charge:						
Raw pyritic ore, pounds	12.10	8.80	7.93	6.18	5.28	....
Nodule, pounds	9.90	13.20	11.90	19.22	7.81	....
Silicious ore, pounds	3.10	5.76	8.86	10.75	12.50	....
Limestone, pounds	....	2.73	3.01	3.02	3.03	....
Matte produced:						
Copper, per cent	1.03	1.22	1.33	1.05	1.00	1.12
Iron, per cent	63.99	64.18	64.18	56.21	62.49	....
Sulphur, per cent	24.87	22.38	25.47	20.28	24.03	....
Gold, ounces per ton	.28	.72	.80	1.60	2.33	....
Silver, ounces per ton	.68	.96	2.00	4.00	3.60	....
Slag produced:						
Copper, per cent	.05	.05	.24	.13	.29	.15
SiO_2, per cent	47.88	35.25	41.75	53.00	53.05	....
FeO, per cent	40.30	41.30	39.05	25.90	26.20	....
Al_2O_3, per cent	9.24	4.40	5.09	4.01	3.57	....
CaO, per cent	1.33	10.00	10.21	10.37	10.37	....
MgO, per cent	.10	1.64	.10	.14	.11	....
S, per cent	3.84	3.66	3.21	2.23	2.36	....
Gold, ounce per ton	.02	.03	.08	.04	.08	.05
Silver, ounce per ton	.16	.32	.17	.29	.34	.25
Length of run, minutes	35	26	27	20	18	....
Mean volts on the furnace	53	51	56	70	80	62
Mean amperes on the furnace	656	635	920	792	726	745
Mean kilowatts on the furnace	31	32	51	54	56	46
Mean power factor of circuit	.89	.98	.98	.97	.96	.95
Kilowatt-hours (by meter)	18.00	13.5	25.5	17.0	15.0	17.8
Electrode consumption per short ton of charge, pounds	20.2	10.8	18.6	18.6	18.6	17.3
Weight of matte tapped, pounds	[a] 13.45	10.50	7.00	8.42	9.69	....
Weight of slag tapped, pounds	4.74	10.90	16.90	17.84	19.70	....
Sulphur volatilized, per cent	65.9	63.2	57.0	65.1	54.6	61.7
Concentration ratio	1.57	2.84	4.16	3.60	3.85	3.2
Loss of copper in slag, per cent	1.21	4.26	([b])	([b])	([b])	2.73
Copper unaccounted for, per cent	8.48	....	([b])	([b])	([b])	....
Total recovery of copper, per cent	90.31	95.74	[b] 90.30	[b] 100.00	[b] 100.00	95.3
Loss of gold in slag, per cent	2.16	4.18	11.40	5.07	([b])	5.7
Gold unaccounted for, per cent	11.75	....	10.5	0	([b])	....
Total recovery of gold, per cent	86.09	95.82	78.1	94.93	95.2	90.0
Loss of silver in slag, per cent	6.28	17.20	9.40	....	16.20	12.2
Silver unaccounted for, per cent	15.40	32.90	12.30	....	8.37	17.2
Total recovery of silver, per cent	78.32	50.10	78.30	94.50	75.10	75.2
Total value of copper, silver, and gold per ton of charge,[c] dollars	4.96	6.76	8.95	11.14	12.90	....
Value of copper, silver, and gold recovered per ton of charge, dollars	4.29	6.34	7.21	10.62	12.17	....
Total value of copper, silver, and gold recovered, per cent	86.7	94.0	80.8	95.8	94.2	90.3
Kilowatt-hours per short ton of charge	1,435	890	1,610	1,064	1,045	1,209
Kilowatt-years per ton of charge	.164	.101	.184	.13	.12	.174

[a] From previous run.

[b] In this and subsequent experiments the copper content of both matte and slag is high, probably because of a concentration of copper in the furnace from the numerous experiments made previously.

[c] Copper at 15 cents per pound, silver at 50 cents per ounce, gold at $20 per ounce.

DISCUSSION OF RESULTS.

In these experiments the furnace was charged and, when fusion was completed, tapped as cleanly as possible. Hanging of the charge and accumulation of material in the furnace are responsible for some results not being uniform. An attempt was made to make the results as quantitative as possible, but with a small furnace the losses affect the final calculation much more than with a furnace of commercial capacity.

SMELTING CONDITIONS, SULPHUR VOLATILIZATION, AND RATIO OF CONCENTRATION.

If the electric furnace acts simply as a melting agent in the smelting of sulphide ores, the fact will be apparent in the amount of sulphur volatilized and the ratio of concentration. In simple melting of a charge of pyrite one atom of sulphur would be driven off from the FeS_2, or 50 per cent of the sulphur in the charge would be volatilized. To determine the extent to which this reaction could be carried in the electric furnace, the crucible was roofed and was kept closed, as far as possible, in order to prevent oxidation. In the five experiments given in Table 3 an average of 61.1 per cent of the sulphur in the charge was volatilized, showing that a little more than one atom of sulphur was driven off. Hence without the introduction of an oxidizing agent the concentration ratio in the electric smelting of copper sulphide ores can not be increased much by volatilization of the sulphur in the charge.

The low ratio of concentration of 3.2, the average for five runs, verifies this statement and shows that no concentration is to be expected beyond a simple separation of gangue and matte and the loss of slightly more than one-half of the sulphur. As had been stated, there is no slagging of iron oxide beyond that of the iron oxide present in the ore as such, because there is no oxygen present to oxidize the iron sulphide formed by volitilization of the one atom of sulphur.

The smelting conditions in an electric furnace treating sulphide ore without introduction of air are then similar to those in a reverberatory furnace or a blast furnace in which the heat is supplied almost entirely by coke. The electric smelting of a copper sulphide ore under these conditions is merely a melting operation, in which reactions occur between the oxides and the sulphates of the charge and a removal by heat of a little more than one atom of sulphur.

The amount of sulphur driven off in the blast furnace, running with a reducing atmosphere, over and above that accounted for by the loss of one atom per molecule is not more than 5 per cent, whereas in the electric furnace, as these experiments show, it is about 10 per cent.

The condensation of the sulphur driven off as elemental sulphur was qualitatively possible, as was shown by analysis of the fume in the condenser, but was not quantitatively possible, because the condenser was not long enough for complete condensation of the fume, and some of it came out of the end. The uncondensed fume consisted chiefly of elemental sulphur. An average analysis of the condensed material was as follows: SiO_2, 13.96 per cent; Al_2O_3, 3.85 per cent; Fe, 9.11 per cent; CaO, 2.06 per cent; S, 46:6 per cent; Cu, 0.61 per cent; gold, 0.48 ounces per ton; and silver, 4.60 ounces per ton. A very small amount of dust from the ore was deposited at the mouth of the condenser.

COPPER, GOLD, AND SILVER LOSSES, AND LOW-GRADE MATTE AS A COLLECTOR.

The average copper content of the slag was 0.15 per cent. With proper means for settling the matte instead of tapping into a mold, the low copper content should be reduced to a still lower figure. The average percentage of copper recovered was 95.3 per cent. This shows that there was no material volatilization of copper, for the condensed fumes, which amounted to only 4.75 per cent of the total ore charged, contained only 0.61 per cent copper.

The percentage of gold recovered in the matte was 90 per cent of the total quantity of gold in the ore charged. The remaining 10 per cent was not all held by the slag, as the average loss of gold in the slag was 5.7 per cent of the total amount of gold in the ore charged. The other 4.3 per cent was probably lost mechanically in handling the charge and products. The slags contained 0.03 to 0.08 ounce of gold per ton. This indicates the collecting power of a low-grade copper matte for gold, as the copper content of these mattes varied from 1 to 1.33 per cent and averaged only 1.12 per cent. The charge contained from 0.31 to 0.83 per cent copper, the average being 0.46 per cent.

These results agree with the results of experiments by Dr. Carpenter[a] at Deadwood, S. Dak., where for several years the plant was run with mere traces of copper in the matte and never with more than 1 per cent. These slags carried from 0.025 to 0.075 ounce of gold per ton. As at Deadwood the experimental runs conducted by the writers show a lower gold loss in the slag with a greater matte fall. The results of these experiments seem to indicate that a matte containing 1 per cent copper will serve as a good collector of gold in the electric furnace and that a charge containing as low as 0.31 per cent copper may be successfully used.

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 426.

The silver recovery was not as high as that of the gold, the total percentage of silver recovered being 75.2 per cent. A large part of the silver loss must be charged to the unaccounted loss, which includes mechanical losses and loss by volatilization. The proportion of silver in the fume was higher than the proportion of gold, a fact which, with the greater unaccounted loss, indicates that there was volatilization of silver. The average silver content of the slag was 0.25 ounce per ton, which can not be considered as very high, so that the low-grade matte seems to collect the silver as well as the gold.

In order to ascertain the actual saving of values, copper is figured at 15 cents per pound, gold at $20 per ounce, and silver at 50 cents per ounce. On this basis the average value recovery for five experiments was 90.3 per cent. This would indicate that it is feasible to use a matte containing 1 per cent copper in the smelting of an ore for recovery of gold and silver. With a higher matte fall the recovery would be greater. The loss in the slag seems to depend mostly on the proper separation of slag and matte. With an acid slag this separation is easy, but with a basic slag, as the specific gravity of the slag and that of the matte approach each other, separation becomes more difficult. As was stated by a recent writer on the subject, "Copper is not essential in mattes which accompany acid slags, but with basic slags it may be necessary."

ELECTRODE CONSUMPTION.

The average electrode consumption was 17.3 pounds per ton of charge, the lowest figure obtained being 10.5 pounds. In practice, with a large furnace the electrode consumption should be easily reduced to 5 pounds per ton of charge. The reduction of iron by the carbon electrode has been shown, in the experiments previously discussed, to be of small account.

POWER CONSUMPTION.

The power consumption was abnormally high in all of the experiments given in Table 3, with the exception of No. 2, because the runs were continued longer than was necessary, owing to the difficulty of telling when fusion was complete with the furnace top roofed. A power consumption of 890 kilowatt-hours per ton was attained several times in experiments not given in this bulletin and may be taken as being about the best it was possible to obtain with the furnace used. The efficiency of this furnace had been determined to be 36.76 per cent. On the basis of a properly designed furnace of several thousand kilowatts capacity in which an efficiency of 70 per cent should be attained, the power consumption would be (890×36.76)÷70=480 kilowatt-hours per ton of charge smelted, or 0.055 kilowatt-years. From the general trend of 20 experiments on the smelting of charges similar to those given in Table 3, there seems to be a slight increase in power consumption with a higher proportion of pyrite in the charge.

CONCLUSIONS.

1. The smelting of sulphide ores in an electric furnace without admission of air consists simply of melting the charge, volatilization of about 60 per cent of the sulphur as elemental sulphur, and separation of the slag and matte.
2. The ratio of concentration is simply that possible from separation of matte and slag, reaction of oxides with sulphates and sulphides, and the volatilization of about 10 per cent more than one-half of the molecular sulphur, with no formation of iron oxide to enter the slag, because there is no oxidizing agent present to oxidize the iron sulphide.
3. Qualitatively, it is possible to condense some of the elemental sulphur driven off.
4. The loss of copper by volatilization and in the slag is low.
5. A matte containing about 1 per cent of copper from a charge containing 0.30 per cent copper makes a good collecting agent for gold and silver, if a clean separation of slag and matte is obtained.
6. There is very little loss of gold by volatilization in the electric furnace, but some silver is lost in this way. In a larger furnace with closer temperature regulation there would probably be no loss.
7. The electrode consumption in the smelting of a sulphide ore is low and in practice need not exceed 5 pounds per ton of charge.
8. In a large commercial furnace the power consumption for most ores would be about 480 kilowatt-hours per ton of ore, or 0.055 kilowatt-years.

COMPARISON OF ELECTRIC FURNACE WITH REVERBERATORY FURNACE AND BLAST FURNACE AS A MELTING AGENT.

So far only the use of the electric furnace as a melting furnace in the treatment of copper ores has been considered. From what has been stated, the authors believe that they are justified in making the following statements as to the possibility of using the electric furnace for this purpose.

1. Native copper, oxide, or sulphide ores of copper can be melted just as efficiently, and perhaps even more efficiently, in the electric furnace than in either the reverberatory furnace or the blast furnace.
2. The reactions desired in reverberatory smelting, or ordinary blast-furnace smelting, can be obtained just as well and perhaps better in the electric furnace than in either of those furnaces.

3. The loss of electrodes is small, varying from 5 to 10 pounds per ton of ore smelted, and the presence of the carbon electrode does not cause enough reduction of iron to be troublesome or to increase the consumption of electrical energy appreciably.

4. The losses of copper, gold, and silver by volatilization and in the slag would be no greater, as is shown in all the experimental work cited, than they are in reverberatory smelting or ordinary blast-furnace smelting.

5. A matte containing as low as 1 per cent of copper can be used as a collecting agent for gold and silver in the electric furnace as well as in the combustion furnace.

6. The cost would depend entirely on the nature of the ore to be treated and the relative cost of coal or coke and electrical energy. In general, from the work done in the experiments, it may be said that from 500 to 700 kilowatt-hours per ton of charge, depending on the nature of the ore, would be required for smelting copper ore.

SEMIPYRITIC SMELTING OF SULPHIDE ORES.

PRINCIPAL FEATURES OF THE PROCESS.

In discussing the smelting of copper ores by ordinary blast-furnace smelting, the fact has been noted that the process consists essentially in melting down the charge, and that during the melting certain reactions take place between the constituents of the charge which bring about a concentration of the copper and precious metals (if the latter are present) into a matte and the excess iron and the gangue materials into a slag, that in this method of smelting the heat necessary for carrying out the process is obtained by the combustion of coke at the tuyères, and that the only purpose in using the blast is to supply the oxygen required for combustion of the coke. In other words, the oxygen entering the tuyères plays no part in the reactions which take place in the furnace so far as the constituents of the charge other than coke are concerned.

Such being the case, the essential feature of the process is the melting down of the charge and, so far as the reactions of the process are concerned, the charge could be melted down just as well in an electric furnace as in a blast furnace. Those processes of copper smelting in which the air entering at the tuyères not only furnishes the oxygen necessary for producing heat, but also that required for bringing about the reactions which take place during the melting, and which result in the production of a matte and a slag, as in ordinary blast-furnace smelting, will now be considered. In this connection it is to be remembered that no matter what process is employed in obtaining metallic copper (as a final product) from a sulphide the three essential features of the process are: (1) The removal of sulphur; (2) the oxidation of the iron with which the sulphur was combined; and (3) the removal of this iron oxide by causing it to unite with silica to form a slag.

Hence, as stated by Peters[a], the first step in the smelting of a sulphide ore of copper must be to oxidize it. In the process that will now be considered the object is to bring about the required degree of oxidation and the melting at practically the same time, and to obtain either the whole or a part of the heat required for the process from the oxidation of the iron and sulphur contained in the ore. As is well known, if practically all the heat necessary for melting down the charge is obtained in this manner, the process becomes what is known as pyritic smelting, and if only in part, as semipyritic smelting. However, Peters[b] states that, so far as he is aware, in all pyrite furnaces of the world a small amount of fuel (usually coke) is added to the charge, and this amount may vary anywhere from 0.5 per cent of the weight of the charge up to a proportion that might be actually enough to melt the ore without any heat at all being derived from the sulphides.

SUBSTITUTION OF ELECTRIC HEAT FOR HEAT DERIVED FROM COKE.

The question, therefore, that presents itself in connection with the smelting of copper sulphide ores in the electric furnace is to determine whether electric heat may be used to replace the heat which is derived from the combustion of coke in pyritic and semipyritic processes. In other words, whether it would be possible, and if so, whether it would be commercially feasible, to carry out these processes in an electric furnace so constructed as to permit the use of a blast, thus obtaining as much heat as possible from the oxidation of the sulphur and iron, any additional heat required being supplied by electric energy.

To begin with, let it be assumed that the construction of the furnace is similar to that of a modern copper blast furnace and is practically the same for the upper part of the furnace, including the tuyères. Below the tuyères, *a*, the furnace could be constructed as shown in figure 5. The electrodes, *b*, extend down into the crucible, *c*, and are arranged along the sides of the crucible as shown in figure 6.

At this point will be considered briefly the behavior in the electric furnace of a charge that would be suited to semipyritic smelting; that is, one in which so much coke would have to be used that "its influence upon the oxidizing power of the focus begins to be plainly discernible," which, as stated by Peters,[b] is "the division line between true and partial pyrite smelting." Inasmuch as no coke would be used in a charge smelted in an electric blast furnace, the point to be determined is how the charge would act when such a furnace was used, and if the desired results could be obtained, as before stated, in a feasible and economical manner.

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 220.

[b] Peters, E. D., loc. cit., p. 214.

BEHAVIOR OF A CHARGE SMELTED WITHOUT COKE.

The object aimed at in operating a copper blast furnace in semipyritic smelting is to supply enough blast to completely oxidize the coke, and also to oxidize as much of the iron sulphide for slag-forming purposes as can be spared from the matte. If, therefore, there were no coke to be oxidized, only enough air need be forced through the tuyères to effect the required degree of oxidation of sulphur and iron.

In general, it may be assumed, as does Peters,[a] that in semipyritic smelting the furnace is "provided with but a scanty supply of pyrite" and that it produces a slag "rather high in silica and earths" and "consequently would require a considerable amount of coke." In the electric blast furnace, as before stated, coke would not be used, the equivalent heat value of the coke being obtained from electric energy instead. If, therefore, the electric blast furnace be supplied with a charge that is suited to semipyritic smelting, what difference may be expected in the behavior of the charge as compared to the behavior of a similar charge when smelted in a copper blast furnace with coke? In making the comparison the behavior and effect of the coke itself will first be considered.

EFFECT OF COKE IN THE BLAST FURNACE.

Peters[b] discusses the action of coke in the blast furnace as follows:

> The amount of coke present carries the melting process high up above the proper zone of oxidation, and to regions where there is yet no formation of FeO. Half-fused masses of acid earthy silicates are formed, and much, in some cases all, of the free silica is combined with the alumina, lime, magnesia, manganese, alkalis, and already oxidized iron, all of which substances are likely to be present in the ore mixture. It is not that the affinity of the silica is satisfied in forming these preliminary, temporary, refractory acid silicates, but the edge of its appetite is blunted and the eventual formation of ferrous silicate seems to proceed somewhat sluggishly, even when ample air is blown into the furnace.
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> As may be imagined the main endeavor of the metallurgist in this type of smelting is to keep the proportion of coke to ore as low as possible, not merely because coke costs money, but still more because any excess of coke causes a lowering in the grade of matte, due to its interference with the oxidation of the sulphides. A furnace in proper condition and running on a suitable charge is extraordinarily sensitive on this point. The increase of the coke from a standard charge of 60 pounds per 2,000 pounds of ore up to 65 pounds per 2,000 pounds of ore may be followed—as has come under my personal observation—by a dropping of the grade of matte from 35 per cent copper and a raising of the silica contents of the slag from 41 per cent. This results, of course,

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 326.
[b] Peters, E. D., loc. cit., p. 333.

from the consumption by the new coke of a certain amount of oxygen which previously had been employed in burning iron sulphide. Under this new condition this iron sulphide entered the matte at the same time, robbing the slag of just so much FeO.

REACTIONS IN THE ELECTRIC FURNACE.

If such a charge be smelted in the electric blast furnace without coke and with just enough blast to supply the oxygen required for oxidation of the sulphur and iron, there would doubtless be no "half-fused masses of acid, earthy silicates, formed," as stated by Peters, and since, as shown by him, the control of the coke is a matter of great importance, due to the "extraordinary sensitiveness of the furnace on this point" and its resultant great irregularities, it would seem that it would be a great advantage to do away, if possible, with coke. If coke can be dispensed with and the smelting be done in some such furnace as the one shown in figure 5, the charge would probably enter the tuyère zone of the furnace practically unaltered except, perhaps, for loss of elemental sulphur. Inasmuch as the sulphides are easily melted (950° C.), it may be assumed that there would be enough heat present from the oxidation of the sulphur and iron and radiated from the crucible to cause the iron sulphide to melt and to become oxidized to FeO and then be instantly seized by SiO_2 to form a slag. However at a temperature of 950° C. the result might be the formation of a silicate high in SiO_2, due to the fact that at low temperatures the saturation point of silica for iron is low, and this silicate would perhaps "freeze," so to speak, at once and descend into the crucible and be gradually melted in the neighborhood of the electric current. In other words, there would be formed above the tuyère zone the artificial boshes observed by Sticht, Freeland, and others, and described by Peters,[a] thus contracting the smelting and slag forming area "to a comparatively narrow opening running along the middle of the furnace shaft." Sticht states that these boshes "are the contact lines between the active combustion zone and the relatively dead part of the shaft above" and that the absence of coke (speaking of pyrite furnaces) accentuates the sharp division line between the active and stagnant regions of the furnace. Reasoning by analogy, it would seem that this division line would be as pronounced, if not more so, in an electric blast furnace, such as the one under discussion, as in the pyrite furnace.

COMPARISON OF ELECTRIC BLAST FURNACE WITH THE COPPER BLAST FURNACE FOR SEMIPYRITIC SMELTING.

METALLURGICAL CONSIDERATIONS.

To recapitulate, there does not seem to be any metallurgical reason why the chief objects of the pyrite smelter can not be carried out in an electric copper blast furnace as well as in a coke copper blast furnace; namely, to melt the great mass of SiO_2 and inert earths, and to oxidize enough of the sulphides in the charge to insure a suitable matte, incidentally obtaining the heat evolved by the oxidation. On the other hand it would seem that the difficulties ordinarily encountered in operating a pyrite furnace might be avoided when using an electric blast furnace, that is, the difficulties which arise when too much or too little coke is added to the charge. Moreover, electric heating not only permits "introducing heat into the furnace without at the same time robbing the combustion zone of oxygen"—in regard to which Peters[a] states, "nothing would be so welcome to the furnace man as to do this"—but would also permit of the heat being entirely under control and easily regulated and thus avoid "freeze-ups" with their consequent vexations and costly delays. By this latter statement it is not meant that any sort of charge could be put through the furnace and not "freeze," an idea which seems to be quite prevalent in regard to electric furnaces, but that, if the charge be properly calculated, a much wider variation in the composition of the slag from that calculated would be permissible than would be the case in ordinary blast-furnace smelting.

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[a] Peters, E. D., Principles of copper smelting, 1907, pp. 234, 235.

MECHANICAL CONSIDERATIONS.

By referring to figures 5 and 6 it will be noted that the chief difference in the construction of the electric blast furnace from that of the ordinary blast furnace would be in the part below the tuyères; in other words, the crucible of the furnace. Due to the fact that electric furnace construction has received the attention of some of the very best mechanical and electrical engineers, and because a crucible based upon the principle of the design shown in figure 5 is now extensively used in the electric iron reduction furnaces of Norway and Sweden, the writers believe that no difficulty would be experienced in this respect. Various methods and arrangements can be used for connecting up electrodes. If this be true, there remains to be considered only the matter of costs.

COMPARISON OF COSTS.

Inasmuch as it has already been stated that the use of the electric furnace as a competitor of the blast or combustion furnace, or that it should replace the copper blast furnace, is not advocated by the writers, it will be evident that a comparison of costs is made simply for the purpose of giving some idea as to the outlay that would be necessary to erect and operate an electric furnace plant for the purpose of smelting copper ores.

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 334.

Naturally, the first thing to be considered is whether it would be feasible to attempt to substitute electrical energy for coke, due to the cost of the former.

COST OF PLANT.

The cost of an electric-furnace plant would be about the same as the cost of a regular plant plus the cost of electrical installation but exclusive of the cost of the generating plant, as it is assumed that it would be possible to purchase electric power from some other company, or, if not, that the generating plant be considered as a separate organization selling power to the smelter at so much per unit. Therefore by electrical installation is meant the cost of transformers, bus bars, cables, instruments, etc. In order to have a basis for computing this cost, it is assumed that the furnace is to smelt 384 tons of charge a day, the composition of the charge being as follows:

Composition of charge.

	Per cent.
Cu	5.21
SiO_2	26.41
FeO	18.60
S	11.46
Al_2O_3	4.26
CaO	17.49

This is the average composition of the entire blast-furnace charge that was smelted at the Washoe smelter, Anaconda, for several months.[a] At the Washoe plant 8.2 per cent of coke was used. The amount of electrical energy that would be required to smelt a charge of this nature in an electric furnace will now be considered.

At the Washoe smelter the furnaces formerly used were 56 by 180 inches in cross section (hearth area 70 square feet) and smelted on an average a little less than 400 tons of charge daily, or 5.6 tons per square foot of the hearth area.[b]

AMOUNT OF ELECTRICAL ENERGY REQUIRED TO REPLACE COKE.

Since 8.2 per cent of coke was required at the Washoe smelter to smelt 1 ton of the above charge, the consumption of coke per ton of charge was 2,000 × 8.2 = 164 pounds.

The coke used contained 80.24 per cent of fixed carbon; therefore, the number of pounds of carbon in 164 pounds of coke is 164 × 80.24 = 131.59. If 1 pound of carbon be completely burned to carbon dioxide, 8,100 pound-calories are obtained. Therefore the number of pound-calories that would be obtained from the 131 pounds of carbon contained in the 164 pounds of coke is 131 × 8,100 = 1,061,100. As 1 kilowatt-hour is equivalent to 1,897 pound-calories, it would require 1,061,100 ÷ 1,897 = 559 kilowatt-hours to replace the theoretical calorific value of the 164 pounds of coke. However, as a copper blast furnace under the most favorable conditions probably does not have an efficiency of over 50 per cent, the amount of energy really obtained from the coke, so far as work performed in the furnace is concerned, is only 530,550 pound-calories, which is theoretically equivalent to $\frac{530,550}{1,897}$, or 280 kilowatt-hours per ton. Inasmuch as the efficiency of the electric furnace of this type may be as much as 85 per cent and as that of an open-top ferrosilicon furnace is said to be about 60 per cent,[a] it is fair to assume that the efficiency of an electric furnace, such as is shown in figure 3, would be 20 per cent greater than that of an ordinary blast furnace, or about 70 per cent as compared with 50 per cent for the blast furnace. If this be true, then the theoretical 280 kilowatt-hours needed to smelt 1 ton of charge would have to be increased 1.42 times (280×1.42=397), or, say, in round numbers, 400 kilowatt-hours would be required to smelt 1 ton of charge. Considering that part of the heat in this case is supplied by combustion of sulphides, the figures agree closely with the 480 kilowatt-hours calculated from the experiments on melting only. As 16 tons of charge per hour are to be smelted, the constant load on the furnace would be 400×16=6,400 kilowatt-hours, and as the power supply would doubtless be three phase, this would mean that the electrical equipment should consist of three 2,500-kilowatt variable voltage transformers. Transformers of this capacity would allow ample margin for sudden overloads, instruments, bus bars, cables, and miscellaneous supplies.

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[a] Peters, E. D., Practice of copper smelting, 1911, p. 264.

[b] Peters, E. D., Principles of copper smelting, 1907, p. 156.

The cost of this equipment at the plant would, of course, depend upon the location of the plant and would be the factory costs plus transportation charges.

As to the total cost of the plant, that would also, of course, depend, as in every instance, upon the location of the plant. In short, as before stated, it would be the cost of an ordinary copper blast-furnace plant plus the cost of the electrical installation. The cost of the furnace itself would be no more than that of a coke blast furnace. As only enough air is blown in to oxidize the iron and sulphur, the blower capacity would not need to be so large as for a regular blast furnace.

COST OF SMELTING.

It may be stated that the cost of coke and the cost of electrical energy are about on a par when coke costs $7 a ton and electrical energy costs $0.15 per kilowatt-hour, or $13 a kilowatt-year; that is, when the ratio of the cost of coke per ton to the cost of power per kilowatt-year is about 1:1.8. Any advantages which there may be in the use of an electric blast furnace instead of an ordinary blast furnace, such as the saving in blowing by reason of not having to furnish air for burning the coke, will be disregarded, and it is assumed that the cost of electrodes and of the electricity on the one hand, and the cost of its equivalent in heating value of coke on the other, would be equal; in other words, that the cost of electrodes and electricity would balance the cost of the amount of coke that would be necessary to smelt the same quantity of ore.

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[a] Conrad, W., and Piels, W., La fabrication du ferrosilicium à haute teneur au four electrique: Revue Mét., vol. 9, 1912, p. 362.

Cost of electrodes.—The average consumption of electrodes in smelting iron ore at Trollhättan, Sweden, is about 10 pounds per ton of iron produced. Assuming that 1 ton of iron is equivalent to 2 tons of iron ore, the consumption of electrodes per ton of charge smelted in a furnace such as the one under discussion would not be over 5 pounds per ton. If it is assumed that the electrodes do not cost over 6 cents per pound, which allows a fair margin for transportation over their cost at the factory, the cost of electrodes would be about 30 cents per ton of charge smelted. The consumption would probably be considerably less than 5 pounds per ton, as in an experiment performed by the writers in which air was blown into the furnace, the electrode consumption was only 2 pounds per ton of ore treated. Such being the case, if the combined electrical energy and electrode cost is not to exceed the equivalent coke cost, with coke at $9 a ton, the cost of electrical energy would be $9 \times 1.8 = \$16.20$ a kilowatt-year, minus the cost of electrodes, or about $16. Compared on the basis of calories, if the coke contains 80.24 per cent fixed carbon, it would contain 1,604.8 pounds of carbon per ton of 2,000 pounds, and this carbon would, if completely burned to CO_2, give a calorific value of about 13,000,000 pound-calories. As one kilowatt-year of electric energy is equivalent to 16,616,000 pound-calories, it would require something like 1.2 tons of coke containing 80.24 per cent fixed carbon to equal in calorific value a kilowatt-year, if it be assumed that all the carbon of the coke is completely burned to CO_2, a condition which rarely occurs in practice. It has been determined from actual practice that from 1.5 to 1.8 tons of coke are required to yield the same number of useful calories as may be obtained from 1 kilowatt-year of electrical energy.

THE USE OF THE ELECTRIC BLAST FURNACE IN THE SMELTING OF PYRITIC ORES.

As was previously stated, in all true pyritic smelting a certain amount of fuel, usually coke, that varies with the nature of the ore is added to the charge. Assume, for the purpose of this discussion, that the ore being treated is entirely suited to pyritic smelting and that it is not necessary to add more than 0.5 per cent of coke to the charge, could such an ore be treated as well in an electric blast furnace as in an ordinary furnace, assuming that the costs would be the same in each case? As what has been stated in regard to the construction of the furnace and the costs of applying the process is as applicable to true pyritic as it is to semipyritic smelting, the discussion will be limited to the metallurgical and chemical features involved.

The nature of the atmosphere in a pyrite furnace will first be considered. As the air enters the tuyères practically all of its oxygen enters into immediate combination with sulphur and iron to form SO_2 and FeO. Although some oxygen escapes combination, the amount is not sufficient to support combustion within the furnace, as has been shown by Sticht and others.[a] So much, then, for the nature of the atmosphere within the furnace.

The changes which take place in the charge as it descends from the charging door to the fusion zone are few and may briefly be stated as follows:

1. The iron sulphide loses one-half of its sulphur as elemental sulphur and becomes FeS.
2. The chalcopyrite loses about one-fourth of its sulphur and becomes practically a mixture of Cu_2S and FeS.
3. The silica is unchanged.
4. The coke is for the most part consumed in the upper zone of the furnace, not by the oxygen of the air, as no free oxygen exists in that part of the shaft, but by the O from SO_2; that is, the carbon of the coke reduces the SO_2, the products of the reaction being S and CO, and this CO reacts with more SO_2, forming S and CO_2. Aside from these reactions, which are of no great importance in the chemistry of the process, the chief office of the coke, as pointed out by Sticht,[b] "is apparently to heat up the sulphides and the quartz in preparation for their active oxidation deeper in the furnace." If, therefore, coke was omitted from the charge, this preparation of the sulphides and the quartz would not take place, and it becomes necessary to determine how detrimental to the process this lack of preparation would be.

In order to answer this question it must be borne in mind that in true pyritic smelting there is "no heat to spare," and that although, as pointed out by Sticht, the heat evolved by the combustion of the coke by the oxygen of sulphur dioxide is only one-third as much as it would be if the oxygen were furnished direct by the blast, and that it is given off at a point considerably higher than the focus of the furnace, it nevertheless "assists in preparing the charge for the reactions which take place in the focus of the furnace, for with 1 per cent of coke in the charge enough heat is generated to supply one-third of the heat required to melt the entire pyrrhotite contents of the charge." This extra heat "doubtless furnishes just the necessary aid to bridge the operation over some critical point." From this it is evident that the office of the coke, then, is solely for the purpose of furnishing heat. Inasmuch as the nature of the charge in true pyritic smelting is such as to demand the oxidation and removal of a large excess of iron as silicate; and as it is necessary to cause the silica present to unite with as much FeO as possible, due to the fact that as a rule silica-bearing materials, which often are barren and hard to obtain, have to be added for slag-forming purposes, and as the saturation point of silica for iron oxide is greater, the higher the temperature attainable in the pyritic furnace; it is quite necessary that the reactions in the melting zone of the furnace proceed as energetically as possible in order to meet these requirements and to provide enough heat to keep the process going. If, therefore, pyritic smelting be attempted in an electric blast furnace it would be necessary to supply the heat from the electric current in such a manner as to insure there being enough heat at the focus to carry on the reactions at that point in the same manner as if coke had been added to the charge. The writers are of the opinion that it would be possible and perfectly feasible to do this in a furnace of the type shown in figure 5. Moreover, the use of electric heating would permit of a more uniform operation of the furnace than is now possible, for with the electric current it would be an easy matter to quickly increase or decrease the heat required for the successful operation of the furnace. In this way there could be maintained the "degree of concentration at a constant" which, as pointed out by Sticht, is very important, but is extremely difficult to do. Also the troubles could be avoided that arise from the constant fluctuations "which are bound to occur and which at present demand unbroken attention and frequent changes in the charge, especially in its proportion of silica, due to the fact that in true pyrite smelting there is no heat to spare," because with the electric blast furnace it would be possible as just stated, to increase or decrease the heat whenever it would be necessary to do so in order to meet these fluctuations.

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 225.

[b] Peters, E. D., loc. cit., p. 227.

CONCLUSION.

The electric smelting of copper ores is nothing more than the substitution of electric heat for the heat derived from the combustion of carbon. Inasmuch as the carbon which is used either in the reverberatory furnace or in the blast furnace plays no important part in the reactions that take place in these furnaces, there is no reason metallurgically why electric heat may not be substituted for the heat derived from the combustion of carbon. In fact, as the authors have attempted to point out, in some cases the reactions would take place to better advantage in the neutral atmosphere of the electric furnace than in the reducing or partly reducing atmosphere of the combustion furnace. Therefore, the practicability of using the electric furnace for the smelting of copper ores would largely depend on the relative cost of coke and electric power. As the use of the electric furnace is not advocated as a competitor of the combustion furnace, but as a substitute for it, in those localities where it is not advisable, because of the high cost of fuel, to use the combustion furnace, there is no apparent reason why the electric furnace may not be developed as a substitute for the combustion furnace where the conditions are such as to warrant its use, especially in the treatment of copper-bearing ores. In this connection it is to be remembered that the development of the electric furnace in the iron industry for the reduction of iron from its ores was due to necessity. As a matter of fact, the field for the electric furnace in the reduction of iron from its ores is a limited one. Perhaps the same is true as regards the possible application of the electric furnace to the treatment of copper ores, but, judging from the comparative costs, as shown in the preceding pages, it would seem that the possibilities of the electric furnace for the treatment of copper ores are greater than those for the treatment of iron ores, because there is not so great a difference between the cost of coke and of electric power in copper-mining districts as there is in iron-smelting centers. Also the cost of electric power is constantly becoming less, through improvements in gas engines and steam turbines, so that in districts where water power is not plentiful but cheap fuels unsuited to coke making are available, it may be found more advantageous to use electric heat than the heat derived from the combustion of coke.

It is sincerely hoped by the writers, although it is impossible at this time to present but few facts, that the comparative study herewith presented may serve to stimulate interest in the subject of the electric smelting of copper ores and to cause others to attempt a further investigation of the subject. As a result of their investigation, the writers are convinced that experimental work on a larger scale would lead to the development of an electric blast furnace which in some cases could be used to better advantage for smelting sulphide ores of copper than the combustion blast furnace.

PUBLICATIONS ON MINERAL TECHNOLOGY.

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Technical Paper 81. The vapor pressure of arsenic trioxide, by H. V. Welch and L. H. Duschak. 1914.

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Bulletin 82

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

INTERNATIONAL CONFERENCE OF MINE-EXPERIMENT STATIONS

PITTSBURGH, PA., U. S. A.
SEPTEMBER 14–21, 1912

COMPILED BY
GEORGE S. RICE

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INTERNATIONAL CONFERENCE OF MINE-EXPERIMENT STATIONS, PITTSBURGH, PA., U. S. A., SEPTEMBER 14 TO 21, 1912.

Compiled by GEORGE S. RICE.

PURPOSE OF CONFERENCE—MINUTES OF SESSIONS.

INTRODUCTION.

In 1911 the Director of the United States Bureau of Mines instructed the chief mining engineer of the bureau, during an investigation of coal-mine safety conditions in Europe, to ask the directors of the various mine-experiment stations whether they would favor a European conference. The majority of the officials in immediate charge of experiment stations favored the plan, and Victor Watteyne, inspector general of Belgium, suggested that the United States call such a conference to be held immediately after the Eighth International Congress of Applied Chemistry, which was to assemble in New York on September 3, 1912.

Accordingly, invitations were issued by the United States Government to those countries having mine-experiment stations, namely, Austria-Hungary, Belgium, France, Germany, and Great Britain, and to those that contemplated the establishment of such stations, Canada and Mexico. All but Great Britain and Canada accepted; Mexico designated a representative, who later found it impossible to come. The Government of Great Britain in declining, expressed regret at its inability to accept; an informal statement was made that the opening of the experimental gallery at Eskmeals, England, occupied the attention of the officials in charge.

Delegates were appointed by their respective Governments as follows:

Austria-Hungary, Julian Czaplinski, royal mining engineer.

Belgium, Victor Watteyne, inspector general of mines.

France, J. Taffanel, director of the Lievin experiment station.

Germany, Carl Beyling, bergassessor, director of Dortmund experiment station.

United States, Charles E. Munroe, consulting explosives expert and George S. Rice, chief mining engineer of the Bureau of Mines.

Subsequently Mr. William O'Connor, a mining engineer and mine operator of Wales, who opportunely appeared at the Pittsburgh station with letters of introduction from the South Wales Institute of Engineers and from prominent mining officials of Great Britain, was unanimously invited to take part in the meeting as an unofficial representative of Great Britain.

Messrs. Beyling, Taffanel, and Watteyne were met in New York on September 3, by George S. Rice, who accompanied them to Washington, where they were welcomed by Director J. A. Holmes, of the Bureau of Mines, and where, with the delegates to the Eighth International Congress of Applied Chemistry, they were presented to President Taft.

Mr. Taffanel was unfortunately called back to France just before the scheduled time of the conference on account of a mine disaster; the other delegates, including Mr. Czaplinski, convened at Pittsburgh on the morning of September 14. Mr. Rice, representing the Director of the Bureau of Mines, acted as temporary chairman, and called the meeting to order.

CHAIRMAN'S OPENING ADDRESS.

The chairman opened the conference with the following remarks:

GENTLEMEN: The director of the United States Bureau of Mines was to have opened this conference. Unfortunately, circumstances delay his arrival, so he has asked me to represent him in this pleasing duty. I therefore, on his behalf and on behalf of the Government, bid you a cordial welcome to this country and to this conference.

It is a source of pleasure to the representatives of this Government that, in spite of the late date at which the invitations were issued through the Department of State, of the six great nations that now have coal-mine testing stations, five, including the United States, have appointed official delegates. The Government of Great Britain replied that it regretted that there was not sufficient time to permit making arrangements to send a delegate, and that the members of the commission are now conducting important experiments at the Government station and are so occupied with that work that they are unable to attend.

It is with great regret that I have to announce that the eminent investigator, M. Taffanel, one of the two official delegates to this conference from France, who has been attending the International Congress of Applied Chemistry, was recalled a few days ago on account of a serious explosion at the Clarence mine, in the Pas de Calais district, France. His absence is a serious loss to this conference, and I am sure that I am stating the views of the various members in expressing sympathy for M. Taffanel in the misfortune of this mining disaster. It will undoubtedly interest the conference to report what M. Taffanel told me recently in Washington that the Clarence mine is very gaseous and subject to spontaneous fires—a most serious combination, as all mining men know. I have personally observed the excellent preventive means against dust explosions taken in the French mines as a result of M. Taffanel's investigations at the Liévin station. M. Weiss, the other delegate from France, has not yet arrived, I believe, but we hope that he will be with us at the subsequent sessions.

The plan of a cooperative arrangement between the representatives of the different testing stations was discussed by the mine-accident committee of this bureau, and last summer when I was sent to Europe with a party of engineers to visit European coal mines and testing stations, the director suggested that I make inquiry informally of the directors of the several stations as to whether they would approve of the plan of holding a conference. All those whom I saw expressed themselves as being pleased with the idea. M. Watteyne, the delegate from Belgium, suggested that in view of the international congress of applied chemistry meeting in America this summer it would be an excellent idea for the United States to invite the various countries to send representatives who might attend this congress, and then have a special conference on the mine-testing work. On my return the director expressed pleasure at the suggestion, and invitations would then have been issued had there not been uncertainty about the action of Congress regarding appropriations for the year's work of the bureau. Owing to delay in making these appropriations the invitations could not be sent earlier than they were.

Besides the official delegates who are present, we are fortunate in having with us a number of visitors from abroad as well as from this country.

As a result of a preliminary conference with the official delegates, I am pleased to announce that the visitors present may attend all those meetings, tests, and demonstrations indicated on the program, except the business meeting on Thursday evening and possibly certain other business sessions.

I now take pleasure in introducing Mr. Herbert M. Wilson, engineer in charge of the Pittsburgh testing station, who will give you a few words of greeting from the local staff of the bureau.

ADDRESS BY H. M. WILSON.

Mr. Wilson spoke as follows:

Mr. Chairman and Gentlemen: The engineers of the United States Bureau of Mines, with headquarters in Pittsburgh, greet you and are delighted to have you with them. Not only we, but the people of Pittsburgh, and especially the mining people centered in Pittsburgh, feel greatly honored that an official international conference having to do with such important subjects as you are to consider should meet in their city. Pittsburgh is very properly chosen for the convening of the first international conference to consider matters that may improve the conditions of safety and of efficiency in the mining of coal, because it is one of the greatest coal-mining centers in the world, representing a production of 200,000,000 tons annually, and with a mining pay roll of over 300,000 men.

The program of the conference is before you, and I am sure that this country, as well as your own countries, look forward with great expectations and hope of the good that will ultimately come of the proceedings of this body.

ADDRESS BY CHARLES E. MUNROE.

The temporary chairman then introduced Prof. Charles E. Munroe with these words:

I take pleasure in introducing Prof. Charles E. Munroe, of Washington, D. C., who is consulting explosives chemist of the bureau, and whose name is well known to many of you, who will extend some words of greeting on behalf of the Washington contingent of the bureau, and also for the explosives chemists.

Prof. Munroe spoke as follows:

I am very glad to add my voice to the expressions that have been made to you. I had no intimation that I was to bear a part in this manner, but what has been said has been said so admirably that it leaves me little to say. I may perhaps call attention to the fact that our country has been a little laggard in these investigations, and that you gentlemen in Germany, France, Belgium, and England were years in advance of us in investigating the problems of explosions in mines and the methods by which they might be reduced in number and might be mitigated or suppressed.

Our peculiar governmental organization prevented our sooner undertaking the solution of these problems, though they were before the minds of many and had been brought forward publicly to attention. This peculiar governmental organization requires a special kind of individual to solve its Gordian knots, and it was not until the present director of the Bureau of Mines appeared, who has a gift for cutting knots and expressing a purpose, that we were able to do what we have done. I think that in the time this bureau has been organized a reasonable amount of work has been accomplished—all I think that might have been expected of it in this time.

I am speaking in this way because my attention has been limited; I am a viewer rather than an actor; I have seen what others are doing; I have noticed what they have initiated. I remember that at the outset of the organization of this station we had the benefit of your advice, your comment, and your criticism, and of your long experience, all of which have proved so valuable to us in the work we have done.[a]

We have now gained some little experience, in that we have seen some of the problems that are to be solved; also we have seen many great difficulties that must be overcome.

ADDRESS BY MR. WATTEYNE.

Mr. Victor Watteyne, of Belgium, was introduced by the chairman and made the following remarks:

First, I congratulate the Bureau of Mines for the great work it has done so rapidly. Only four years ago you were just beginning; now you have done a great many things. I congratulate Dr. Holmes, who is the promoter of all this, and I congratulate also the Bureau of Mines, which has practically initiated the idea of uniting the work of the testing stations.

The questions under investigation are not new. The testing station at Frameries was established many years ago; and at Berlin nine years ago it was suggested by an English delegate to conduct the testing station at Frameries as an international testing station, and to appoint a committee that would decide what experiments should be made. Belgium was not the first to have a testing station; Germany and Austria already had testing stations. Belgium, being a neutral country, was considered eligible, and had been suggested as the place for an international testing station, but the idea was not practicable. It was impossible to have an international committee coming from all parts of the world working at Frameries and indicating the program of the experiments to be made. However, the idea was presented again at the congress at Liège in 1905, but it was rejected also.

Now, Dr. Holmes proposes that the five or six testing stations in the world unite their work. The question of unification has already been spoken of in a preliminary meeting, and we quite agree. We, meaning Dr. Holmes, Mr. Beyling, Mr. Taffanel, and myself, had a meeting in New York before coming here, as Mr. Taffanel had to return to France. We considered the leading idea that ought to prevail in our conference. This must not be a question of indicating to each testing station, generally speaking, the way in which it must work, for such a course would prevent the development of initiative at the different testing stations. The initiative and the desire to do better than another must be preserved because they make for the good of humanity, for the good of the problem with which we are dealing hand in hand. Some of the results obtained at the different testing stations—Pittsburgh, Gelsenkirchen, Frameries, Liévin, and Altofts—are rather contradictory. There must be a reason for that, and seeking the reason will be one of the purposes of the international studies. Of course, if the same question is solved differently in different countries the solution is certainly not exact in several points; if we get different results it is a proof certainly that we at first made mistakes—perhaps all of us. It is very important to make no important error. For example, it would be a deadly mistake if we accept an explosive as safe when it is not; we might cause the death of some workmen by such a mistake.

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[a] See Technical Paper 21, Bureau of Mines: The prevention of mine explosions, by Victor Watteyne, Carl Meissner, and Arthur Desborough.

It is very important to study such questions with care. At this moment I can not enter into details of what might be done, but you understand it is one of the points we have to consider together, one of the points that would make the conference of great and positive utility. Another point—we may have some experiments that we will study together, but such as are not susceptible of being tried in all the testing stations. For instance, in Frameries, if we wish to have a large explosion like one tried in the gallery of another country and know what has occurred in that explosion and what are the means of preventing its propagation, I could not experiment with exactly the same conditions because my testing station is not made or arranged for that; but I could have a test of a similar kind. Such a matter is, therefore, one of our problems to be considered at this conference. This conference should, therefore, prove of great value to all nations. I congratulate again the Bureau of Mines, and we all feel very grateful for the opportunities presented through its invitation.

REMARKS BY MR. BEYLING.

Mr. Beyling, of Dortmund, Germany, was introduced. He asked to be permitted to speak in German. His remarks as translated were as follows:

We appreciate very much the calling together of the international conference for the purpose of trying to settle some mining problems, and, as Mr. Watteyne has stated, there are very difficult problems, and we can do only our best. We have nothing to add to what Mr. Watteyne has stated and his general suggestions as to the conduct of the conference.

ADDRESS BY DR. CZAPLINSKI.

Dr. Czaplinski, of Austria-Hungary was then introduced, and also addressed the conference in German. A translation of his address follows:

The Austrian Government accepted very gladly the invitation to attend this conference, because of the disagreement that has arisen heretofore between the various mining testing stations, and it is high time that we come together and try to unify our methods.

In Austria we deal mainly with three problems. The first problem is that of mine gases, the handling of which we solved in Austria during 1890 to 1898; we have advanced so far in devising and applying safety measures that we have had no accidents of any importance from this cause during the past 15 years in Austria, although some of the coal mines of Austria give off the greatest volumes of methane of all the mines in the world.

The second problem is that of explosives. Without explosives, of course, little work can be done in mines. The problem is, therefore, to procure explosives that, within certain limits, are safe to ignite in the presence of gas or coal dust. This problem we have not as yet solved.[a] We have two testing stations now and a third is now being erected in Bohemia; these will act together in investigating the question of safe explosives.

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[a] In Austria, blasting with dynamite is prohibited in mining coal. For this work safety explosives have to be used, one of which is mainly an ammonia-saltpeter compound (Gasdynammon), the other an ammonia-gelatin compound (Pannonit).

The third problem is that of coal dust. Because of the disaster at Courrières this problem has been called to our attention more forcibly.

In my district near Segen-Gottes, in the mines of the Russitz Mining Co., I have erected a large station for the experiments on coal dust; I have also published a report that is probably familiar to you. We have found means of preventing coal-dust explosions, and have been working along the same lines as the experimental station at Liévin. I am corresponding with Mr. Taffanel and exchanging my publications with him, and I am very sorry he is not present. Lately Mr. Beyling's experiments on coal dust simulated my own. I have also investigated the results in England, which differ greatly from ours.

I welcome, therefore, the international conference because its object will be to reach an agreement. In Austria we have three stations, and we have established a basis so they will be able to work together along parallel lines. I will take pleasure later in describing the basis for the work of the three stations.

ADDRESS BY WILLIAM O'CONNOR.

The chairman then introduced Mr. William O'Connor, of South Wales, who had been invited to attend the meetings as an unofficial delegate. Mr. O'Connor spoke as follows:

I am scarcely prepared to take part in the discussion. I might say that in England experiments have been carried on by our mine officials at Altofts under the direction of Mr. W. E. Garforth, and the results have been considered so important that after a little agitation the Government has taken up the question of erecting a special testing station. One of the difficulties in experimenting on the scale on which the experiments were carried on at Altofts was that for some miles around the explosion broke all the windows, frightened a great many people, and had a very unpleasant effect on those in the vicinity, and I think it was particularly on that account that the new location was decided upon.

The new station will be erected in close proximity to Messrs. Vickers' testing works. Any explosion effects will be little noticed because of the large number of explosion tests which are carried on by Vickers without difficulty, owing to the isolation of the site. There are no habitations within a considerable distance, so that no one will be affected by them. I do not think that any experiments have been conducted on the new site, but I think the gallery will be about 800 feet long and about 7½ feet in diameter. I have not the exact dimensions with me, and must give them from memory. In addition there will be smaller galleries. In the main gallery it is hoped to carry experiments a step farther than at Altofts. I may say that the results attained at Altofts have convinced all of the serious dangers from coal dust.

One of our most serious dangers in regard to coal dust is that of ignition, and we consider as our two principal sources of ignition the igniting of small quantities of fire damp and the use of explosives. We have a number of instances in which ignition took place in mines long abandoned, but as there was no loss of life we dismissed this phase of the matter as not very important. In regard to the ignition of small quantities of fire damp in such a way as to create explosions, a new act came in force in July which lays down very stringent regulations as to the percentage of gas that may be allowed in mine air. In some mines the use of explosives is prohibited altogether. About half of the coal produced is now brought down without a single shot. In regard to palliative measures relative to coal dust, under the new act there is a special officer appointed to see that such measures are taken. We hope that they will have some good effect on the death rate caused by coal-dust explosions.

FORMAL OPENING OF FIRST MEETING.

At the conclusion of the remarks of Mr. O'Connor, the temporary chairman stated that it was understood at the preliminary conference that the chairman of each meeting should be one of the foreign delegates, and that each would serve in turn in the alphabetical order of their respective countries, but that for the first day's meeting it had been informally suggested by some of the foreign delegates that, as they were not quite familiar with the situation, it might be best to call upon Prof. Munroe to preside as the chairman. This suggestion was formally adopted by the conference, and Prof. Munroe took the chair. He expressed his pleasure at the honor of presiding and then stated that it appeared advisable to elect a secretary who should remain in office throughout the conference and keep the records of the meetings. The conference thereupon elected Mr. Rice as secretary.

The chairman (Prof. Munroe) announced that a series of papers prepared by members of the staff of the Bureau of Mines, presenting various matters for the consideration of the conference would, with the consent of the delegates, be read. Accordingly, he called upon Mr. Rice to present his paper "on The Methods Employed by the Bureau of Mines in Investigating Coal Dust or Gas."

After the paper had been read, the meeting adjourned to witness a coal-dust explosion test at the experimental mine near Bruceton. The explosion or inflammation, the details of which (No.17) are given on pages 44 and 45, was small, the flame traveling outward only 100 feet and inward 50 feet.

The smallness of the explosion was a surprise, as two days previously, in an experiment with similar loading and other arrangements, the explosion had been strong, flame extending 175 feet outside the opening, as described on page 44. The only apparent difference in the conditions of the two tests was that the relative humidity of the outside air was 46 per cent on September 12, the day of test 16, whereas it was 78 per cent (following a rain) on September 14. This difference may have caused the walls of the mine entry to be wetter on the latter date.

MINUTES OF MEETING OF MONDAY, SEPTEMBER 16.

On Monday, September 16, 1912, the meeting of the conference was called to order at 9.30 a. m., by Mr. Victor Watteyne, of Belgium. The minutes of the meeting follow:

Mr. Watteyne. I think that by reason of the efforts of the Bureau of Mines the mining-fatality rate in a few years will be very much lowered. It seems to me that in the papers presented at the last meeting we had the causes of the many explosions in coal mines clearly indicated. It must be added that the operators in the coal mines of America are obliged by the stress of circumstances to produce very much and very rapidly, because they are obliged to produce very cheaply, and this is indirectly the cause of the bad conditions, but I think there is nothing that we can do to prevent this. We must take that into account and direct the struggle, not against the conditions of the coal business, but against the consequences that result, for example, from the vast and seemingly unavoidable use of explosives in this country. What must be done is to have the explosives used with greater safety. We can not make them entirely safe, but we can make their use much safer if they are of good quality, that is, the permissible explosives that the Bureau of Mines has tested and has already introduced extensively in the coal mines of America. The precautions that are to be taken in using these explosives are being considered by the Bureau of Mines, and I know that the special engineers of the bureau are making the operators acquainted with the best ways of using explosives. I think it is one of the best things you can do for the prevention of mine explosions.

As to the question of "shooting off the solid," I formerly deemed that the importance attributed to it in America was somewhat exaggerated, and I was afraid that the operators might be induced to consider as dangerous the blown-out shots only, and to take little care of the others. Since, we have made at Frameries many experiments, which prove that the blown-out shots are indisputably much more dangerous than the others. But I repeat that all kinds of shots must be reported dangerous, and precautions ought to be taken against them, especially by using exclusively safety explosives, the "bourrage extérieur" ("outside stemming"), and the like.

Another problem that has been dealt with in the previous papers is that of humidifying the mine. I say for my colleagues of Europe that this problem presents itself more acutely in America than in Europe. Generally speaking, it is much more easy and much more profitable in America than in Europe to spray or dampen the ventilating air current, because the mines, as Mr. Rice states, are shallow, the roof strata are generally good, and the conditions are favorable for humidifying the air. It is evidently a good precaution against the dangers of coal dust, which are very great here. In European coal mines, especially in Belgium, the seams are very thin and many of the operations are in rock. This difference may explain why American mines have a tendency to have more coal dust than European mines.

Consideration was next given to the instruments used in coal-dust experiment galleries, the meeting adjourning to Dr. Clement's laboratory.

Dr. CLEMENT. We have been experimenting with the manometer for recording the pressure of coal-dust explosions in the mine. Dr. W. O. Snelling, formerly connected with the bureau, suggested that we could get rid of the inertia of an ordinary gas-engine indicator by using some kind of material whose electrical resistance would change with the pressure. So we have tried a great many different materials—metals, carbonized cloth, and carbon disks of various kinds—and we finally decided on a carbon disk. You, of course, know that if you apply pressure to a pile of carbon disks the electrical resistance changes. Now, if you can find out and define the relation between resistance and pressure, then if you measure the variations of the resistance, in that way you can determine the pressures. We have not perfected this machine yet; we are still making some experiments with the oscillograph to measure the pressures.

Demonstrations and explanations of the apparatus (described in Dr. Clement's paper, p. 41) were then given in the physics laboratory of the experiment station.

At this point the chairman read a letter from the Secretary of State for the Home Office of Great Britain, stating that Sir Edward Grey regretted that His Majesty's Government found it impossible to be represented at the conference.

The conference next adjourned to the gas laboratory, where Mr. Burrell demonstrated and explained methods and apparatus for gas analysis.

The conference returned to the library, where a demonstration was made of a new instrument designed by members of the bureau for the purpose of taking automatically three gas samples in rapid succession.

Dr. Clement then described a number of curves showing the results obtained with laboratory apparatus for testing the inflammability of coal dust.

The conference adjourned for lunch, later going to the experimental mine, near Bruceton, to witness a second coal-dust experiment (No. 18, described on p. 45). In this test ignition was made at the face of the right-hand stub to determine whether an explosion would go inby the main entry, as well as outby. The explosion did go inby, although the explosion was not violent.

EXCURSION OF SEPTEMBER 17.

On Tuesday, September 17, 1912, the members of the conference took part in a Monongahela River excursion with the delegates to the International Congress of Applied Chemistry, visiting the Clairton plant of the Carnegie Steel Works, and in the evening attended an entertainment given in honor of the congress.

MINUTES OF MEETING OF WEDNESDAY, SEPTEMBER 18.

The meeting of Wednesday, September 18, was called to order by Mr. Carl Beyling, of Germany. The chairman presented for discussion the subject of explosives.

Dr. Czaplinski. The results in Austria in the testing of explosives have varied, owing to lack of uniformity in methods, and for this reason we have arrived at a common basis for investigations at all of our stations. Explosives have been investigated most extensively by the army department. My investigations have been in the development of so-called "safety explosives," as they are the only ones allowed in our mines. Such explosives are safe only up to a certain point, and the main object in testing them is to determine the proper safety limit. We test our explosives under the most dangerous conditions; that is, under such conditions as are most favorable to ignition of gas and coal dust.

Second, we test our explosives under exactly the same conditions, so as to have a uniform basis and thus enable us to compare the safety of the various explosives.

Our method differs entirely from the method used in England, France, or Belgium. In England, for instance, a small charge is used in a cannon with tamping; in France the ignition temperature is determined; in Belgium the charge is fired in a cannon without tamping; in Austria we fire the explosive without stemming, the effect being that of a suspended shot, because in this way we get identical conditions for all our tests. The explosives rest freely on steel plates and stand vertically. On the steel plate we place two lead cylinders or blocks on which the explosive is placed in a parchment wrapper. The diameter of each explosive is 40 mm. The explosive is fired by an electric detonator. In the test the charge starts with 50 grams, being increased by 50 grams in each successive test up to the limit. The cap is either 1 or 2 grams, according to the explosive. The lead blocks show whether the entire explosive detonates. The tests are the same for all explosives. We formerly used a cannon, but as the size of the bore hole changes after each shot, we did not have the same conditions in all tests. Neither does a cannon simulate the real conditions in a mine. In a mine the shot is made in the coal, and it has to perform some work, whereas in a cannon it does not perform any work. In the explosion chamber we use either gas or coal dust, or both. Through these shots we determine the safety limits of the explosives.

We also test each explosive in bomb cylinders (Bichel gages) according to the method adopted in Berlin. Each explosive must show certain results; otherwise it can not be used in a mine.

Third, this method of testing on the lead blocks shows whether there is complete detonation. Those explosives showing incomplete detonation, by leaving a residue, are excluded from further tests because they are dangerous.

We also measure the detonation temperature. An explosive that gives a temperature of over 1,500° C. is excluded.

A hole is made in the ground and a cylinder made of sheet iron is placed in it. The cylinder rests on two lead blocks, and the latter rest on a steel plate. The whole thing is surrounded by clay. We put 200 grams of the explosive in the hollow sheet-iron cylinder; a cap is placed on the top. We discharge the shot and photograph the flame. The flame must be short and dark. If the flame is too long the explosive is excluded.

After the explosive has stood all of these tests, it is tested on a small scale in the mine under strict supervision, and the gases resulting from an explosion are analyzed. If the explosive stands the small-scale test, then it is applied on a large scale, and the combustion gases are analyzed, and if it succeeds on a large scale it is declared a safe explosive. It must not produce any monoxide. The manufacturer is obliged to make his explosives exactly the same as the one tested. If he furnishes more than one kind of explosive each kind is tested in exactly the same manner.

Mr. BEYLING. The main difference between the tests in Austria and those in other countries is that in Austria the charge stands upright, and in other countries it is shot in a cannon. I myself do not think that the Austrian method is the proper one. First of all, the conditions for ignition are not as favorable as when the charge is confined. In the second place, with an unconfined charge it is impossible to ignite coal dust. I do not think that the test of coal dust can be carried out with a free charge standing upright and unconfined. Third, I consider the conditions in a cannon as more in correspondence with the conditions in our coal mines.

Mr. Czaplinski has stated that in a cannon the explosive does not perform any work; we know that the most dangerous shots are exactly those that do not perform any work in the coal, that is, the blown-out shots, exactly like those in the cannon.

The objection of Mr. Czaplinski as to increase in size of the bore hole I do not think so important. Of course, if it has changed its shape entirely it must be used no longer. There is another point in favor of shooting from a cannon. We do not know yet what causes the explosion, whether it is the flame, or the after damp, or the duration of the detonation, but I do not doubt that the composition of the detonation gases has considerable influence on the result—whether the explosive will cause an explosion or not. Naturally these detonation gases will differ according as to whether the charge detonates when it is free or when in a cannon. It is probable that we will obtain the same gases if fired from a cannon as when fired in bore holes in a mine. I would consider it desirable to reach some uniform method of testing explosives.

Mr. Watteyne. In Frameries the methods of mining, the charge of explosives, and the relative safety of explosives are almost identically the same as in Germany, and the reasons put forth by Mr. Beyling in order to justify his methods are the ones that guided us in Frameries. I thoroughly agree with the views expressed by Mr. Beyling. The importance of blown-out shots is very great; the difference in the safety of explosives when doing work and when they do not do any work is exceedingly great. It is an additional reason for preserving the method of using a rigid cannon, in which no work is done by the explosive. The cannon gives rather severe conditions. On studying at Frameries the "bourrage extérieur," that is, an accumulation of stone dust at the mouth of the shot hole, we have been led to make some experiments under quite the same conditions as are found in coal mines and, for that purpose we have begun an underground gallery in a forest (Bois de Colfontaine) not far from Frameries. That forest is over the coal measures, so that in tunneling we have the same rocks as encountered in coal mining. The gallery is a rock drift. It is to be noted that, unlike in America, Austria, and often in Germany, where they shoot principally in coal, in Belgium the underground galleries are frequently in rock. Shooting in coal is not permitted.

At this point the discussion was adjourned for lunch. Following lunch the members of the conference again visited the experimental mine at Bruceton to see the third experiment (test 19; see p. 45). The igniting charge in a cannon was at the face of the main entry. Only a limited inflammation occurred, owing probably to the damp condition of the mine. The relative humidity of the mine air was 95 to 97 per cent, there were pools of water standing in a side ditch along the entry, and the walls were damp.

MINUTES OF MEETING OF THURSDAY, SEPTEMBER 19.

At the meeting of Thursday, September 19, Dr. Julian Czaplinski, of Austria-Hungary, was in the chair and announced a continuance of the discussion on explosives, which is recorded below.

Mr. Clarence Hall. Before reading my paper I want to say a few words relative to testing explosives in a cannon and relative to suspended shots. It is my opinion that shots fired from a cannon more nearly approach the natural conditions than when tests are made with the explosive suspended. When testing explosives in a cannon we experience no difficulty in getting proper results provided we adjust the cannon after it has been used a certain number of shots. When the drill hole reaches 3 inches in diameter we put in asbestos rope or fire clay and bring the charge forward. We find that the length of the bore hole is not a factor in determining the safety of explosives. In other words, if the bore hole of the cannon is 21 inches long or 17 inches long it does not make any difference in the results. When the shots are fired without stemming, the rate of detonation is apt to be less than when confined.

Mr. Hall then presented his paper (see pp. 81–92). He also showed a number of charts giving results of tests of certain unnamed foreign explosives, the composition being given.

Mr. Watteyne. The reason we have selected coal dust with about 22 to 25 per cent volatile matter for our standard in testing explosives is that the mines that contain such coal dust are the most dangerous, being the most dusty in Belgium. The dust made from coal with higher percentages of volatile matter is indeed more inflammable but, generally speaking, the mines where such coals are mined are, in Belgium, much less dusty and, therefore, the danger of coal-dust explosions is not very great.

Mr. O'CONNOR. We have found that where the volatile matter is lower, the coal dust is more explosive than with high volatile matter.

Dr. BEYLING. The same conditions have been found in Germany. Although, in general, coal dust must be more dangerous the more gas it contains, our so-called "gas coal," which contains 50 per cent of gas, is hard, and does not yield much dust; moreover, the gas of the gas coals requires a higher temperature for liberation than the gas of a "fat" coal. The "fat" coal has only 20 to 30 per cent of gas, but the gas escapes at a lower temperature, and at 100° to 200° C. we can get much of this gas out. For this reason coal-dust explosions have taken place in the mines producing such lower volatile coals.

Dr. CZAPLINSKI. We find the same conditions in Austria. The explosions depend upon the gas content, and have more often occurred in the same kind of coal, containing 26 per cent of volatile matter. In the case of fresh coal there is more liability of explosions than when the coal has been exposed several days, as it then loses a great deal of its volatile content. We also found that ammonium nitrate explosives must be used when in a fresh condition, as they readily absorb water, and either detonate partly or not at all. We always have to protect them against moisture. We also found that the density of the explosive is of great importance. The ammonium nitrates used must have a specific gravity of 0.8 to 0.9. If used with density of 1 there is no detonation. By changing the density we change the explosibility. I recognize the importance of making experiments with the cannon, and will try to make another series of experiments with the cannon. I have not, however, been convinced by the objections brought against the methods of using free shots. I have always found that free shots cause coal dust to explode.

Mr. HALL. It is proposed that the United States Bureau of Mines shall have a new requirement limiting the age of an explosive, as it becomes insensitive after, say, 4 months, when it should be taken off the "permissible list." I would like to ask the views of the conference on this matter.

Dr. CZAPLINSKI. We have experienced no such difficulties in Austria for the reason that we do not have to transport explosives such long distances from the factory to the mines as you have to in the United States.

Mr. O'CONNOR. Whenever we have the least cause to suspect anything not up to the standard in any explosives in Great Britain, we prohibit their use altogether and send them back to the manufacturer. We find a great deal of difference in the time an explosive will remain unaltered in strength, according to the time of year.

Dr. BEYLING. The question raised by Mr. Hall relates to the desirability of establishing a test for explosives so as to find out after a certain length of time whether they will still detonate. Then if they do not come up to this requirement they are to be taken off the permissible list. We have no such requirement. It is very difficult to arrive at conclusions in this respect. The first consideration is the conditions under which the explosives change, and when they do not detonate properly. In Germany the explosive must be stored in such a way that it will not absorb moisture. The factories are in the neighborhood of the mines, and require less transportation than in this country.

Dr. CZAPLINSKI. I agree with Dr. Beyling in every respect. The same conditions prevail in Austria as in Germany. We have very strict regulations as to the packing of explosives. Explosives that absorb moisture must be packed four times, and the packing soaked in liquid paraffin or wax, and if stored underground they must be stored in a dry magazine.

Mr. HALL. May I ask the views of the conference regarding restrictions in the production of carbon monoxide from explosives?

Dr. Czaplinski. The restriction in our country is that the explosive on detonation must not produce any carbon monoxide gas; otherwise it is rejected.

Mr. Rice. Is the determination of CO made from samples gathered from the detonation of the explosive in a Bichel gage, or from samples gathered in the mine after a shot has been fired?

Dr. Czaplinski. In a mine.

Dr. Beyling. There are no restrictions as to the production of monoxide in Germany. The miners must not go back to their places for at least 10 minutes after a shot has been fired. We have tried to determine the CO in the afterdamp. We have never found large quantities, although we know that we produce large quantities at times. It becomes so quickly mixed with the air that we can not determine the amount.

Mr. Watteyne. In Belgium there is no special regulation about the gas or products of explosives. We have had no special inconvenience along this line. Our mines are well ventilated, especially at the points where explosives are used, though we have more narrow places on account of the thin seams than in other countries.

Mr. O'Connor. We have found carbon monoxide produced by the use of carbonite. It has a cumulative effect on men, and therefore the use of carbonite is to some extent restricted. We have no regulations regarding production of gases from explosives, but we do not use carbonite in places where there is any danger of affecting the health of men.

Mr. Rice. The production of deleterious gases is a matter of great importance in America on account of the almost universal use of the room-and-pillar method of mining. Very often the ventilation up to the face of a room or heading is not perfect. The faces of rooms or entries may be 100 feet beyond where the air current is moving, and the miners tend to go back too quickly to the face.

Dr. Czaplinski. The conditions in Austria are the same as Beyling has described in Germany. Several shots are fired at the same time. The men do not return for 5 or 10 minutes. We have never had any trouble from poisonous gases. The ventilation is good in the working places.

Dr. Beyling. If you fire a shot in the coal you may get carbon monoxide which does not arise from the explosive, but from the great heat to which the coal is exposed. Twenty years ago before we had any safety explosives we used dynamite in the coal. This explosive does not give off any carbon monoxide, still we had many cases of CO poisoning, which always came from the partial combustion of the coal.

Mr. Hall. We have found that carbon monoxide is produced from the coal, when a shot is in coal, owing to the heat of the explosive.

Mr. O'Connor. I would like to propose that the detonators used in all countries should be standardized, and that a number should be placed on each indicating the strength. We always investigate the caps, placing them on lead plates 4 to 6 inches square for testing.

Dr. Czaplinski. Our blasting caps are all numbered. The numbering is based on the rapidity of breaking through lead testing plates of different thicknesses.

Dr. Beyling. I am in favor of having a standard adopted in all countries. Have you any standard to propose?

Mr. O'Connor. I would propose that the detonators in all countries be numbered and that the numbers in each country represent an equal strength of detonator.

Dr. Czaplinski. How shall the strength be determined?

Mr. O'Connor. It would be necessary to test them on one standard. I take it that a detonator that perforates a certain thickness of plate should have the same number in all countries. The standard to be in millimeters.

Dr. Beyling. In Germany we number the caps from 1 to 10, representing definite efficiency.

Mr. Hall then explained the system in use in the United States, speaking as follows:

The bureau is now conducting an investigation to determine the relative strength of detonators and electric detonators of different compositions. The direct method of determination comprises tests to ascertain the mechanical effect of the detonators; the indirect method comprises tests to ascertain the mechanical effect of commercial explosives with which the detonators are used. The direct method, which includes the tests on lead and iron plates and on small lead blocks, and other similar tests, offers the advantage of simplicity, but it is not believed that the results of these tests show the relative efficiency of detonators. A new test, known as the nail test, which was recently devised at the Pittsburgh experiment station, has been found to give satisfactory results in determining the relative strength of detonators. This test depends on the angle formed by a nail when a detonator is fired in close proximity to it. The results of the nail test approximate more closely those obtained by the indirect test than do any of the methods previously used for the direct determination of the relative strength of detonators.

The indirect method for determining the mechanical effects of explosives, that is, the energy developed by the explosives, approximates practical conditions, and, it is believed, offers an accurate means for determining the relative efficiency of detonators. The results of tests show that when high explosives that are not too sensitive are used, the energy developed increases with the grade of the detonator. For example, the average efficiency of four different explosives was increased 10.4 per cent when a No. 6 detonator was used instead of a No. 4. Furthermore, the average percentage of failures of explosives to detonate was increased more than 20 per cent by using lower grades of detonators instead of No. 6. It is noteworthy, however, that when sensitive explosives, such as the higher grades of nitroglycerin dynamites, were tested under conditions ideal for detonation, the same energy was developed irrespective of the detonator used.

PERMANENT ORGANIZATION.

At this point the chairman announced that the technical discussion would be interrupted to allow consideration of plans for a permanent organization.

Mr. WATTEYNE. At an informal meeting in New York, with Director Holmes and Messrs. Taffanel and Beyling, we agreed on certain features that should be adopted in a permanent organization. I believe that the proposition should be made by the representatives of the United States, through the proper diplomatic channels.

Mr. RICE. It is suggested that the organization be made permanent; that meetings be held at certain intervals; that between the time of meetings communication be maintained between the representatives of the different testing stations in order to obtain prompt information as to the nature and results of tests that are being carried on; that so far as possible there shall be a uniformity or standardization of tests. I do not mean that all shall be alike, but that some method of standardization be agreed upon; that we shall interchange samples of explosives, coal dust, and other material that enter into the tests, so that we can try them in different kinds of apparatus, and thus be able to compare the results obtained at the different stations. It would be desirable that we select as chairman a representative from the country in which the next meeting will be held, and that the secretary be selected from the same country, in order to expedite the arrangements.

Mr. WATTEYNE. It seems to me that the Bureau of Mines should send officially to each government an outline of the project; this would be referred through the official channels to the testing stations.

Dr. Czaplinski. I am afraid the use of diplomatic channels would take too long a time, perhaps ten years.

Mr. Watteyne. It should not take ten weeks. The representatives must be appointed by their respective governments. It is a question of form, but that form must be insisted on. The purpose of the committee must not be exaggerated. If it is exaggerated, you will find many objections. It must be understood that each testing station has to preserve its entire initiative and liberty of working for the best. Some points that have been investigated by the different countries are still in question and ought to be decided for the good of humanity. Certain testing stations are better equipped to make certain tests, which others could not make at all, and such tests will be arranged by the permanent committee.

Dr. Beyling. I agree absolutely with Mr. Watteyne that the United States Government or the Bureau of Mines should write out the project and submit it to the other governments. I myself do not have the authority to decide matters for our Government. I can not say whether they want to participate in such an arrangement, but hope they do. I am convinced that they will. It must be done in an official way.

Mr. Watteyne. I do not understand that the requirements are that the different governments should be represented by only one person. There might be several. Each testing station might desire to be represented.

Dr. Czaplinski. I was appointed by the department of public works as representative of the experiment station of Austria, and can not take any responsibility as to what the Government might desire. We have two stations and the third is being prepared. The two are private, and the new one will be governmental. The one in my district is a private station. I can not tell what the Government will do, but will try to influence it to adopt the requirements that we shall recommend, and try to see that the requirements are imposed on all the stations. I agree entirely with Mr. Watteyne's proposal that this project should come in an official way, addressed to the Government.

Mr. Watteyne. I think we should agree as to the details, but that this should not be communicated to the Governments, because it might cause complications. We should give them merely the principal objects of the organization.

Mr. O'Connor,[a] after giving a historical review of the stations of Great Britain, including the establishment of an official station, at Eskmeals, stated that he thought the project a most desirable thing, and thought that the British Government would be perfectly willing to participate.

The meeting then adjourned for luncheon, with the staff of the bureau.

DEMONSTRATION OF MINING EQUIPMENT.

After luncheon, James W. Paul, mining engineer of the bureau, in charge of the bureau's mine-rescue operations, presented a paper on breathing and other rescue apparatus (see pp. 61–64), and gave an exhibition of rescue apparatus, safety lamps, etc., during which there was an informal discussion on the points brought forth by the exhibit.

H. H. Clark, electrical engineer of the bureau, exhibited various pieces of electrical apparatus, and demonstrated certain tests. He had prepared papers, which are presented on pages 68 to 75, but were not read as time was not available.

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[a] Although Mr. O'Connor was not the official representative of Great Britain, it was unanimously agreed that he should remain in the private session to assist in perfecting the plans for organization.

Clarence Hall, explosives engineer of the bureau, exhibited some of the apparatus used for testing explosives. During the demonstration further informal discussion took place.

The meeting then adjourned.

FOREIGN GUESTS AT BUREAU'S ANNUAL DINNER.

That evening (September 19) the foreign delegates to the conference attended, as guests of honor, the annual dinner of the staff of the Pittsburgh station of the Bureau of Mines, at which the director of the Bureau, Dr. J. A. Holmes, and the assistant director, Van. H. Manning, were present.

MINUTES OF MEETING OF FRIDAY, SEPTEMBER 20.

Mr. William O'Connor, as chairman, called to order the meeting of Friday, September 20. The minutes are presented below:

Mr. RICE. As this will be the last formal session, I think we had better first continue the discussion of permanent organization in order to be sure to arrange the details. As my colleague, Prof. Charles E. Munroe, has had to return to Washington, leaving me the only official representative of this country, I feel the need of instructions from the conference so that I may be able to report the matter to administrative officials of my Government.

Dr. CZAPLINSKI. I propose that we make two separate matters of it. I propose, first, that each experimental station, whether private or governmental, shall send representatives to the meetings. In addition to this, I propose that we shall have from time to time conferences to which each station shall be permitted to send a representative.

Mr. RICE. If it is in accord with the idea of having an interchange of information between the times of conferences, to whom should communications be addressed?

Mr. WATTEYNE. This was the first point, that each experimental station should have a representative in the organization, and to him communications should be addressed.

Mr. RICE. Would the communications go direct to him, or through the embassies?

Dr. CZAPLINSKI. The first communications should go through the governmental channels and subsequent one, no doubt, directly to the stations.

Mr. WATTEYNE. We can not determine now details of how communications should be sent.

Dr. BEYLING. At the preliminary meeting it was proposed to have the next meeting in England in 1914, but as England has no official representative we can not make any decision as to that point.

Mr. WATTEYNE. The first point to arrange is permanent organization. The Government of the United States has this to do—to write a letter in a diplomatic way stating the purpose and utility of the proposed organization, what has been done at this conference, and asking the Governments to appoint delegates to an international conference. The Governments will take the matter up with their testing stations and appoint the delegate or delegates. The matter of chairmanship and secretaryship is a matter that can not be taken up until later.

Dr. CZAPLINSKI. I agree thoroughly with Mr. Watteyne. In Austria communications should be addressed to the minister of public works, who will send the communications to a committee consisting of Government men.

Dr. **Beyling**. There are two ways of treating this matter—first, to call meetings together through diplomatic channels; second, to have the various experimental stations arrange matters among themselves. It would be more practical to adopt this second way, as it is much simpler. It goes without saying that the various stations in each country should agree among themselves as to the organization of the general conference. If, however, the majority prefer that we proceed in the diplomatic way I will agree to it.

Dr. **Czaplinski**. In Austria the committee can not do anything without asking the Government's permission. It is advisable, however, after the Government shall have given its permission to this organization that all further transactions and communications should go directly to the station.

Mr. **O'Connor**. As our British testing station has now been turned over to the Government it will be necessary to take the matter up with the Government. Sir Edward Grey is the minister of foreign affairs. He would send the matter through the proper channels.

Mr. **Rice**. As to the Canadian cooperation, should the matter be taken up through the home office, or direct with the Canadian representatives?

Mr. **O'Connor**. I believe that the Canadian station is entirely independent, and that the matter should be taken up direct with the Canadian authorities.

He suggested that the details be discussed and settled in order, as outlined in the New York meeting.

Mr. Rice offered a motion to the effect that it was the opinion of the delegates attending the conference that there should be a permanent organization of the mine-experiment stations of the various countries. This proposal was seconded and was unanimously adopted.

Mr. Rice then offered in the form of a motion the suggestions made in the preliminary conference, namely, that the chief purpose of the organization should be the discussion of plans and methods, the interpretation of the results of experiments, suggestions concerning special experiments that might be made most advantageously at different stations, and that there should be an interchange of samples and information.

Dr. Beyling thought that the unification of methods should be one of the main purposes, but that sufficient advancement had not as yet been made to establish the best method for certain investigations.

Mr. Rice thought that certain standards for comparison should be adopted; that although complete uniformity of methods could not be obtained, it would be desirable that as far as possible standards be adopted for comparison and future interpretation of results. Informal manifestations indicated the unanimous agreement of the conference.

Mr. **Watteyne**. When the committee is finally organized, every member who receives the appointment is to let the others know, and to send to the other delegates whatever publications his Government or station may issue, and put each other "en courant" of the work. When the delegates are appointed there will be no further necessity of using diplomatic channels. I agree completely with Dr. Beyling that it is only the first appointment that will have to be handled officially, and after that each member may do as seems best.

Dr. **Czaplinski.** I agree entirely with Mr. Watteyne that it is sufficient to exchange publications. If we would send samples of coal dust, the composition might change during the time required to make the exchange. The railroads would not accept transportation of explosives.

Mr. Rice said that in view of the opinions expressed it might be better to drop temporarily the matter of exchange of samples, but that he was glad to note that all agreed to the exchange of information.

Dr. Czaplinski replied that he did not mean that certain samples could not be exchanged from time to time, as the occasion might arise, even if such exchange were not provided for in an agreement. He said that, for example, one station might test a new safety lamp in which other stations would be interested, and that the lamp could then be sent to the other stations.

Mr. Watteyne thought the section covering the matter might include "exchange of information and eventually other things."

It was finally agreed that the section should read as follows: "To further this purpose by the exchange of information, publications, and eventually of other material."

Mr. **Watteyne.** As to place of meeting, I would like to see the next one in England if it can be arranged. This was the opinion of those at the preliminary meeting in New York.

Mr. **Rice.** I propose that if this conference meets with the approval of the various governments, and if an invitation is extended from England, that those attending this conference approve the plan of having the next conference in England; if not held in England, that it be held in Belgium.

After a little discussion it was decided that the next conference should be held in 1914 if arrangements could be made by that time.

Mr. **Watteyne.** Belgium and France being so near to each other, we might perhaps arrange to visit Frameries and Liévin at the same time. I shall see M. Taffanel. If the next meeting can not take place in England I shall invite the conference to Belgium.

SUMMARY OF INFORMAL AGREEMENT AS TO FUTURE CONFERENCES.

A summary of the details agreed upon in the conference is presented below:

Name of organization.—International Conference of Mining Experiment Stations.

Chief purpose.—Discussion of plans and methods; interpretation and comparison of results of experiments; suggestions concerning special experiments that can be made most advantageously at different stations; and furthering these purposes by the exchange of information, publications, and eventually other material.

Method of organization.—After the official approval by the nations taking part has been given, each testing station, whether private or official, shall be eligible to take part in the conferences; but the respective governments shall determine how the delegates shall be appointed to future conferences.

Time of meeting.—Each alternate year, at such date as may be agreed upon from time to time. If the conference is duly constituted as proposed, the first meeting is to be held in 1914.

Place of meeting.—For 1914 the meeting is to be held at the new English mine experiment station, if the plan is agreeable to the authorities in charge of that station; otherwise the conference is to be held in Belgium. Thereafter the place of meeting is to be agreed upon by a majority vote.

Method of voting.—Voting shall be by letter ballot, each participating country having one vote.

Officers.—The matter of officers for future conferences is to be settled after the approval of the several governments has been received. Meantime the secretary of the Pittsburgh conference, George S. Rice, is to assemble the papers and reports of discussions, with a view to their publication by the Bureau of Mines.

ADJOURNMENT OF CONFERENCE.

At the suggestion of Mr. O'Connor, a vote of thanks was tendered to the Bureau of Mines for its part in the conference. The meeting then adjourned sine die.

In the afternoon the members of the conference went to the experimental mine at Bruceton to witness the final coal-dust experiment, which was on a larger scale than the previous experiments. The explosion was very violent (see details on pp. 46–48), and the delegates were much interested in examining the effects of heat and violence within the mine.

CONFERENCE PAPERS.

A few of the papers read at the conference are presented at length in the following pages. Most of the papers, especially those dealing with topics that have been discussed in publications of the Bureau of Mines since the conference, are not given in detail, being merely summarized.

METHODS EMPLOYED BY THE BUREAU OF MINES IN INVESTIGATING COAL-DUST OR GAS EXPLOSIONS.

By George S. Rice.

ESTABLISHMENT OF THE BUREAU OF MINES.

In 1908, Congress appropriated funds for the investigation of the causes of mine explosions. This action was the direct result of the terrible loss of life in a number of great coal-mine disasters in January, 1907, which aroused public interest in the dangers of mining. The investigation was intrusted to the technologic branch of the United States Geological Survey, and in July, 1910, was transferred to the Bureau of Mines, established on that date, of which Joseph A. Holmes, formerly chief of the technologic branch of the Survey, was made director.

SCOPE OF THE BUREAU'S WORK AND ORGANIZATION.

The bureau's investigations, as developed by Director Holmes, may be classified under four heads, as follows: (1) Mine-accident investigations; (2) fuel investigations, relating to the production and use of coal, coke, oil, and natural gas; (3) investigations of mineral losses in mining, quarrying, concentrating, and metallurgical operations; (4) investigations of health conditions in the various branches of the mineral industry.

The organization of the staff of the bureau includes, besides an administrative division, under the director and the assistant director, and the engineer in charge at Pittsburgh, five technical divisions, as follows: Mining division, under the chief mining engineer; chemical division, under the chief chemist; mechanical division, under the chief mechanical engineer; division of mineral technology, under the chief mineral technologist; and the metallurgical division, not yet fully organized.

In this paper only the work of the mining division that relates to mine accidents is discussed.

MINE DISASTERS.

Since 1908, each mine explosion in the United States involving loss of life and many mine fires have been investigated by the bureau's engineers. Early in the work rescue or oxygen-breathing apparatus was obtained for use in investigating explosions and for the possible rescue of entombed men.

MINE-SAFETY CARS AND STATIONS.

Railroad sleeping cars, of which there are now eight, were procured and equipped as mine-safety cars, to enable engineers and other employees to render efficient aid at disasters and to give lectures and training in the use of breathing apparatus and in first aid to the injured. The full crew of each car consists of a mining engineer, a foreman miner, a first-aid miner, and a cook. In remodeling the cars, lower berths in one-half of each car were removed to make room for storing apparatus, receiving visitors, and for training work in mining towns in which more suitable quarters are not available. The remainder of each car is divided into rooms—a kitchen, an office for the engineer, and a room for the foreman and first-aid man.

Besides the cars there are also six permanent training stations, including the Pittsburgh station, in different coal-mining districts of the country. These stations, each in charge of a mine foreman, are equipped with apparatus and have smoke rooms in which training may be given with breathing apparatus.

Through the bureau's activity thousands of miners have received training. Miners who take the complete course prescribed are given certificates. A further result of the training has been that the larger coal-mining companies have equipped cars and stations similar to those of the Bureau of Mines, and have trained miners.

When this work was first undertaken by the Government in 1908 only a few sets of modern apparatus had been purchased by mining companies. Now there are several thousand owned by private mining companies, and many training galleries have been established.

By the aid of breathing apparatus, the mining engineers and miners are able to investigate mines or parts of mines that could not be entered without it. Following explosions and mine fires, a number of lives have been saved. In the case of mine fires, the use of breathing apparatus has prevented loss of much valuable property by enabling men to attack the fires directly or to seal the fire areas.

FIRE-DAMP INVESTIGATIONS.

The geological as well as the chemical conditions governing the occurrence of fire damp in the mines of certain districts and the origin of methane and other mine gases are being studied. Gas samples for analysis have been gathered from mines in all parts of the country and investigations are being made to determine the relation of the outflow of gas to changes in barometric pressures and to the structure and composition of the coal.

DANGER FROM GAS WELLS.

In many mining districts of Pennsylvania, West Virginia, Illinois, and Indiana the coal measures are underlain with strata that contain oil and natural gas, the indiscriminate drilling for which constitutes a serious menace to the coal mines in those districts. Old wells are sometimes found in working the mine, and in a number of instances explosions or mine fires have been caused by the workings encountering gas wells. In some districts the wells are close together, and interfere seriously with the proper arrangement of the mine workings, for a pillar must be left around the well when its position is known. The natural gas found in the United States contains about 85 per cent of methane and 14 per cent of ethane. The chemists find that the inflammable mine gases normally do not contain ethane, so that ethane content is a strong indication of gas-well leakage. In certain instances this indication has possibly prevented disasters—proper changes having been made in the well casings to relieve the pressure or to prevent further leakage into the mines.

METHANE IN ANTHRACITE MINES.

Many of the mines in the Pennsylvania anthracite fields are very gaseous, the return air of a few of them occasionally having as high as 2 to 3 per cent methane. As much as 1,400 cubic feet per minute of pure methane, equivalent to about 28,000 cubic feet of fire damp, is found in the return air of some of these mines under normal conditions of working.

METHANE IN BITUMINOUS COAL MINES.

In the great bituminous coal fields of the United States the volume of gas issuing into the mines varies widely. In the interior coal fields there is practically none, except in a few mining districts. In the Rocky Mountain coal fields and also in the southern Appalachian fields some methane is found. In West Virginia and Pennsylvania the gaseous conditions vary widely. Possibly the most gaseous bituminous mines are in southwestern Pennsylvania, but in these the main returns rarely carry over 0.5 per cent methane. Even in the mines that are considered very gaseous, the percentage is generally less than 0.3 per cent, although the return air from some parts of a mine may carry as much as 1 per cent of methane. The methane carried out by the air current from a so-called gaseous bituminous mine under normal conditions rarely exceeds 1,000 cubic feet a minute.

GAS ANALYSIS AS A GUIDE IN FIGHTING FIRES.

In fighting mine fires, and for determining whether an atmosphere is explosive, gas analysis has been found of the utmost value. Therefore mine-safety cars and rescue stations are equipped with portable apparatus of the Orsat type, as rearranged by G. A. Burrell, a chemist of the bureau. The samples are collected in glass bottles with ground-glass stoppers or rubber corks, a rubber bulb with tubes being used to fill the bottle with the sample. G. A. Hulett, chief chemist of the bureau, recently conceived the idea of taking such a sample in an evacuated flask or bulb in which the opening through the stem has been sealed by melting. When the sample is to be taken, a part of the stem is broken off at an annular filed nick. The gas rushes in, and then the small opening is sealed with wax. This method has proved successful. Samples gathered in the field, not requiring immediate analysis, as, for example, a sample from a mine fire, are put in containers and sent by mail to the Pittsburgh station.

It is generally conceded that the presence of a small proportion of gas renders coal dust extremely inflammable. The bureau desires to study in the laboratory and at the experimental mine the extent of this influence.

MINE-EXPLOSION INVESTIGATIONS.

Owing to mine explosions, the mine-fatality rate has been greater in the United States than in other large coal-producing countries; and unfortunately the rate for the past 10 years has been higher than in any previous decade. It is possible that the explosions may be attributed to the following causes:

1. The increased use of explosives in many western mines, and the practice of shooting off the solid without undercutting.
2. The failure to remove the fine dust made in undercutting with machinery before blasting.
3. The increased volume and velocity of the ventilating currents required by the larger number of miners employed in the average mine. One result of the increased velocity of the air current is that the fine coal dust is blown from the loaded cars as they pass through the entries, and another result is that the increased quantity of air dries the coal dust more readily.
4. The increased proportion of inflammable gas (methane) in the mine air, owing to the increased depth of the mines, and, in certain instances, to leakages of gas from wells.

5. The employment as miners of many unskilled men, a condition resulting from the tremendous expansion of the coal-mining industry, the output now amounting to over 450,000,000 metric tons annually.

6. The inability of many miners to speak English, and the consequent misunderstanding of orders given by English-speaking foremen.

7. Lack of discipline.

8. Lack of sufficient knowledge of gases and dust by foremen and fire bosses.

9. An insufficient number of foremen for the supervision of the work at the face and in other parts of the mine.

10. The great increase in the use of electricity. In 1910, 42 per cent of the total output of the bituminous coal and lignite was mined by machinery, and probably two-thirds of the machine-mined coal was mined by electric machinery. A large proportion of the hauling underground is by electric-trolley locomotives. The amount of coal thus transported is not known, but inspection would indicate that it is about one-half of the entire output. Many explosions have been attributed to electricity, especially those due to the use of trolley wires. Previous to 1902 electricity was little used.

RELATIVE RISK OF EXPLOSIONS IN THE UNITED STATES AND EUROPE.

The condition of the coal beds of the United States is generally much more favorable to mining operations than in Europe, except in the anthracite district of Pennsylvania and in some of the western fields, where the beds dip steeply. In the bituminous fields of Pennsylvania, West Virginia, Ohio, Indiana, and Illinois, from which the greater part of the output now comes, the beds are nearly level, lie rather close to the surface, and are generally freer from fire damp than the deeply buried and steeply inclined coal beds of Europe. On the other hand, the advantage of shallow, flat-lying beds introduces danger from coal-dust explosions. The coal is generally hard and requires undercutting, a process that produces fine dust. Moreover, the coal requires larger charges of explosives than the deeper-lying and more crushed coals of the European mines. In some mines, chiefly in the Middle West, the coal is not undercut but is shot "off the solid," a practice that requires excessive charges of explosives, produces much coal dust, weakens the roof, and is a frequent cause of blown-out or dangerous shots.

The passages, headings, or entries of most American mines follow the coal bed instead of cutting across them, as in the inclined beds of most European mines. Thus the walls of the passageways are of coal instead of rock or shale, the coal ribs tend to splinter, and the fallen coal is ground to dust by the feet or by the mine cars. Also, if an explosion has a good start, the coal-rib surfaces give off a certain amount of inflammable gas when struck by the hot flame of the explosion, and it is even possible that the advance air wave tears off particles of coal from the walls, which may enter into the explosion, as after a violent explosion it is often found that the coal ribs are scoured as by a sand blast.

To summarize the comparison, there is much more inflammable gas in European coal mines as a whole, but coal dust is produced in much larger quantities in the American bituminous mines. On the other hand, the temperature of the average shallow American mine is not high, being generally less than 70° F. (21° C.), and the roof is relatively strong, so that there is less difficulty in watering the dust or in humidifying the mine atmosphere than in the European mines.

HUMIDIFICATION.

Humidification of the ventilating current has been introduced in many American mines since 1908 as a result of the experiments of the bureau and the study of the problem by American mining engineers. A widely adopted method is to humidify the entering air current in winter by steam jets. Such humidification immediately raises the temperature of the intaking air considerably, sometimes as much as 10° F., and at the same time supplies the needed moisture for producing a nearly saturated condition of the air. It has generally been found that if the average relative humidity of the entering air is maintained above 90 per cent the floor becomes damp and any falling coal dust becomes damp enough to stick together, so that it is not brought into suspension by an ordinary concussion. No coal-dust explosions have occurred in any of the mines in which humidification has been thorough, and there must be several hundred such mines. Extensive coal-dust explosions have occurred in mines in which imperfect sprinkling has been done by water car or by fixed sprinklers, or in which so small a steam jet has been used that the humidification has not been complete or continuous. It must be admitted, however, that large masses of coal dust are not readily wetted and should be loaded out of the mine.

ROCK OR SHALE DUST.

The use of rock or shale dust for preventing and limiting explosions, developed at the Liévin station, France, and at Altofts, England, has been tried in only a few mines. There are two reasons, as follows: (1) The bureau has not yet been in a position to thoroughly test the method at the experimental mine, and many operators in districts where water is scarce, or where wetting or humidifying injures the roof, are waiting for the results of tests to be made under American mining conditions; (2) roof shale sufficiently free from carbonaceous matter is not found as frequently in American as in European mines, where the levels cut the inclined strata. The absence of a suitable shale may be noticed at the experimental mine, where the shale selected contained 8 per cent of combustible matter. On the other hand, there are limestone quarries near many coal-mining districts and the waste might be obtained at comparatively small cost.

DELIQUESCENT SALTS.

Crude calcium chloride is being used a little to supplement sprinkling by hose, or by water cars, and seemingly might be more used to supplement other means. It is of particular value in dampening the dust that settles in goaves or other places difficult of access which may be overlooked by the employee using hose or not reached by sprinklers. It may also be used to advantage in mines where the occasional wetting of gobs may lead to spontaneous fires.

RELATION OF THE WORK OF THE BUREAU TO STATE INSPECTION DEPARTMENTS.

Each State regulates mining operations within its domain; it makes its own laws and has its own inspectors. The State laws differ widely—some are detailed and specific, as are the Pennsylvania and Illinois laws; others are very incomplete and leave much to the discretion of the inspector. The Federal Government exercises no police control or supervision for safety in mining except in the Territory of Alaska. In those western States in which the Government owns the mineral rights of any land not yet sold the terms of sale are defined by the Land Office of the Department of the Interior, but the Government does not exercise any police supervision over the development of mines.

The United States Bureau of Mines does only investigative and educational work relating to various phases of the mineral industry. Its mining engineers have no power to effect improvements or changes in mining operations, or even to enter mines without the permission of the State authorities or the individual operators. However, the operators and State inspectors generally have welcomed the activities of the bureau and have freely permitted its engineers to enter mines and gather samples and make investigations. Many improvements that these engineers have been able to suggest have been adopted by the mine managers and owners.

Detailed reports of most mine accidents of any size and all mine explosions are submitted confidentially to the director by the engineers of the bureau. If the conditions are liable to lead to a repetition of the accident a confidential report is made to the mine operator. General reports covering each class of accidents are prepared by the bureau for publication in such a way as not to conflict with the reports of the inspectors of the various States. Later, it may be possible that those parts of the reports of individual mine disasters of scientific or educational value will be published.

EXPERIMENTAL MINE.

In 1910, the bureau decided to open a mine in which experiments relating to mine explosions could be conducted on a scale comparable with the operations of ordinary mining. The site finally selected, in December, 1910, is near Bruceton, Pa., 13 miles from Pittsburgh, on the Baltimore & Ohio R. R. The features considered in this selection were as follows:

1. The mine should be in a coal bed, like the Pittsburgh seam, in which inflammable dust is produced.
2. The coal bed at the place selected should be practically free from inflammable gas.
3. A natural-gas pipe line should be at hand to supply gas for experiments in the mine.
4. The mine should be free from dripping water and preferably self-draining, to permit experiments with dry coal dust.
5. The main openings should be drifts, to obviate the difficulties from shafts wrecked by violent explosions.
6. There should be a water supply for boiler use and for sprinkling or humidifying the mine.
7. The mine should be near a railroad to allow shipment of the coal, yet away from dwellings that might be damaged by explosions.

DESCRIPTION OF MINE.

The mine at this time (September, 1912) consists essentially of two main entries or headings, 1,260 feet long, driven in the Pittsburgh coal bed, and a diagonal heading 198 feet long, which intersects the air course at an angle of 55° at a point 117 feet from the mouth of the air course. A pair of left "butt entries" 210 feet long are turned at right angles to the air course at 850 feet from the main entrance. A single right "butt entry" 100 feet long is turned at right angles to the main entry at a point 775 feet from the main entrance. The total length of single entry is about 3,300 feet.

The mine is opened in a hillside, the crest of the ridge being 150 feet above the level of the entrances. The roof strata are chiefly carbonaceous shale. The entrances are of reinforced concrete, and for some distance inside the passages are lined with reinforced concrete, arched overhead. As finished, the width is 8 feet and the height of the crown above the floor 7 feet 6 inches. Where the entries are unlined the width is 9 feet and the height 6½ to 7 feet. The main coal which is exposed in the ribs averages about 5½ feet thick; above it is a 1-foot layer of soft "draw slate" or laminated shale, which is taken down in driving the entries; above this shale is a stratum of impure coal 6 inches to 2 feet thick; generally this forms an excellent roof, and does not require support except where "rolls" (small faults) are encountered. The tracks laid in the entries are ballasted with roof shale, or with furnace cinders.

SHELVING FOR COAL-DUST LOADS.

For experiments with coal dust it was necessary to introduce artificial shelving, since the coal dust could not be as effectively placed upon the ballast of the floor as on the smooth floors of concreted or steel galleries; moreover, in winter, when the mine is dry, it is necessary to keep the floor moist and packed so that the finer road ballast may not be thrown into suspension by the violence of an explosion. There are five shelves made of 3 by 4 inch timber, with a 3-inch face upward on each rib. Where the entries are lined with concrete the shelves are held by bolts projecting from the lining; where the entries are unlined the shelves are bolted to posts recessed into the ribs and tied in by bolts and concrete.

INSTRUMENT STATIONS.

At 200-foot intervals along the main entry and the diagonal entry are large instrument stations, each lined with reinforced concrete, and divided by a strengthening wall, on one side of which are the instruments and on the other, guarded by a heavy steel door, is the entrance. The instruments are placed on a shelf, and connection with the atmosphere of the entry is through holes in a thick steel plate flush with the rib of the entry. These large stations are for pressure manometers, circuit breakers, and other automatic apparatus.

FLAME AND PRESSURE CIRCUIT-BREAKER STATIONS.

Halfway between the large stations are small stations or boxes for flame and pressure circuit breakers, so that the circuit breakers may be placed at 100-foot intervals. Access to each station is had by opening a hinged steel-plate door.

INSTRUMENT CABLES.

Electrical connections for the various instruments in the large and small stations are made by cables that run from the observatory outside of the mine to the main entrance, and thence through a 4-inch pipe placed behind the concrete lining or carried in a groove in the rib, which is faced with concrete. The cables enter each station (large and small), where they are cut and the individual wires fastened to binding posts on a board so that any desired connection can be made to the various instruments.

METHOD OF IGNITING SHOTS.

In some of the coal-dust explosion tests, the source of ignition has been by a "blown-out shot," prepared by drilling into the coal face and lining the drill hole with a pipe; in others the igniting shot has been fired from a cannon. It is probable that the latter method will be used during the coming series of experiments in order to obtain a more regular initial pressure. Other methods of ignition, such as local gas explosions behind light brattices or bulkheads, may be tried.

SHOT-FIRING LINE.

The shot-firing lines are in the pipe carrying the other electric wires; immediately outside of the mine all of the wires are brought into a locked switch box. All switches are left open when men are in the mine preparing for an explosion test and the switch box is kept locked. When the men charging the igniting shot have come out, the switches are closed. At the observatory similar switches, also in a locked box, are closed. The keys of these boxes are carried by the men who charge and connect up the shot. When all connections are ready a button is pressed, completing the circuit, and thus firing the shot.

VENTILATING SYSTEM.

For ventilation while the mine is being developed, a small fan, capable of delivering 5,000 to 10,000 cubic feet of air per minute at 1 to 1½ inch water gage, is used. It is driven by a gas engine. During the first series of experiments, it was arranged as a blowing fan and was at the end of a 120-foot steel tube, in line with the diagonal heading. During the explosions the velocity of the ventilating current in the main entry was about 100 feet a minute, making the volume of air about 5,000 cubic feet a minute. This fan is now at the top of the small air shaft connecting with the air course, being used as a suction fan, the intake being through the main entry.

A large fan for experimental purposes (rated by the manufacturer as having a capacity of 80,000 cubic feet of air a minute with a 2-inch water gage, or 15,000 cubic feet of air a minute with 6-inch water gage) is driven by a 100-horsepower engine, through belting so that in case of sudden stopping of the fan by a violent explosion there will be less danger of wreckage. Reversing the fan is easily and quickly accomplished. This fan is near a branch of the external steel gallery, the connecting passage having two offsets with relief doors overhead and on the side, so as to afford full protection from an explosion. It is intended to run the fan without stop during an explosion, and to make some coal-dust experiments with the ventilating current moving first in one direction and then in the other, the influence of the direction of the air on the movement of an explosion being much debated by American mining men.

FIRST SERIES OF EXPLOSION EXPERIMENTS.

The opening test of the first series of coal-dust explosion experiments took place on October 24, 1911. The mine then consisted of two main entries, each about 700 feet long. Lack of funds that year (1911–12) prevented consecutive testing. In all there were 15 tests made. A full report of this series is now in the course of publication,[a] so that the details are not discussed at length.

SUMMARY OF RESULTS OF FIRST SERIES OF COAL-DUST TESTS.[a]

The chief conclusions from these tests were as follows:

(1) Explosions can be produced at will in underground passages having walls, roof, and humidity conditions typical of American mines.

(2) A blown-out shot in the coal face will raise in suspension coal dust placed on the floor or ribs, and will ignite it without a preliminary shot or concussion.

(3) Coal-dust explosions may be produced when the mine atmosphere is saturated and when the floors, ribs, and walls are damp, though not wet, provided enough dry coal dust is present.

(4) Although a coal-dust explosion often develops pressure and velocity slowly, it may reach a violent stage at less than 600 feet from its origin, and at that point may attain a velocity of 2,000 feet or more a second.

(5) The maximum pressures were not registered near the outlet, but 150 feet from the portal, although the dust load extended to the mouth, thus indicating measurable relief of pressure near the mouth.

PRESSURE-WAVE AND FLAME VELOCITIES.

The maximum pressure-wave velocity indicated by the pressure manometers and time markers was 1,954 feet per second, but the maximum flame velocity, indicated by the flame circuit breaker instruments, was 2,277 feet per second. The experiments were not numerous enough to warrant drawing conclusions as to the relation between the position of the flame and the forefront of the pressure wave.

MAXIMUM PRESSURES AT MANOMETER STATIONS.

The maximum pressure in the first series of tests in which the pressure manometers were in operation was recorded in the last experiment, 103 pounds per square inch being indicated at station 150, 150 feet from the mouth of the mine, whereas at station 50, 50 feet from the mouth of the mine, the maximum pressure was 60 pounds per square inch. A similar drop near the outlet was recorded in other experiments.

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[a] Rice, G. S., and others, First series of coal-dust explosion tests in the experimental mine: Bull. 56, Bureau of Mines, 1913, pp. 66–113.

The maximum pressures mentioned above were probably not as high as those in one or two other tests in which the instruments failed to act properly. If the recorded pressures only be considered, the maximum pressures obtained in this first series were much less than those obtained by Taffanel in his surface gallery, and about the same as those obtained in the Altofts gallery. The number of experiments was insufficient for a complete comparison between the results at the experimental mine and those at the several surface galleries.

KIND OF COAL DUST USED.

The coal dust used in the first series of experiments came from a mine in the Pittsburgh seam about 10 miles from Pittsburgh, Pa. For future experiments the coal dust that will be used for the standard tests will be made from the run-of-mine coal produced in the experimental mine. The proximate analysis, averaged from a series of standard full-section face samples taken in that mine, is as follows:

Moisture	2.7
Volatile matter	36.0
Fixed carbon	55.0
Ash	6.3
	100.00

The average ultimate analysis is as follows:

Hydrogen	5.38
Carbon	76.16
Nitrogen	1.53
Oxygen	9.23
Sulphur	1.40
Ash	6.30
	100.00

Samples of the dust, for analysis, are gathered after it has been distributed, prior to each test.

INFLUENCE OF SIZE OF DUST.

M. Taffanel, in his experiments, has given much time and attention to the size of the dust used. In the bureau's preliminary tests made to determine the availability of mine passages for explosion experiments, the dust used was fine enough for about 95 per cent of it to pass through a 100-mesh screen (100 openings to the linear inch) and about 75 per cent through a 200-mesh sieve. This coal dust was ground at the Pittsburgh station in a small ball mill, as the grinding plant at the experimental mine had not been finished.

All investigators agree that the finest coal dust, when freshly mined, is the most explosive, but all sizes are found in coal mines in regular operation. Samples of road dust have been obtained in various mines of the country, and although gathered with care to exclude large pieces, when screened, as a rule, 50 to 75 per cent will not pass through a 20-mesh sieve. Previous gallery tests have indicated that the part passing through a 20-mesh sieve is the explosive part, but in making tests of the maximum dust dangers a finer dust should be used. Therefore the grinder will be set to produce dust that will pass a 100-mesh sieve. Such dust contains a large proportion of the finest sizes. In the tests to follow, the proportions of the different sizes will be exactly determined. The finest or lightest dust is first brought into suspension by a concussion; if more coal dust is used there is more of this fine dust in the dust cloud preceding the explosion wave.

OBJECTS OF THE COMING SERIES OF TESTS.

Briefly, the objects of the coming series of tests are:

(1) To study the conditions under which coal-dust explosions may start, and to determine the effect of different conditions, such as location, size of passage or chamber, temperature, humidity, and composition of dust.

(2) To study the propagation of an explosion, and to determine the effect of the conditions named, the arrangement of the mine, and coal-dust loading on the velocity of the explosion.

(3) To study the development of pressure and the relation of the pressure to the velocity and the temperature of the flame.

(4) To determine, if possible, whether an explosion with uniform dust loading, in a uniform passageway, constantly increases in velocity and intensity to the outlet; or, whether after the explosion has reached a certain velocity and intensity those manifestations become constant or fall and subsequently rise to another culmination.

(5) To study all promising means of preventing explosions from originating or propagating, such as humidification or use of rock dust.

(6) To study the means of checking or stopping dust explosions after once started.

(7) To determine the influence of ventilating currents as to direction and volume upon the propagation of an explosion.

(8) To study the influence of the temperature and humidity of the atmosphere on the starting and propagation of an explosion.

INFLAMMABILITY OF COAL DUST.

By J. K. Clement.

Several methods for the laboratory study of the inflammability of coal dust have been used in England and in France. An apparatus devised for this purpose by J. C. W. Frazer, formerly chemist of the Bureau of Mines, consists of a glass bulb having a capacity of approximately 1,500 c. c., provided with tubulures at the top and bottom, and having an electrically heated platinum-wire grid above the lower tubulure and near the center of the bulb. A brass tube, closed at its upper end with a steel ball, communicates with the tubulure at the top of the bulb. Fifty milligrams of dust is blown through the lower tubulure against the glowing platinum grid, and the pressure developed by the explosion of the dust is measured by determining the smallest weight that must be placed on top of the steel ball to prevent its being lifted. The range of temperature of the platinum wire is 800° to 1,200° C.

With this apparatus more than a hundred samples of coal dust from the various fields of the United States have been tested, in addition to samples of coal dust containing various inert substances, such as shale dust and sodium chloride. The results show that sodium chloride is more effectual than shale dust in reducing inflammability.[a]

INSTRUMENTS USED IN COAL-DUST EXPERIMENTS.

By J. K. Clement.

The instruments required in the investigation of coal-dust explosions in galleries are as follows: Gages or manometers for measuring pressure; devices for measuring the velocity of propagation of explosions; apparatus for taking gas samples.

PRESSURE APPARATUS.

Under pressure apparatus are included recording manometers and maximum-pressure gages.

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[a] A description of the apparatus and the results of the tests are given in "A Laboratory Study of the Inflammability of Coal Dust," by J. C. W. Frazer, E. J. Hoffman, and L. A. Scholl, jr.: Bull. 50, Bureau of Mines, 1913, 58 pp.

RECORDING MANOMETERS.

In a recording manometer the moving parts of the instrument must have small inertia and high frequency. In the manometers used at Altofts, England, the pressure acts against a steel spring, the movement of which is recorded, by means of stylus, on smoked paper carried on a rotating drum.

In the manometer used by Taffanel at Liévin, France, pressure is exerted against a steel membrane which is connected by a series of levers with a small rotating mirror. The movement of the membrane is recorded photographically.

In the manometer designed by the Bureau of Mines the pressure is transmitted by means of a plunger to a cylinder composed of a large number of thin carbon disks about 2.5 cm. in diameter. The variation in the resistance of the carbon-disk column, due to changes of pressure, is recorded by means of an oscillograph.

MAXIMUM-PRESSURE GAGES.

Maximum-pressure gages of the ordnance pattern have been used in the gallery tests at Liévin and in several tests at the experimental mine. In the latter tests the records obtained simultaneously on different gages at the same station did not agree. This disagreement was probably due to lack of care in the construction of the instrument cylinders.

INSTRUMENTS FOR MEASURING VELOCITY.

To determine the rates of propagation of flame and of the pressure wave, circuit breakers that are connected in series with a chronograph and operated either by flame or by pressure are placed at intervals in the gallery.

PRESSURE CIRCUIT BREAKERS.

In the pressure circuit breakers the circuit is broken by the action of pressure on a piston. In the tests at the experimental mine it was difficult to adjust the pressure circuit breakers so that they would be actuated by the pressure produced by the coal-dust explosion and not by the pressure from the blown-out shot. The pressure from the latter was found to be as great as 4 pounds per square inch at 100 feet from the shot.

FLAME CIRCUIT BREAKERS.

Two types of flame circuit breakers—the detonator and the tinfoil types—have been used. In some of the tests at the experimental mine, detonators coated with silver carbide were used. In these tests the detonator type of circuit breaker proved to be unreliable, as the detonators were not always ignited by the flame. The behavior of the tinfoil circuit breaker was entirely satisfactory.

CHRONOGRAPHS.

For the B. C. D. chronograph used at Altofts, the circuit breakers are connected in series with the same electromagnetic recorder on the chronograph by means of an automatic commutator switch. At Liévin a chronograph with separate recorders for each circuit breaker is used. The objection to the first type of chronograph is that, when one circuit breaker fails to operate, circuit breakers placed to operate subsequently are not connected to the chronograph and therefore fail to record.

GAS-SAMPLING APPARATUS.

Satisfactory automatic instruments for taking gas samples during the passage of the flame have been designed in England and in France. A new gas sampler has recently been designed at the Pittsburgh station for the purpose of taking three samples in rapid succession during the passage of the flame.

REPORT OF EXPLOSION TESTS 16 TO 20 SEEN BY THE CONFERENCE.

By GEORGE S. RICE and L. M. JONES.

EXPLOSION TEST 16.

On September 12, 1912, in the preliminary test, the first of the second series made in the enlarged experimental mine, coal dust was placed on the side shelves in the outer 500 feet of the 1,270-foot main entry. Ignition was caused by a blown-out shot in a cannon such as is used in the explosives testing gallery in Pittsburgh. The axis is parallel with the floor and 24 inches above it. The charge prepared was 2¾ pounds of black blasting powder, tamped with 4 inches of clay stemming. The cannon pointed toward the mouth of the main entry, parallel with the axis of the latter. A platform of boards 5 feet wide was placed parallel with the axis of the cannon and 10 inches below it. This was loaded with 25 pounds of coal dust. Five hundred pounds of coal dust, or 1 pound per linear foot of entry, was placed on the shelving. That load was equivalent to about 0.26 ounce per cubic foot of gallery or entry space, or 260 grams per cubic meter. The coal dust used in this and the following tests was ground from coal mined from the Pittsburgh seam at the experimental mine, which is in the gas-coal district. An average of analyses of dust samples taken from shelves before the tests is as follows:

Analysis of coal dust used in tests.

Moisture	2.49
Volatile matter	35.29
Fixed carbon	53.66
Ash	8.56
	100.00
Sulphur	1.20

More than 95 per cent of the dust passed through a 100-mesh sieve. The fan was running during the test; the air current returning in the main entry had a velocity of 166 feet per minute, so that the ignition occurred in the direction of the moving air.

When the shot was ignited in the usual manner by a button pressed in the observatory, smoke and flame issued from the mouth of the entry. The explosion was not loud, and was accompanied by little violence beyond the movement of dust and small particles flying. The flame extended 50 feet inby the cannon and outby to 175 feet outside the main opening, or a length of 675 feet from the cannon. No instruments were used in this test.

EXPLOSION TEST 17.

The second explosion test of this series was made on Saturday, September 14, in the presence of the delegates to the international mine-experiment station conference, Messrs. Watteyne, Beyling, Czaplinski, Monroe, and Rice, and Mr. O'Connor, unofficial representative. The experiment was similar to the first. Coal dust in the amount of 1 pound per linear foot was spread from the mouth to station 650. The cannon was placed at station 550, so that the dust load extended 100 feet inby its position. The cannon was charged with 2¾ pounds of FFF black blasting powder and 4 inches of clay stemming as in the previous test. A 16-foot bench with a load of 25 pounds of coal dust was placed in front of the cannon. The volume and direction of the ventilating current were as in the preceding test.

It had rained from 8.30 to 10.30 a. m., and the mine surfaces were much damper than in the previous test, being wet to the touch. There was standing water in the ditch extending along one side of the entry. The coal dust on the shelves had been in position an average of two hours before the test, and although the absorbed moisture probably did not affect to any great degree the explosibility of the coal dust when in suspension, it may have increased the adhesiveness of the dust, thus making more difficult the throwing of the dust into the air near the cannon.

When the shot was fired, coal dust was blown out of the mouth of the entry, but no smoke or flame issued. There was a muffled sound of an explosion, slightly in excess of what would be caused by a shot alone. After an interval of two or three minutes the ventilating current brought out a considerable quantity of smoke, although not nearly so much as in the previous test. The pressure circuit breakers, which had been set at such a pressure that they would not be actuated by the shot itself, did not act. Examination showed that the guncotton tufts at 25-foot intervals had been ignited for 100 feet toward the entrance and for 50 feet inby against the air current. The result was of interest because it has been suggested that ignition inby could not be obtained from a shot pointed outby. No violence was manifest.

EXPLOSION TEST 18.

The third test was on Monday, September 16, in the presence of the foreign representatives and others. The igniting shot was fired from the cannon at the face of the right butt entry, 100 feet from the main entry. It had the usual charge of 2¾ pounds of FFF black blasting powder with 6½ inches of clay stemming. Coal dust was loaded in the amount of 1 pound per foot from the opening to station 950, the latter point being 175 feet beyond the mouth of the butt entry. The loading in the butt entry was in the amount of 2 pounds per foot. The fan was drawing 9,500 feet of air into the main entry. The humidity of the current passing the butt entry was 92 per cent.

A cloud of smoke and dust issued from the main opening and a smaller cloud from the gallery explosion-door section. The plates and boards on the explosion-door sections and the door at the mouth of the air course were disarranged, but otherwise no damage was done. At E 1050 a car standing on the main entry was moved inby 10 feet. The guncotton tufts were consumed inby the butt entry to station E 1150, or 200 feet beyond the coal-dust load. The tuft at station E 100 was unburnt, but the one at station E 75 was gone, so that the flame or hot gases extended for probably 700 feet outby the butt entry, but did not reach the end of the coal-dust zone.

Depositions of coked dust were noted on the shelving and posts uniformly on the outby sides, both inby and outby the butt entry.

Manometer records were obtained at stations E 150 and E 250, but the pressure at station 550 was too low to be recorded. The maximum pressures at stations 150 and 350 were 3 and 8 pounds per square inch, respectively. The velocity of the flame from station E 750 to station E 650 was 97.3 feet per second.

EXPLOSION TEST 19.

For the igniting shot of the fourth test, made September 19, the cannon was placed at the face of the main entry, 1,270 feet from the mouth of the mine, and was charged with 2¾ pounds of FFF black blasting powder and 4 inches of dry clay. Coal dust was loaded on shelves throughout the main entry, the right butt, the crosscut at station 1250, and from station A 1110 to station A 1160 at the rate of 1 pound per foot. A "Taffanel barrier" consisting of fifteen 20-inch shelves spaced 6 feet from center to center and loaded with finely ground shale dust was installed in the air course from station 1160 to station 1250.

Samples of air taken at the faces of the entries failed to show any methane. The humidity near the face of the main entry was 95 to 97 per cent. The volume of incoming air on the main entry was 8,800 cubic feet per minute.

When the shot was fired only a small cloud of dust appeared at the main opening. The flame extended only to station 1150, or 120 feet from the igniting shot. It passed through the crosscut and about 50 feet outby in the air course. Failure to obtain an extended propagation was probably due to the dust, which had become slightly dampened, not being raised as a cloud; and it is possible that the moisture on the surfaces of the entry and water standing in the side ditch along the entry may have cooled the igniting flame.

A second shot was fired after 200 pounds of fresh, dry coal dust had been scattered for 150 feet from the cannon. The flame did not extend any farther than in the previous explosion.

EXPLOSION TEST 20.

The explosion of the fifth test (Sept. 20, 1912) was originated as in the previous test at the face of the main entry. A charge of 2¼ pounds of FFF black blasting powder was placed in a hole drilled in the coal face and cased with 1½-inch pipe. In view of the wet condition of the mine, the cannon, loaded with a similar charge, was so placed in the crosscut that the flame would pass the outby corner of the crosscut and cross the flame of the shot in the face, an arrangement that miners term "cross firing." The coal-dust load was doubled. In addition to the 1 pound per foot previously used on the shelves, 1 pound per foot was scattered over the damp floor. This load extended throughout the main entry, through the right butt entry, and through the crosscut at station 1250 into the air course where a Taffanel barrier was placed, extending from the crosscut for 60 feet outby; outby this, 50 feet of coal dust was placed to show whether the flame would pass the barrier. Including the coal dust strewn on the damp floor there was about 0.56 ounce per cubic foot of (entry or gallery) space or 560 grams per cubic meter. A total of 3,000 pounds of coal dust was used in the loading. This dust was from the run-of-mine coal. A screen test gave the following results:

Percentage passing through screen.	Mesh.
100	80
97.5	100
93.3	150
75.9	200

The Taffanel barrier consisted of 10 shelves 20 inches wide heaped with shale dust. A feature unintentionally introduced, which may or may not have increased the efficiency of the stopping, was the closing of a light door across the crosscut at station 1250, and probably did not affect the velocity of the explosion wave passing into the air course where the barrier was placed.

It had been decided to make the explosion in quiet air, without moving current, by shutting off the air outby the explosion area in the air course and closing a door in the crosscut. This door and its frame were totally destroyed in the explosion.

The humidity at the face was 92 per cent.

An immense cloud of smoke and dust, accompanied by a loud report, issued from the main opening a few seconds after igniting shots had been fired. When the cloud had traveled to the end of dump, the flame shot out of the opening and disappeared in the cloud 100 feet from the opening.

An empty mine car standing 50 feet from the opening was hurled over a truck (a mine car without sides) that had been standing 50 feet beyond and landed, bottom up, in the field opposite, 210 feet from its first position; from there it bounded 21 feet to an upright position. The truck was thrown off the track 75 feet from its first position. Two small telegraph poles, 50 and 100 feet from the opening and 4 feet east of the center line, were broken off near the ground and one was hurled 294 feet into the field opposite.

Considerable damage was done inside the mine. The concrete along the springing line was badly cracked and in a number of places the cracks extended across the arch. At a point 74 feet from the mouth of the entry the old concrete arch was lifted and remained 8 inches higher than a later arch that had been built beneath it. The posts and shelves in front of all of the crosscuts were torn out. The shelving from a point 650 to a point 700 feet from the entrance was thrown down and that from point 775 to point 850 was torn apart. Three posts were found to have gone inby to points 834 and 1025, one being moved from its former position fully 235 feet in a direction opposite that of the explosion. The face of the entry was half covered with a pile of broken boards and blocks that had been thrown into the face. The cannon in the crosscut was moved back against the rib.

The Taffanel barrier was thrown down, the shelves broken into small pieces, and the shale dust from the shelves distributed along the air course for a considerable distance. The flame passed through the crosscut but not beyond the barrier.

The 18-inch brick stopping in a crosscut at 850 feet was blown out and some of the bricks were thrown into the No. 1 left butt entry opposite, as far as the face, 235 feet distant, and others were carried 150 feet outby in the air course. The water from a pool on the aircourse side of this stopping had been thrown in the air course outby the crosscut.

The track in the air course opposite the cut-through was torn up and bent toward the No. 1 butt entry. Shelves and ties were broken by flying bricks. Track ties were disarranged for 100 feet in this butt entry.

In the main entry the maximum pressures registered by the manometers in the stations 550, 350, and 150 feet from the mouth of the mine were 34, 114, and 88 pounds, respectively, the highest pressure not being recorded by the manometer nearest the mouth.

SPONTANEOUS COMBUSTION.

By Horace C. Porter.

Experiments in which powdered coal was heated to 100° C. in a current containing 50 per cent oxygen have proved that the process of absorption of oxygen by coal (not producing CO_2) is exothermic. Furthermore, the carbonaceous material absorbs oxygen more rapidly than does pyrite, thus showing that the oxidation of pyrite is not the principal factor in spontaneous combustion.

Coal subjected to this action, the absorbing of O_2 without forming CO_2, decreases in calorific value, which is another evidence, as Taffanel has pointed out, that the action is exothermic.

Coal artificially dried, or even partly dried, will recombine with water, with evolution of heat. Fifteen grams of coal dried at 100° C. was exposed in a crucible to the air of the room, and in 15 minutes its temperature rose from 25° to 31° C. When a few drops of water were added, the temperature rose quickly to 38° C. More precise experiments showed that this coal, when partly dried, so as to contain 2 per cent water, developed, on being moistened, 14 calories per gram. Hence, at least a part of the moisture of coal (expelled by drying at 100° C.) is held in a form of hydration, not as free moisture. In some coals this rehydration after incomplete drying may contribute to the initial spontaneous development of heat.

THE DISTILLATION OF COAL.

By Horace C. Porter.

In 1908 to 1910 experiments[a] were made to compare the composition of the volatile products obtained from different types of coal at different maximum temperatures. These experiments, though they were imperfect and unsatisfactory in that they involved a somewhat gradual heating of the coal from room temperature to the maximum in each test, and also considerable uncertainty because of secondary decomposition of the volatile products by passage over heated surfaces, served, however, to bring out differences in the volatile matter of different coals.[a]

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[a] Porter, H. C., and Ovitz, F. K., The volatile matter of coal: Bull. 1, Bureau of Mines, 1910, 56 pp.

The results show the following facts applicable to the study of coal-dust explosions and possibly other mining problems:

(1) The volatile products are not entirely composed of combustible matter; the noncombustible or inert constituents are a large and important part of the volatile matter from some coals, as shown in the table below:

Comparison of volatile matter and combustible volatile matter from four American coals.

Source of coal.	Volatile matter including moisture.	Volatile matter without moisture.	Combustible volatile matter.
	Per cent.	*Per cent.*	*Per cent.*
Pocahontas, W. Va	21.4	20.9	17.8
Pittsburgh bed, Pa	31.8	30.7	26.0
Zeigler, Ill	37.9	30.4	21.9
Sheridan, Wyo. (subbituminous coal)	50.0	38.8	17.1

The inert constituents of the volatile matter are largely water of composition and CO_2.

Although the proportion of volatile matter may be in some degree a measure of the inflammability of a coal, inflammability may depend not so much on the percentage of volatile matter as on some other factor which varies as the volatile matter. Certain coals that have a high volatile-matter content and are highly inflammable yield, on the other hand, a relatively small amount of combustible volatile matter and a large amount of the noncombustible, deadening materials, CO_2, and water, which would be likely to lessen the inflammability of the volatile products. The table below shows the results of heating 10 grams of the four coals mentioned above for 10 minutes to a maximum temperature of 600° C.

Combustible ingredients of volatile matter from four coals (not including CO and ethylene hydrocarbons).

Sources of coal.	Tar.	Higher homologues of CH_4	CH_4 and homologues.	Hydrogen.
	Per cent.	*C. c. per gram.*	*C. c. per gram.*	*C. c. per gram.*
Pocahontas, W. Va	4.4	11.0	41.5	19.1
Pittsburgh bed, Pa	11.5	19.4	48.4	21.0
Zeigler, Ill	8.2	6.8	29.6	15.7
Sheridan, Wyo	8.6	7.5	28.0	16.6

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[a] Since the conference the bureau has designed an apparatus with which a sample of coal may be quickly heated and the volatile products may be quickly withdrawn over cooled surfaces.

The results support the conclusions drawn above from the first table on page 49. The high-volatile Wyoming and Illinois coals yield less inflammable gas than the Pocahontas and Pittsburgh coals and less tar than the Pittsburgh, although inflammability tests seem to show that the Wyoming and Illinois coals are the more inflammable.

(2) Although the volatile matter distilled from coal during momentary heating in a coal-dust explosion is doubtless a factor in the ignition of the dust, or in the propagation of the explosion, other phases of the chemical character of the coal are probably important.

(3) The highly bituminous coals, such as those of the Pittsburgh bed, decompose on moderate heating (not over 600° C.), so as to yield volumes of C_2H_6 and other CH_4 homologues nearly equal to that of CH_4. All the coals tested give a large yield of these higher hydrocarbons on moderate heating. The high heat of combustion of these hydrocarbons and their low inflammability limit may have an influence on the inflammation of dust.

These facts also lead to the theory that different coals contain different proportions, first, of substances yielding "paraffins" on distillation, and, second, of substances yielding chiefly CO_2, CO, and water on distillation.

ESCAPE OF GAS FROM COAL.

By Horace C. Porter.

Coarsely broken coal from some mines, if placed in bottles as soon as possible after mining and kept at ordinary temperature and pressure, liberates a remarkable volume of methane. Practically the same amount of gas is set free at atmospheric pressure as in a vacuum.

The gas liberated is largely CH_4. A little CO_2 was found, but no CO, H, or higher homologues of CH_4 (within the limits of error of the analysis).

The various coal types varied widely in the amount and rate of gas evolution.[a] No definite connection was observed between this tendency and the composition of the coal.

The largest evolution of CH_4 was 0.67 liter per kilogram of coal in 50 days, from a southern Illinois coal from a region where the mines are decidedly gaseous.

The tendency of a coal to set free inflammable gas, even though the coal be not finely powdered, must add materially to the danger of gas or dust explosions in mines. Not only the suspended dust in the atmosphere of the mine, but also the coarsely broken coal that lies on the floors, continually liberates gas. In poorly ventilated parts of the mine the accumulations of gas may become dangerous.

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[a] See Porter, H. C., and Ovitz, F. K., The escape of gas from coal: Technical Paper 2, Bureau of Mines, 1911, p. 11.

WEATHERING OF COAL IN THE BED.[a]

By Horace C. Porter.

Samples of coal were taken at intervals along the entries at the experimental mine, according to the standard method used by the Bureau of Mines.

The analyses of these samples computed to the ash-free and moisture-free basis show that the coal is abnormally low in C and H and high in O for a distance of 35 to 40 feet from the outcrop. Beyond that point there is little variation, and the composition is normal.

A few samples were reduced to dust and compared as to inflammability. Coal taken 50 feet from the outcrop showed practically no difference from coal 600 feet from the outcrop, but coal 1 foot from the outcrop was much less inflammable. The outcrop sample absorbed oxygen and liberated methane much more slowly than did fresh coal from 600 feet in. The coal at 50 feet in showed, in these respects, some effect of weathering.

Seemingly weathering has not materially affected the chemical character of the Pittsburgh coal at the experimental mine more than 40 to 50 feet from the outcrop.

THE OXIDATION OF COAL.

By Horace C. Porter.

The experiments made to show the escape of methane from coal incidentally called attention to the rapid absorption of oxygen by coal at ordinary temperatures. The rate of absorption seemed to be affected by the percentage of oxygen in the atmosphere surrounding the coal and, of course, by the kind of coal and its fineness of division.

The possibility that this absorption may be surface condensation, as with charcoal, seems to be precluded by the fact that the oxygen absorbed could not be liberated again from the coal either in boiling water or in a vacuum at high temperatures. The absorption increases rapidly as the temperature rises. Only slight amounts of CO_2 are produced.

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[a] Porter, H. C., and Fieldner, A. C., Weathering of the Pittsburgh coal bed at the experimental mine near Bruceton, Pa.: Technical Paper 35, Bureau of Mines, 1914, 35 pp., 14 figs.

More precise experiments on finely powdered coal (80 to 100 mesh fineness) exposed to the action of pure oxygen at a constant pressure approximately as that of the oxygen in air, have shown remarkable variations in the rate of oxygen absorption by different coals.

Those coals that show strong tendencies toward inflammability, deterioration, and spontaneous combustion appear to have the highest rates of oxidation. These are the types, generally speaking, that contain a high percentage of oxygen. This coefficient of oxidizability may be an important factor in the inflammability of coal dust.

INFLUENCE OF CARBON DIOXIDE ON THE EXPLOSIBILITY OF MINE GASES.[a]

By J. K. CLEMENT.

Experiments to determine the influence of carbon dioxide on the explosibility of methane have been made with two objects, as follows:

(1) To determine what proportion of carbon dioxide would have to be added to a mine atmosphere to render it nonexplosive.

(2) To furnish data on the explosive limits in mixtures of methane, oxygen, nitrogen, and carbon dioxide, with which to interpret the results of analyses of mine gas.

The experiments, which were made in a gas-explosion pipette and in a steel cylinder with a capacity of about 2 liters, show these results:

1. The limits of explosibility depend greatly on the kind of ignition used.

2. A mixture of air and carbon dioxide containing 22.5 per cent CO_2 or more will not form with natural gas a mixture that is explosive in a gas pipette, with the ordinary type of spark gap.

3. By replacing the nitrogen of the air with carbon dioxide, it was found that when 95 per cent of the nitrogen was thus replaced no mixture of natural gas would explode in the gas pipette. Hence carbon dioxide, when present in mixtures of natural gas and air, reduced the explosibility of the mixture to a much greater extent than did the same volume of nitrogen. As carbon dioxide does not react chemically with any of the constituents of the mixture, under

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[a] Clement, J. K., The effect of inert gases on the inflammability of gaseous mixtures: Technical Paper 43, Bureau of Mines, 1913, 23 pp.

the conditions of the experiment, the dampening action of carbon dioxide is probably to be attributed to its high specific heat. The mean specific heats between 0° and 650° C. of carbon dioxide and nitrogen are respectively 10.6 and 7.2.

4. As might be expected, wider limits of explosibility were found in the steel cylinder with an arc from a 220-volt circuit than with the spark from the induction coil.

THE SAMPLING AND CHEMICAL ANALYSIS OF COAL DUST.

By A. C. Fieldner.

The analysis of a complex and somewhat unstable substance like coal dust is beset with difficulties.[a] The determination of the chemical elements composing the dust is a precise analytical operation that can be duplicated by different chemists, and thus afford comparable data, but to have comparable determinations of moisture, ash, volatile matter, and fixed carbon the procedure must be made strictly uniform. Hence, it is important to obtain by international cooperation the adoption of uniform methods for the proximate analysis of coal, and thus permit direct comparison of the analytical data relating to inflammability investigations.

To avoid moisture changes during sampling and analysis, it is the practice of the bureau's investigators to air-dry all samples of coal and coal dust immediately on opening the original sealed container in which the sample has been transported to the laboratory. The loss in weight on air drying is recorded and the analysis of the air-dried sample is computed to the "as-received," as well as to a dry and to a moisture-free and ash-free basis. After the sample has been air dried it is at once passed through a set of rolls that pulverize it to 10-mesh size, is rapidly mixed and reduced in an inclosed riffle sampler to 500 grams, which is placed in the porcelain jar of a pebble mill and sealed air-tight. The mill is rotated until the sample is of sufficient fineness to pass through a 60-mesh sieve. The laboratory sample of about 60 grams is preserved in a rubber-stoppered glass bottle. In this manner there is obtained for analysis a sample in which the moisture content is in approximate equilibrium with the atmosphere and consequently will undergo the minimum moisture change in the time required by the analyst for weighing his sample.

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[a] For detailed methods of coal analysis see Stanton, F. M., and Fieldner, A. C., Methods of analyzing coal and coke: Tech. Paper 8, Bureau of Mines, 1913, 42 pp., 12 figs.

Many experiments have shown this method of preliminary air drying and pulverization in the inclosed pebble mill to be more accurate than any other practical method.

PROXIMATE ANALYSIS—DETERMINATION OF MOISTURE.

Moisture in the air-dried sample is determined by heating 1 gram of the sample, placed in a shallow porcelain capsule, for one hour at 105° C. in an oven through which is passed a current of dry air. The air is dried by being passed through sulphuric acid. The sample is then cooled in a desiccator over sulphuric acid and weighed. Loss in weight is reported as moisture. This method for the determination of moisture, although not as accurate as drying in a vacuum or in a current of inert gas, is in rather general use both in this country and in Europe.

There is, however, one essential difference between the bureau's method and the usual technical method. In the oven an atmosphere of air dried by having been passed through sulphuric acid is maintained. The extent to which a coal sample can be dried by heating varies with the humidity of the surrounding atmosphere. In the ordinary air bath, with stagnant air, the humidity will vary from day to day, being dependent on atmospheric conditions, and will vary also with the quantity of moisture coming from the samples in the oven. A number of experiments have shown that the circulation of dry air through the oven chamber gives higher and more concordant results than heating in stagnant air.

DETERMINATION OF ASH.

Ash is determined by igniting the sample in a muffle maintained at a temperature of about 750° C. The part of the mine-section sample previously used in the moisture determination is used in the determination of ash. Here again there is an opportunity for disagreement between different laboratories, especially with coals containing notable quantities of calcium carbonate and iron pyrite. On ignition the calcium carbonate is decomposed and carbon dioxide driven off, either partly or completely, depending on the temperature. The iron pyrite is changed to ferric oxide, and, in the presence of lime, calcium sulphate may be formed. Experiments with a certain Illinois coal have given an ash percentage of 14 per cent by ignition to constant weight and complete combustion of carbon at a low red heat, whereas at a bright yellow heat only 12.5 per cent ash was obtained. Therefore, there should be a standard temperature for the ash determination.

DETERMINATION OF VOLATILE MATTER AND FIXED CARBON.

Volatile matter is determined by heating 1 gram of the air-dried coal in a well-burnished, 30-gram platinum crucible, closed with a well-fitting cover, for exactly 7 minutes over the full flame of the No. 3 Meker burner. The crucible is supported on a platinum triangle so that the bottom of the crucible is 2 cm. above the top of the burner. The orifice of the burner is adjusted so that the free-burning flame is 17 to 20 cm. in length. The temperature attained in the crucible is about 950° C. The loss in weight, less moisture, is called volatile matter.[a]

This determination is purely arbitrary in character, and duplicate results are attained only by a rigid duplication of conditions. The method used by the bureau is essentially that recommended by the American Chemical Society committee on coal analysis.[b] It is modified in that the Meker burner is used in place of the Bunsen.

ULTIMATE ANALYSIS.

The ultimate analysis of coal dust affords no particular difficulty. The usual method of combustion in a current of oxygen is followed. The products of combustion are passed over red-hot copper oxide to insure complete oxidation and over lead chromate to remove the sulphur. The water and carbon dioxide produced are absorbed in calcium chloride and caustic potash, respectively.

For the determination of combustible or organic carbon in stone dust or shale the following method is used: The pulverized sample is evaporated to dryness on the water bath with hydrochloric and hydrofluoric acid, this treatment being repeated twice. The residue is then taken up with hydrochloric acid, digested, filtered on ignited asbestos, transferred to an oven and dried at 105° C., and then immediately placed in the combustion furnace and carbon and hydrogen determined in the usual manner. The hydrofluoric and hydrochloric acid treatment decomposes the inorganic carbonates and hydrous silicates, from which the water of composition is then removed by drying.

DETERMINATION OF NITROGEN.

The nitrogen is determined by the Gunning-Arnold-Dyer modification of the Kjeldahl method. No difficulty is experienced in obtaining accurate results.

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[a] Since this paper was written, the apparatus for determining volatile matter has been changed to an electrically heated crucible furnace, which, by means of a thermocouple, is maintained at 950° C. The coal is heated in a 10-c.c. platinum crucible which is closed by a capsule cover. More concordant results are obtained than with the former method.

[b] Noyes, W. A., Hillebrand, W. F., and Dudley, C. B., Report of committee on coal analysis: Jour. Am. Chem. Soc., vol. 21, 1899, p. 1116.

DETERMINATION OF CALORIFIC VALUE.

Determination of the calorific value is made with the Mahler bomb calorimeter. The Bureau of Mines follows the usual American practice of recording the gross heating value—that is, with all products of combustion cooled to the temperature of the calorimeter.

DETERMINATION OF SULPHUR.

Sulphur is determined by the Eschka method. Accurate results are obtained.

RESULTS OF EXAMINATION OF COAL DUST UNDER THE MICROSCOPE.

By Reinhardt Thiessen.

PURPOSE OF EXAMINATION.

In the microscopic examination of coal dust, either as road dust, rib dust, or in the artificially ground form, many important details may be observed. Many of these lead to further investigation, the results of which lead to important conclusions bearing on the explosibility of coal dust.

RELATIVE SIZE AND NUMBER OF PARTICLES.

In coal dust that will pass through a 200-mesh screen a comparatively few of the largest particles measure about 50 to 70 microns in diameter. From this size down all imaginable sizes to ultramicroscopical particles are found, and the smaller the size the larger the number. The limit of visibility with ordinary illumination is reached with a size of about 0.25 micron. The number of particles still visible in any given field under the microscope with ordinary illumination is enormous. If when a 1 per cent mixture of coal dust and Canada balsam is placed under a microscope field with medium magnification the largest particles can be counted by units, those particles that are just within the range of ordinary microscopic vision must run up toward the millions. Under a magnification of 2,000 diameters it was impossible to count them in any given field. Although the smallest particles visible have a diameter of about 0.25 micron, it is fair to assume that there are countless smaller particles. In fact, ultramicroscopic investigation reveals an enormous number of particles not visible with ordinary microscopic observation. Unfortunately, however, there was no adequate means at hand to determine their number. It can readily be seen that the number of particles in a cubic centimeter of coal dust is enormous—almost beyond comprehension.

FORM AND STRUCTURE OF PARTICLES.

Most of the particles are angular, many approximating cubical, tetrahedral, or polyhedral forms, some having rather smooth edges and surfaces, others being irregular, and many showing numerous irregular concave and convex surfaces. The specific surface of all of them is much greater than that of a sphere, and of the majority is larger than that of a cube.

Furthermore, in observing minute particles of extremely thin coal sections, with proper dark-ground illumination, or other ultramicroscopic means, it is found that most of the particles that, under ordinary microscopic vision, appear as if perfectly glassy or homogeneous are spongy, or as if composed of a large number of micellæ; in other words, it is shown that they are not homogeneous solids, but are highly heterogeneous.

BROWNIAN MOVEMENT.

Another point that deserves especial attention is the Brownian movement of the fine dust particles. It is phenomenally active alike in water, alcohol, or xylol. Even in Canada balsam it is pronounced. The amplitude is particularly large and the translatory speed high. The motion is not confined to very fine particles, but may be noticed in rather large ones; that is, particles of a size that, in other bodies, would show no movement at all here show active movement. As the cause of this phenomenon has never been satisfactorily explained, no reason for such great activity in coal dust can be given, but it must be significant.

On the dry dust the minute particles tend to aggregate in small lumps, and also a great number adhere to larger particles.

A microscopic examination of coal dust shows that it contains particles ranging from coarse to colloidal. Furthermore, the larger proportion of the particles, if not all of them, are not true solids, but are grouped into heterogeneous masses, probably in a biphased condition, with solid-gaseous surfaces of separation.

A microscopic examination of coal dust screened with a 200-mesh screen shows that the sizes range from 70 microns to an ultramicroscopic size—certainly below 0.25 micron; also that the smaller the particles the greater their number. As the specific gravity of Illinois coal is about 1.55, 1.55 grams of coal dust, if the particles be assumed to measure 1 micron, would represent approximately one trillion particles, with an absolute surface of certainly more than 6 square meters and a specific surface of 60,000; and if the particles be assumed to be 0.1 micron in diameter, there would be approximately one quatrillion of them, with an absolute surface of 60 meters and a specific surface of 600,000.

ABSORPTION PHENOMENA.

Platinum sponge glows when exposed to hydrogen because the hydrogen is condensed upon a film of condensed oxygen. Charcoal freed from air will absorb 60 to 80 times its volume of CO_2. Meerschaum, fuller's earth, and powdered glass exhibit similar phenomena.

Mixed gases are absorbed more slowly than is a pure gas; one gas is able to suppress a second already absorbed, and the equilibrium is often reached only after days. In a mixture of two gases both are absorbed; the one that is absorbed most when alone is absorbed most from the mixture. Much of the nature of the absorption depends upon the gas or gases already present; it is also certain that the nature of the gas absorbed is of much greater importance than the chemical nature of the absorbent.

The foregoing discussion leads to an inquiry as to whether coal dust may show the same phenomena. Experiments have been made by the bureau's investigators, but the figures so far obtained are not altogether reliable, although enough has been done to show that coal dust should be classed with charcoal, meerschaum, fuller's earth, etc., as regards gas absorption.

The bituminous coal from Sesser, Ill., used for the experiments was ground to 200-mesh fineness, no precaution being taken to keep it from the air. Natural gas, oxygen, carbon dioxide, hydrogen, and ammonia were used in the test, no attempt being made to remove gases already absorbed by the dust.

7.688 grams of dust absorbed 32.5 c.c. of natural gas. In an average of three experiments $\frac{V}{M}=5.6$, in which V is the volume in c.c. of gas absorbed and M the weight of coal in grams.

After equilibrium had been established the natural gas was replaced by dry ammonia until the equilibrium had again been reached. The total volume of ammonia absorbed amounted to 261.3 c.c. In this case $\frac{V}{M}=44$.

The absorption of oxygen and carbon dioxide was small, probably because the dust was already well saturated with both gases.

The absorption of hydrogen in 6.88 grams of coal dust was 11.3 c.c.

These experiments must be considered as preliminary only. More accurate apparatus must be devised to procure reliable results.

SAMPLING AND ANALYSIS OF MINE GASES.[a]

By G. A. Burrell.

SAMPLING MINE GASES.

Hitherto most of the mine-gas samples taken by the bureau have been collected with the aid of small rubber syringe bulbs. Under the present method of sampling, glass containers are exhausted at the laboratory by means of a vacuum pump and are sealed in a flame. At the place in a mine where the gas sample is to be collected a capillary glass-tube extension on the container is broken and the mine air fills the container. The broken end of the container is sealed with a small piece of soft wax.

When a sample is collected, all data pertinent are recorded, such as wet and dry bulb reading, barometric reading, velocity of air current (if sample is taken in the moving air), and cross section of area. Most of the containers have a capacity of 300 to 350 c. c. All samples are collected in duplicate.

ANALYSIS OF MINE GASES.

Haldane's apparatus, slightly modified, is used for much of the analytical work. Carbon dioxide, oxygen, methane, carbon monoxide, and hydrogen can be determined with an accuracy of 0.01 or 0.02 per cent. When more than one combustible gas is present, a determination of the excess oxygen is made. Carbon dioxide is determined by caustic potash absorption, oxygen by alkaline pyrogallate absorption, and the combustible constituents by the slow-combustion method. The Haldane apparatus is used for those mixtures containing less than about 5 per cent methane.

The iodic acid method for carbon monoxide is used for checking the carbon monoxide determinations made with the Haldane apparatus and for the determination of smaller quantities of carbon monoxide than can be detected with that apparatus. As ordinarily used for the determination of small quantities of carbon monoxide—0.3 per cent and less—it has been found that mine air vitiated by explosions, explosives, and gasoline motors, and containing such small quantities of carbon monoxide, does not usually contain olefin hydrocarbons or other gases that reduce the iodine pentoxide in sufficient quantity to necessitate a more complicated train than one consisting of caustic potash solution, sulphuric acid, and calcium chloride.

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[a] See Burrell, G. A., and Seibert, F. M., The sampling and examination of mine gases and natural gas: Bull. 42, Bureau of Mines, 1914, 116 pp., 2 pls., 23 figs.

A special bulb burette has been constructed, consisting of eight superimposed bulbs, each having a capacity of 100 c. c. A narrow constriction joins each bulb, upon which the graduation mark rests. With this burette any number of 100 c. c. of sample from 100 to 800 can be used for measuring the sample and forcing it through the train. An aspirator bottle on the end of the train assists this displacement.

APPARATUS FOR OTHER ANALYSES.

For the analysis of gas mixtures containing large quantities of CO_2, CO, CH_4, etc., the apparatus used contains a burette with a capacity of 100 c. c. and is graduated in tenths of 1 c. c. The burette has the same diameter from top to bottom. Carbon dioxide is removed by caustic potash solution, oxygen by alkaline pyrogallate solution, olefins by fuming sulphuric acid, carbon monoxide by cuprous chloride solution, hydrogen by slow combustion or colloidal palladium, and methane and ethane by slow combustion.

Forms of gas-analysis apparatus have been assembled for special purposes; for example, a portable apparatus for the rapid determination of methane by mine superintendents, inspectors, etc.; a portable apparatus for the approximate analysis of mine gases to determine the composition of the atmosphere in areas that have been sealed because of mine fires; a laboratory form of apparatus for the exact determination of methane by chemists at coal mines, etc.

Hydrogen sulphide is quantitatively determined by means of standard solutions of sodium thiosulphate and iodine.

Palladium chloride or blood solutions are used for the qualitative detection of carbon monoxide.

DETERMINATION OF OXIDES OF NITROGEN.

Experimental work performed by the bureau has shown that Busch and Guthbeier's "nitron" method, used for the quantitative analysis of nitric acid and nitrates, may be used for the determination of small quantities of oxides of nitrogen in mine air.

CHEMICAL CONTROL OF GAS MIXTURES USED AT THE PITTSBURGH STATION.

Natural gas is used at the Pittsburgh station in forming the gas-air mixtures used for testing explosives, safety lamps, electric motors, etc. The natural gas contains about 83 per cent methane, 16 per cent ethane, 0.03 per cent carbon dioxide, and 0.97 per cent nitrogen. Hydrogen, carbon monoxide, and olefin hydrocarbons are absent. A study of the explosibility of the gas indicated that it differed little in that respect from methane, the combustible gas usually found in mines.[a] Each mixture as it is prepared at the gallery is sampled and sent to the laboratory for analysis. The sample is simply transferred to a gas-analysis apparatus, measured, and burned by the slow-combustion method. The contraction produced by this combustion is measured. When the contraction produced by the burning of 100 c. c. of natural gas is known, it is a simple calculation to compute the amount of natural gas in the gas-air mixture from the contraction in volume it undergoes on burning.

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[a] Burrell, G. A., The natural gas used at Pittsburgh: Bull. 15, Bureau of Mines, 1912, pp. 66-77.

At one of the galleries a thermocouple is used by H. H. Clark, electrical engineer, for determining the percentages of natural gas in the mixtures he uses for testing the safety of electrical apparatus in mines. The readings of this thermocouple were standardized by gas analyses.

OTHER METHODS OF DETECTING GASES.

A gas interferometer has been received by the bureau, and has been fully tested. It gives satisfactory results.

Birds for use in exploring mines are kept at all the mine-safety stations and cars of the bureau. They have been found much more sensitive to carbon-monoxide poisoning than mice.

Paraffin hydrocarbons in a mixture of several are separated from each other by fractional distillation at low temperatures.

MINE-RESCUE BREATHING APPARATUS.

By J. W. Paul.

APPARATUS UNDER TEST.

The bureau has had under consideration and has conducted tests of the following types of breathing apparatus:

Types of breathing apparatus tested by Bureau of Mines.

Type of apparatus.	Essential agent.	Place of invention.
Vajin Bader	Compressed air	United States.
National helmet	do	Do.
Barnum's respirator	Fresh air	Do.
Merryman's respirator	Oxone	Do.
Servus	do	Do.
Hall's emergency	do	Do.
Pneumatogen II	$KNaO_3$	Austria.
Aerolith	Liquid air	Do.
Westfalia (Meco)	Compressed oxygen	Germany.
Draeger	do	Do.
Fluess (Proto)	do	England.
Koenig	Fresh air	Germany.
Westfalia	do	Do.

The tests were made in irrespirable gases (sulphur dioxide, formaldehyde, and smoke from burning straw, excelsior, etc.) in training rooms and in gases following mine explosions and mine fires. The plan of procedure of the tests has been similar to that adopted by the

English Royal Commission as outlined in its first report on breathing apparatus.

The tests made resulted in the purchase for the bureau of a number of the Fluess, Draeger, and Westfalia types of apparatus for use in mine recovery and rescue work, those types being the only ones available that met the requirements and local conditions.

The compressed-air types were found to be unsuited for use in poisonous gases, and the duration of their efficiency with one charge was insufficient for mine work.

The types that generate oxygen by chemical means have so far proved objectionable on account of the slowness with which they liberate the required quantity of oxygen and the high temperature and humidity of the regenerated air. No opportunity has as yet been offered for tests of the latest model of the Pneumatogen apparatus recently ordered.

The liquid-air type has met the laboratory test requirements in so far that it is light and furnishes a cool air and in sufficient quantity, but it has not been adopted for actual mine work because of inability to procure with any degree of regularity a supply of liquid air from the factories, which are far removed from the mining centers. The manufacturers of the apparatus have not installed at any mine in this country an experimental unit for liquid-air production, although it is known that this type has received much favor in some parts of Europe.

During the past two years so much trouble has arisen in the use of the helmet type that its further use is doubtful, the mouth-breathing type proving more desirable.

TESTS.

The gas-tight smoke chamber at the experiment station has been utilized for these tests, in which formaldehyde, sulphur dioxide, and smoke have been used. Repeated tests with different subjects have been made with the various types of apparatus, the tests extending over periods of two hours, during which frequent observations have been made as to the composition of the breathable air, and the pulse, respiration, and physical condition of the subject. With the three types, Draeger, Fluess, and Westfalia, the subjects have had no difficulty in performing their prescribed tasks uninterruptedly until the exhaustion of the supply of oxygen in the bottle.

The individual record of each person trained with the breathing apparatus constitutes a test of the apparatus used. The use of the apparatus in mine-recovery work also constitutes a test of the efficiency of the apparatus, as a record of the behavior of the apparatus and its wearer and analyses of the mine atmosphere obtained are recorded.

TRAINING AND RECORDS.

On the seven mine-safety cars and at the six rescue stations a course of training is prescribed for miners and mine officials who wish to qualify for a certificate. This course covers instruction in the mechanical construction, assembling, charging, and care of the apparatus, and training in its use in mines and in training quarters in the presence of irritating fumes and smoke. Training for two hours each day for six days is required for the completion of the instruction and training for those who are certified by the medical examiner as being in proper physical condition to take the course. In addition to the rescue training the applicant, to obtain a certificate, is required to complete the course in first-aid work.

ACCIDENTS IN USING BREATHING APPARATUS.

Two experienced foreman miners in the employ of the bureau have lost their lives while wearing apparatus within mines. The first death appears to have been due to exposure to poisonous gases immediately prior to putting on the apparatus, and to penetrating then the area filled with the gases from a mine fire. The other death was due to failure of the mechanical construction of the cooler of the apparatus worn, which permitted the gases from a mine fire to enter the circulating system of the apparatus.

LACK OF STANDARDIZATION OF PARTS.

Some difficulty has been experienced from reducing valves failing to deliver the required quantity of oxygen. Upon testing new valves it is found that they furnish varying quantities of oxygen and at varying pressures, showing a lack of standardization at the factory.

At the suggestion of the writer, the Draeger factory (Lubeck) has recently submitted two apparatus, one equipped with double reducing valves which may be operated separately or in unison, and a second equipped with a by-pass valve for emergency use.

EQUIPMENT OF THE BUREAU AND ELSEWHERE.

The following constitute the breathing-apparatus equipment of the bureau:

Draeger, 1904-7 type	18
Draeger, 1910 type	47
Draeger, 1911 type	16
Draeger, ½-hour type	3
Westfalia	36
Fluess (Proto)	8
Aerolith	5
Pneumatogen II [a]	1
Koenig Fresh Air	1
Westfalia Fresh Air	4
Hall's Emergency	30
Total	169

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[a] Model III ordered.

There are now installed at mines in the United States about 1,000 breathing apparatus, principally of the Draeger, Fluess, and Westfalia types.

Private rescue-training stations have been equipped for training and emergency work in several of the States, and seven railroad cars for such work have been put in service by different mining companies, and three by the State of Illinois.

The schedule of training that the bureau uses, together with the form for reporting the work accomplished by the person who may wear the apparatus, and the physician's examination card are described in publications of the bureau.[a]

The extent to which persons trained should be tested to ascertain the cumulative physiological effects of artificial breathing so far as the tests will determine the adaptability of certain persons for rescue service is a matter upon which the advice of this conference should be beneficial.

The advisability of training with compressed air in the oxygen bottle and discharging the exhaled air to the atmosphere to avoid the expense of upkeep for oxygen and regeneration is also suggested as a subject for discussion and advice.

MINERS' SAFETY LAMPS.

By J. W. Paul.

LAMP TESTS CONDUCTED BY BUREAU.

The schedule of tests of miners' safety lamps (those burning oil, naphtha, and acetylene) has for its purpose the determination of the following details:

1. The liability of the flame of the lamp to ignite explosive mixtures of air and gas under velocities of 200 to 3,000 feet per minute in horizontal currents and in ascending and descending and vertical and oblique currents.
2. The liability of the flame to ignite external mixtures of gas and air when subjected to sudden concussion.
3. The effect of small percentages of explosive gas upon the flame of the lamp and the effect of inert gases when present with explosive gas.
4. The liability of various types of internal igniters to fire gas outside of the gauzes of the lamps.

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[a] Paul, J. W., The use and care of mine rescue apparatus, Miners' Circular 4, Bureau of Mines, 1911, 24 pp.; Training with mine-rescue breathing apparatus, Technical Paper 29, Bureau of Mines, 1912, 16 pp.

5. The effect of the heat of the lamp flame on glass chimneys, and the effect of impact on the frame and the glass chimney.

6. The candlepower of the lamp and length of time it will burn.

7. The mechanical construction and stability of the lamp.

The lamps are subjected to the following tests:

1. A lamp being tested is placed in mixtures of air and explosive natural gas having 6, 8, and 10 per cent of gas, moving at a velocity of 200 to 3,000 feet per minute, to determine the velocity at which the lamp may ignite the mixture surrounding the lamp. The current is made to move against the lamp in a horizontal direction and while ascending and descending vertically and obliquely.

2. Upon the completion of the series of tests as here outlined, the lamps are given tests similar to those outlined in the preceding paragraph with the mixture of air and gas under pressure.

3. Under the conditions outlined in 1, coal dust is introduced into the current of air and gas to determine its effect, if any, upon assisting the ignition of the external mixture of gas and air.

4. The lamps are placed in a mixture of air and varying percentages of explosive natural gas to determine the action of the gas on the flame of the lamp.

5. The lamp is placed in a mixture of air and explosive gas and varying percentages of carbon-dioxide gas to determine the action of the gas mixture on the flame of the lamp.

6. Lamps equipped with internal igniters are placed in explosive mixtures of air and gas in a quiet state and in a moving current and the effect on the mixture surrounding the lamp of operating the igniter is observed.

7. The oils (illuminants) used within the lamps are tested as to their viscosity, gravity, flash point, congealing point, and composition.

8. A safety-lamp globe is tested by placing the globe in position in the lamp and allowing the flame of the lamp to impinge against the globe for a period of 5 minutes after the lamp has been burning with a full flame for 10 minutes, to determine whether the globe will break.

9. A safety-lamp globe is mounted within a lighted lamp with upfeed and placed for 5 minutes in an explosive mixture of air and gas moving at 1,000 feet per minute, to determine whether the heat will break the glass, and to note the character of the fracture.

10. A safety-lamp globe is broken by impact, by allowing the globe to fall by gravity upon a block of seasoned white oak, the distance the globe falls being recorded.

11. A safety-lamp globe is mounted within a safety lamp and when in a horizontal position a steel pick of 100 grams weight is permitted to fall a sufficient distance to break the globe.

12. To determine the candlepower of a safety lamp, a photometer equipped with a standardized lamp is used. The candlepower is determined along a line at right angles to the axis of the flame, also at angles to the axis of the flame, both above and below the horizontal. The candlepower is read at frequent intervals during the time the flame burns with a full charge of fuel.

13. The time a safety lamp will continue to burn with a full charge of illuminant is determined.

14. The wick of a lamp is of sufficient length always to be in contact with the bottom of the vessel in which the illuminant is contained and before being used the wick is dried to remove moisture.

15. To test the stability of the lamp, it is permitted to fall by gravity from different heights, and the lamp is suspended and subjected to sudden violence by dropping known weights attached by cable to the lamp.

APPARATUS USED IN TESTING.

For tests in moving currents of gas and air a horizontal steel gallery 6 inches wide, 12 inches high, and 20 feet long, having a vertical and two 45-degree compartments, is used. This is similar in detail to the galleries used at Frameries, Belgium, and at Gelsenkirchen, Germany. At the exhaust or outlet end is a steam ejector which may be used for aspiration, and at the intake end is a blowing fan, 10 inches by 24 inches, which may be used for driving the air through at a pressure slightly above atmospheric. The calibration of the gallery was much simplified by the use of the fan, to the exclusion of the ejector. A "Schorndorff" apparatus will be attached to the gallery for igniter tests.

The apparatus for testing under concussion is similar to that used at the University of Leeds, England, and consists of a 3-inch pipe, 15 feet long, having a diaphragm at a central point and a perforated coil into which the lamp is placed for testing.

For testing for flame caps a gallery 8 feet long, 24 inches high, and 8 inches wide, equipped with diaphragms for mixing gas and air, and a fan for a slow movement of the mixture is installed. This is similar in construction to the gallery used for the same purpose at the Birmingham University, Birmingham, England, though the method of operation is different.

Tests of igniters are made in a moving current of gas and air in lamp gallery 1, and in still atmosphere in gallery 3, which is a steel box 24 by 18 by 18 inches.

In the heat test of lamp chimneys the flame of the lamp is permitted to impinge on a chimney for five minutes; if the chimney is broken, the nature of the fracture is determined.

In the impact test the chimney is permitted to fall on a block of wood; also a small anvil weighing 100 grams is allowed to fall on the chimney, the height of fall required to break the chimney, and the character of the break being noted.

The candlepower of the lamps is determined by the use of a Leeds & Northrup precision photometer of the "Reichsanstalt" pattern, using the Lummer-Brodhem comparison screen. The standard of comparison is a Mazda tungsten lamp of 2 candlepower at 10 volts and 0.227 ampere, standardized by the United States Bureau of Standards.

LAMPS TESTED.

The following types of lamps have been procured and subjected to preliminary and official tests:

Type.	Where manufactured.
Wolf	Germany.
Seippel (bonneted)	Do.
Seippel (unbonneted)	Do.
Schenk	Do.
Pieler	Do.
Ackroyd & Best	England.
Davis	Do.
Clowes	Do.
Stokes	Do.
Ashworth-Hepple-White-Gray	Do.
Szombathy	Austria.
Davy	United States.
Clanny (bonneted)	Do.
Clanny (unbonneted)	Do.
Mueseler	Do.
American naphtha	Do.
Routledge-Johnson	England.
Thomas	Do.
Beard Deputy	United States.
Naylor	England.
Marsaut	Do.
Chesneau	France.

LEGISLATION RELATING TO SAFETY LAMPS.

The State of Pennsylvania is the first and only State to adopt a list of permissible safety lamps. Legislation of 1912 requires that all types of safety lamps used in the bituminous mines of the State shall have the approval of the State department of mines. The tests made by this bureau have been accepted by the department of mines of Pennsylvania, and the following types of bonneted safety lamps have been approved by the State mining department for general use in the bituminous mines of Pennsylvania if the velocity of the ventilation current ranges from 600 to 1,500 feet per minute: Clanny, Davis Deputy, Wolf, Schenk (new type), Seippel, and Ackroyd & Best.

The tests of igniters is in an experimental stage. Only within recent months have cerium alloy igniters made their appearance in America. The igniters generally in use are of the tape design. Some percussion igniters were introduced about two years ago, but because of their disrepute in Europe they were withdrawn upon advice from this bureau.

PROPOSED TESTS.

The influence of coal dust in the presence of moving currents of mixed gas and air in aiding the passage of flame through the gauze of a safety lamp provided therewith and the action of the flame of a safety lamp in the presence of methane, carbon dioxide, and air will be investigated in future tests.

The relative safety of lamps when tested in mixtures of gas and air under different velocities and above the normal atmospheric pressure will be determined. As yet no results of any such tests that may have been made in Europe have been received.

INVESTIGATION OF EXPLOSION-PROOF ELECTRICAL EQUIPMENT.

By H. H. Clark.

Several months ago the bureau completed an investigation of several explosion-proof motors.

Five motors were investigated, but the protection of two of them was too crude to be of interest. Of the three that operated well, one was protected by three layers of gauze covered by a poppet valve. Another was protected by several layers of gauze laid between heavy plates of perforated metal and inclosed in a bronze housing with a baffle-plate discharge opening. The third motor was protected by plates arranged after the German practice. This machine was built after the designs of the Allgemeine Electricität Gesellschaft.

All of the protective devices were vulnerable. The type embracing gauze and a poppet valve seemed to work fairly well, although it allowed flames to be discharged through the relief openings. However, ignition rarely occurred unless both poppet valves were open when the gas within the motor casing was ignited. With the poppet valves open, puncture[b] usually occurred. The motor that was protected by poppet valves was equipped with two of these devices. If one valve was fastened shut and the other left open, puncture was not likely to occur. Thus the conclusion was reached that the trouble was due to the low pressure maintained by the explosion when it was initiated with both valves open. The temperature of the discharged gases was no doubt affected by the way in which they expanded upon leaving the lip of the poppet valves, and this expan-

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[a] See Clark, H. H., An investigation of explosion-proof mine motors: Bull. 46, Bureau of Mines, 1912, 44 pp., 6 pls., 14 figs.

[b] The term puncture refers to the ignition of gas surrounding the motor by flames issuing from the motor casing.

sion depended upon the pressure generated within the motor casing by the explosion.

The device that used sheets of gauze laid between metal plates worked perfectly with gas alone, but punctured when coal dust was present just outside of the outer plate. My hypothesis of this is as follows: The gauzes chilled the discharged gases that passed through them first and these gases filled with noncombustible material the space just outside the gauzes, so that when the later flames emerged through the gauzes there was nothing to ignite and propagate the flame. When dust is present, however, it is thrown up in a cloud by the first blast of the explosion, and the later flames raise the dust to its ignition temperature, although, on account of deficiency of oxygen, it can not burst into flame until it reaches the outside air, where the dust takes fire and ignites the gas surrounding it. In support of this hypothesis, when puncture occurs no flames are seen to issue from the discharge openings, but the explosion seems to initiate from a point several inches outside them.

The plate protection was adequate as long as the motor was at rest, but when the motor was running the fan that was attached to the armature shaft refilled the motor casing with gas and caused a series of explosions that eventually heated the plates, so that they no longer prevented the passage of flames, which finally shot out 6 or 8 inches beyond the discharge openings and ignited the gas surrounding the motor.

I shall be glad to hear a discussion as to the best form of protective device and as to a suggested specification for covering "permissible motors"; also as to whether such a motor should be permitted to discharge flames even though no puncture occurs; as to how many trials should be made; as to how we should regard a motor that, when running, permits the occurrence of several successive explosions within the motor casing; and as to whether a motor should be considered unless satisfactory flame-proof starting devices are supplied and tested with it.

IGNITION OF GAS BY STATIONARY INCANDESCENT LAMPS.[a]

By H. H. Clark.

The bureau has just completed an extensive investigation of such incandescent lamps as are used for general illumination in mines.

About 1,200 lamps were tested. They varied in voltage from 50 to 225 and in candlepower from 8 to 50. Most of them had carbon filaments.

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[a] See Clark, H. H., and Ilsley, L. C., Ignition of mine gases by the filaments of incandescent electric lamps: Bull. 52, Bureau of Mines, 1913, 31 pp., 6 pls., 2 figs.; also Clark, H. H., Ignition of mine gas by miniature electric lamps: Technical Paper 23, Bureau of Mines, 1912, 5 pp.

The tests were made by smashing the lamps, by breaking off the tips, and by punching a hole in the necks of the lamps. Tests were also made with the naked filaments.

The favorable results obtained with such lamps as did not ignite the gas were confirmed by increasing the impressed voltage until ignition was produced when the lamp was smashed.

The results of the tests showed that all the classes of lamps tested will ignite gas under certain conditions, although some lamps are less likely than others to cause ignition. The lamps most used in the mines of this country (16 candlepower, 110-volt) gave five ignitions out of 62 trials in the smash test and three ignitions out of 23 trials in the tip-breaking test. The over-energy test also showed this lamp to be close to the danger line.

There are two points that I would be glad to have discussed—one is the most efficient means for protecting lamps from breakage after they are in place underground, and the other is the ultimate cause of the gas ignition.

In 58 tests in which the gas was ignited, the filaments continued to glow from 2 to 59 seconds after ignition had taken place, indicating that glowing filaments may ignite gas.

Fifty tests made upon 50, 60, and 120 watt, 110-volt lamps showed a greater percentage of ignitions when the lamps were connected in multiple than when they were connected in series, although the breaking spark was drawn at 110 volts in the former tests as against 550 volts in the latter. This result seems to show that the unassisted spark can not ignite gas.

By breaking a noninductive, 550-volt circuit between a steel contact and a carbon contact, it was found that when the current exceeded 0.15 ampere an ignition was always obtained when the circuit was opened. Yet 10 tests made with five 110-volt lamps connected in series across 550 volts produced no ignition, although the filament of the tested lamp carried one-half an ampere, and the spark must have been drawn at 550 volts. Neither were ignitions obtained from 11 similar tests made with lamps carrying approximately 1 ampere.

I believe that the spark that is drawn when a lamp filament is broken is to a great extent "blown out" by the entering current of gas. To prove this point, a 50-watt, 55-volt naked filament was connected in series with noninductive resistance, and 0.5 ampere (one-half normal current) at 110 volts was passed through the circuit. The filament was then broken by gradual pressure which distorted it until it snapped off. Five ignitions were obtained from six tests made in this way. Similar lamps were then connected in series with noninductive resistance across 550 volts, and the filaments were disrupted by smashing the lamp bulbs. Five tests were made, three at 0.6 ampere and two at 0.7 ampere. No ignition was obtained in any of the five trials, although the current was 20 to 40 per cent greater and the voltage 400 per cent greater than in the five tests that gave ignitions with the naked filaments. It is therefore manifest that the inrushing gas has a quenching effect upon the spark.

Five tests were then made by connecting a 175-watt, 55-volt lamp (normal current 3.5 amperes) in series with a noninductive resistance of such value that 2.25 amperes flowed through the filament when 550 volts was impressed across the lamps and resistance. The lamp was surrounded with gas and smashed while carrying this current. No ignition resulted from five trials, although the current broken was somewhat greater and the voltage ten times greater than in the tests of lamps that invariably produced ignition when broken while burning under normal conditions.

It is manifest that the spark alone did not cause these ignitions, if ignition could not be caused by a similar spark with ten times the voltage behind it.

I am inclined to believe that the likelihood of gas ignition by the standard carbon filament is directly proportional to the cross section of the filaments. The filaments of 24 different types of standard lamps were measured by Reinhardt Thiessen, assistant chemist, and the cross sections of the filaments were compared with their tendency to ignite gas as expressed in percentage of ignitions obtained in a given number of trials. All filaments having a cross section of 0.000177 square centimeter, or less, failed to ignite gas when the lamps were smashed. All filaments having a cross section of 0.000234 square centimeter, or more, invariably ignited gas under the same conditions, whereas filaments having a cross section of 0.000194 square centimeter ignited gas in 50 per cent of the trials, and filaments having a cross section of 0.000215 square centimeter ignited gas in 83 per cent of the trials. It is true that a similar relation existed between the current flowing in the filaments and the percentage of ignitions that they gave, but I do not regard this relation as significant, because I believe that the spark that is drawn when a filament is broken is not responsibie for the ignition of gas.

If the bulb of a glowing incandescent lamp is smashed while surrounded by an explosive mixture, and if the filament is not broken by the jar of the blow, the filament is first cooled by the inrushing gas and then broken by it. Of course, the cooling and the breaking are almost simultaneous, but I believe that ignition, if it occurs, takes place during the period of cooling and before the filament is broken.

If this hypothesis is correct, there are two reasons why large filaments ignite gas more readily than smaller ones. First, the temperature of the larger filaments is not reduced so greatly before they are broken, and, second, the larger filaments do not succumb so quickly to the breaking action of the inrushing gas.

PORTABLE ELECTRIC LAMPS FOR MINE SERVICE.[a]

By H. H. CLARK.

The bureau is now preparing to conduct an investigation of portable electric lamps for use in mines. Besides dry-battery flash lamps there are now on the market at least half a dozen American portable electric lamps.

The bureau feels that the introduction of the electric lamp will increase safety in mines whether gaseous or not, and hopes to stimulate interest in the manufacture and use of such lamps by thoroughly investigating as many as can be obtained, for the purpose of making a report that shall deal with the efficiency and cost of maintaining the lamps as well as with their liability to ignite gas accidentally.

Of the six kinds of American lamps immediately available, two are of the self-contained type, the lamp and the battery being mounted together; two are of the separate type, the lamp being mounted separately from the battery and connected to it by flexible conductors; and each of two others is designed to be used either as a self-contained lamp or with the lamp separate from the battery.

The self-contained types are, with one exception, designed to be carried in the hand. The design and construction of the exception noted contemplate the wearing of both lamp and battery upon the miner's cap.

The bureau has already tested over 100 such lamp bulbs as would be used in portable lamps and the results show that ignition of gas will be caused by any such bulb that gives about 1 candlepower or more, if the bulb is broken so that the filament remains intact. The bureau has also made a few preliminary tests with sparks from non-inductive circuits. The results so far obtained show that such sparks as would be given from a portable electric lamp will scarcely ignite gas. The principal source of danger from portable lamps is, therefore, the breaking of the bulb so that the filament is intact.

In the bureau's future investigation each lamp will be examined as to its ability to ignite gas and the candlepower, the life of the lamp bulbs and the number of hours that the batteries will operate their respective lamps on one charge or filling will be ascertained. Also, the care and manipulation incident to the use and maintenance of the lamps, the convenience and adaptability of the various lamps to the purposes of mining, and the cost of maintenance and repairs will be compared. As a result of this investigation the bureau will be able to draw specifications for lamps suited for underground service

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[a] Clark, H. H., Permissible electric lamps for miners: Tech. Paper 75, Bureau of Mines, 1914, 21 pp., 3 figs.

These specifications should be of help to the manufacturer as well as to the consumer.

Personally, I would be glad to see a standardized test or a specification for examining or measuring the safety of a lamp that uses a bulb that ignites gas when the bulb is broken. I believe that we shall find that all lamps of any appreciable light-giving power must use bulbs of this sort, and therefore the method of protecting the bulbs from injury becomes the paramount issue. The measure of danger is the likelihood of so breaking the bulb that the filament will not be injured. The measure of safety is the converse, and if a lamp is to be pronounced safe the proof that its bulb can not be broken without destroying the filament must be established. In the test under consideration it is almost impossible to select conditions that will be fair to both sides, and so I believe it to be inadvisable to attempt to determine by test either the absolute or the relative safety of the lamps if a reasonable number of tests do not give positive results.

At present it seems to me that the best solution is to require for gaseous mines a bulb mounting so designed that a blow sufficient to break the bulb will automatically open the lamp circuit. In presenting the standardization there are other details with which we should like your assistance, such as fixing the minimum candlepower, the minimum length of burning on one charge, the most desirable distribution of light, the minimum length of life of the lamp bulbs, and a test for determining the ruggedness of the equipment.

DANGEROUS ELECTRICAL EQUIPMENT AND METHODS OF SAFEGUARDING IT.[a]

By H. H. Clark.

It is the purpose of this paper to start a discussion as to what parts of underground electrical equipment are most dangerous to the miner and how the danger may be lessened, and to bring out recommendations for improving the standard of safety of electrical equipment in mines.

As far as can be determined from official reports, most of the electrical accidents in the mines of the United States are electric shocks, and the major part of these are caused by the trolley wire, which I consider to be the most dangerous part of the electrical equipment of our mines. Some shocks are also received from the trailing cables, service lines, and frames of coal-cutting motors and from the accidental charging of apparatus that is not supposed to carry current.

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[a] See Clark, H. H., The factor of safety in mine electrical installation, Technical Paper 19, Bureau of Mines, 1912, 14 pp.

It is rare that an explosion or a disastrous fire is reported as being caused by the electrical equipment.

In this country, as you probably know, direct current is used almost altogether for underground distribution and much bare wire is used not only for trolley service but also for carrying power to coal-cutting and pump motors. One side of the direct current distributing system is almost invariably grounded in order to use the earth as a return conductor.

It impresses me that the solution of the problem of reducing electrical dangers in coal mines embraces the following details: First, in forbidding the use of electricity in places where fire damp is regularly found in more than the most trifling amounts; second, in using flame-proof apparatus wherever outbursts of gas are likely to be encountered; third, in selecting underground electrical equipment with reference to the service that it must endure, choosing motors that are well insulated, switches that are durable, and other equipment capable of resisting rough handling and moisture; fourth, in properly installing equipment; fifth, in properly caring for and maintaining equipment.

The solution, as far as the operator is concerned, can be expressed in four words—good apparatus, good electricians—and the electricians are to my mind the first consideration. Good electricians mean the selection of good apparatus and the proper installation and maintenance of it. Thus are obtained three of the five essentials of safety as regards the use of electricity. For the remaining two essentials we must rely upon the State inspection departments, which are made up of men admirably equipped with the necessary experience and judgment and provided with full and unhampered authority over the matters at issue.

Does not our analysis therefore disclose the electrician as the important factor in the problem of safeguarding the use of electricity in mining? Of course lack of cooperation on the part of the mine management may so hamper the most competent of electricians that his efforts may be unavailing, but nevertheless the electrician holds the key to the situation.

My idea of a good electrician is, first of all, a good workman, a man who can do good work himself and who therefore can tell when work is well done. He should have a great deal of mechanical skill as well as experience with electrical machinery. He should believe in doing his work well, not merely well enough. His experience should not stop with the knowledge of making a simple splice or even with the ability to improve the commutation of a sparking motor. He should have an active mind, ready to grasp the lessons to be learned from his ever-varying experiences, and above all he should have the desire to acquire knowledge and improve the character of his work. Such a man, dealing with the problems at the closest range and in position of greatest advantage, has the power to safeguard truly and effectively the use of electricity in mines.

NOTES ON HOISTING ROPES.

By O. P. Hood.

The American habit of depending on the manufacturer of a product for engineering advice regarding the quality and use of that product is well exemplified in the current practice regarding hoisting ropes. The usual buyer follows the recommendation of the ropemaker as to the kind and quality of rope best suited to his needs, and is inclined to hold the maker responsible for a reasonable length of life for the rope and for its safe and adequate service. The ropemaker assumes this responsibility, even to the extent of submitting to a money forfeit, to retain the patronage of large customers who may have been dissatisfied with the performance of a particular rope. Some buyers believe that this policy of throwing the responsibility for adequate service entirely on the ropemaker is to be preferred to any method whereby the purchaser prepares a more detailed specification of quality and properties, expressed in the technical phrases of the subject of materials. It appears, however, that ropemakers are sometimes unable to furnish a product that gives a uniform service under identical conditions, or that the user is unable to recognize that conditions are not identical with successive ropes. The tensile strength of the rope as shown by a testing machine, and the tests of individual wires as to strength, ductility, and resistance to repeated bending, are factors not readily translated into the length of service of a construction so complicated as that of a rope. The purchaser may receive with the rope a record of its tensile strength, which usually agrees closely with the maker's catalogue claims, but of which he makes practically no use in determining the life and usefulness of the rope. The development of a method of testing ropes that shall more nearly approximate the working conditions of the rope is desired, and, when this is available, it will aid in interpreting the simpler tests now in use. The present method of condemning a hoisting rope is entirely a matter of judgment and lacks a rational basis.

It is rare to find a man intrusted with the care of hoisting ropes who can give such directions for condemning a rope that others can safely follow his directions, and although the general tendency of inspectors is probably to err on the side of safety, still an occasional unfortunate judgment leads to disaster.

Some more certain method is needed than visual inspection and the judgment of one man more or less skilled.

Inspectors in the same general field, in forming their judgment of the condition of a rope, place varying weight on such factors as length of service, regardless of the appearance of the rope; the tonnage lifted, regardless of the time of service; the ton-miles of service of the rope; the number of visible broken wires in some definite length of strand or rope; the reduction of diameter; the amount of stretch; the decreasing strength as shown by tensile tests of pieces removed from the end near the skip; an estimate of reduction of area due to wear; corrosion and lack of lubrication; and a bending test of individual wires removed from the rope near the skip end. No one inspector is keeping all these data. In practical rope inspection, is it possible with present knowledge to formulate a standard method that shall give proper weight to these several factors?

It seems as though it might be possible to record autographically the changing pitch of the twist of a rope while in use, a change that would show the relative extension of each section of the rope during use and thus call attention to an overloaded section.

BASIS FOR TESTS OF GASOLINE LOCOMOTIVES.

By O. P. Hood.

It is known that the gasoline locomotive may be so used in mines as to endanger the lives of miners and operators by the emission of carbon monoxide. In determining the conditions under which such locomotives may be used, it is necessary to consider the engine as a carbon monoxide generator and discover the conditions under which this production is at a maximum. This maximum is the result of several variable conditions. It is suggested that it be assumed that the operator at the mine will stop his machine only as a result of two easily observed conditions, as follows: (1) The slowing down of the engine to one-half or two-thirds normal speed, as shown by a decided change in the sound of the running parts when running without load, and (2) the inability of the locomotive to pull more than one-half the normal load at any speed. Until the locomotive reaches these two limitations, it can be assumed that the machine may be run in a mine. Bad conditions can be produced by over-carburization due to bad carburetor adjustment and by other variable factors, and so long as the carburetor adjustment is in the control of the operator, bad conditions must be assumed as possible within the mine. Such conditions as bad carburization, misfiring, cracked cylinders, excessive lubrication, lack of compression, etc., as shall be found to give only one-half the normal power and shall at the same time give a maximum of carbon monoxide, must be considered as possible in the mine and as determining the gas output of the device. Under what conditions of ventilation this quantity of noxious gas may be tolerated so as to bring the percentage of carbon monoxide below 0.1 per cent, a limit suggested by Mr. Burrell, chemist, of the Bureau of Mines, can then be determined.

THE ANALYSIS OF EXPLOSIVES.

By C. G. Storm.

PERMISSIBLE EXPLOSIVES.

The "permissible" explosives employed in the United States for use in coal mines may, from the standpoint of chemical examination, be grouped as follows: (1) Modified nitroglycerin dynamites, (2) ammonium nitrate explosives, (3) nitrostarch explosives. A fourth group of chlorate explosives may properly be added, but it is to be noted that as yet no chlorate explosives have passed the tests for permissibility.

In the first group the necessary reduction of temperature is produced by one or more of the following means:

1. The addition of an excess of carbonaceous material in quantity sufficient to largely reduce the amount of carbon dioxide evolved on explosion. Wood pulp, wheat flour, and corn meal are the most commonly employed materials of this type.
2. The addition of free water. Amounts as high as 12 per cent have been successfully used.
3. The addition of salts containing water of crystallization; for example, alum, magnesium sulphate, aluminum sulphate, or gypsum, in quantities from a few per cent to as much as 35 per cent.
4. The addition of volatile salts, such as sodium chloride or ammonium salts, in proportions up to 20 per cent.
5. The addition of inert substances, such as clay, sand, powdered slate, or kieselguhr (infusorial earth).

In most of the ammonium nitrate group of explosives there is some nitroglycerin or nitrotoluenes or both, together with some carbonaceous material. These additions to the ammonium nitrate materially raise the low explosion temperature of the latter, so that some of the means mentioned above must be employed for reducing the temperature. Thus, we have ammonium nitrate explosives containing sodium chloride, ammonium sulphate, or some other volatile salt, or containing some salt holding water of crystallization.

The nitrostarch explosives depend for low temperature upon either a proper proportioning of nitrostarch, sodium, and ammonium nitrates, and carbonaceous matter, or on the addition of large proportions of inert nonvolatile substances; for example, as much as 45 per cent of calcium carbonate.

METHODS OF ANALYZING EXPLOSIVES.

With such a variety of means for reducing the temperature of explosion, it is apparent that the method of analysis employed must in each case be specially adapted to the particular explosive under examination. The general scheme of analysis employed in the explosives chemical laboratory of the Bureau of Mines depends on a separation of the ingredients of the explosive into groups according to their solubility, followed by the separation or determination of each group constituent by some appropriate method. Thus, in the great majority of analyses, the explosive is extracted with ether, then with water, and finally with dilute hydrochloric acid. The ether removes completely nitroglycerin, nitrosubstitution compounds, oily materials, resins, etc., the water dissolves the nitrates and other water-soluble salts, and the acid removes the calcium or magnesium carbonate or other water-insoluble antacid substances. The residue left after each extraction is dried and weighed, the loss of weight indicating the total weight of extracted material.

The components of the final insoluble residue are generally recognized by microscopic examination. Starch, if present, is determined by acid hydrolysis The insoluble residue is finally ignited and an examination made of any mineral residue other than the natural ash of the wood pulp, etc.* Each component of the different extracts is then determined, a complete qualitative examination having previously indicated the substances in each solution.

The above general outline of a method of separation will, of course, not apply in all cases. For example, the presence of nitrocellulose, nitrostarch, or sulphur necessitates the use of additional solvents.

Moisture in the explosive is determined by desiccating a sample of 2 or 3 grams over concentrated sulphuric acid, the sample being spread uniformly over the surface of a 3-inch watch glass. In general, the loss of weight after three days' desiccation is regarded as moisture, experiments having shown that there is a continual loss of nitrogylcerin during desiccation, this loss being a function of the temperature and duration of desiccation. A sample of ordinary nitroglycerin dynamite containing approximately 1 per cent moisture has shown a gradual loss on desiccation over sulphuric acid at a constant temperature of 33° C. for a period of 183 days, the total loss at that time being 7.55 per cent.

When salts containing water of crystallization are present, this method does not of course give even approximate results, and as from most of such explosives it is impossible to determine separately the moisture and the water of crystallization or even to completely remove the total water present without some alteration of other constituents, the moisture is determined by difference; that is, the sum of all constituents directly determined is deducted from 100 per cent and the difference assumed to be the moisture content. This method is unsatisfactory, as any accumulated errors in all other determinations are thrown on the moisture determination.

Among other details deserving special note in this brief outline may be mentioned the determination of mixtures of nitroglycerin and nitrosubstitution products in the ether extract. Experiments have shown that in mixtures of nitroglycerin and dinitrotoluenes or trinitrotoluenes, the nitroglycerin can be accurately determined by means of the Lunge nitrometer, but that the presence of mononitrotoluene introduces serious errors. The monocompound is quantitatively converted in the nitrometer to the dicompound by the nitric acid liberated from the nitroglycerin during the reaction, thus giving a low result for the determination of the nitroglycerin. To be exact, 1 gram of mononitrotoluene takes up completely the nitrogen from 0.5530 gram of nitroglycerin. The experiments mentioned have not as yet been extended to nitrocompounds other than the nitrotoluenes.

Other researches have shown that an approximate separation of nitroglycerin from nitrosubstitution compounds is possible by the use of a combination solvent, as dilute acetic acid (65 per cent) and carbon bisulphide. These two solvents do not mix, but the nitroglycerin is almost completely absorbed by the acetic acid, and a fairly definite proportion of the nitrosubstitution compound is recovered from the carbon bisulphide. This definite fraction is previously determined by experiments on known mixtures. When properly performed the process gives fairly accurate results.[a]

In the analysis of some chlorate explosives it has been found necessary to determine a nitrate and chlorate in the same aqueous solution. The determination was satisfactorily effected by reducing the chlorate to chloride by passing a current of SO_2, heating to expel most of the excess SO_2, and oxidizing the remainder with hydrogen peroxide solution, determining the chloride as silver chloride in one part of the solution and the nitrate gravimetrically in another part by means of the "nitron" method of Bush.

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[a] Improvements in this method are described in a paper by A. L. Hyde - The quantitative separation of nitrosubstitution compounds from nitroglycerin: Jour. Amer. Chem. Soc., vol. 35, 1913, pp. 1173-1182.

DETONATORS AND ELECTRIC DETONATORS.

The detonating composition generally employed in the United States consists of a mixture of approximately 90 per cent mercury fulminate and 10 per cent $KClO_3$. Electric detonators contain a priming charge of either the same composition as the main charge or of fulminate alone, or of long-fiber guncotton. The priming charge is placed loosely on top of the compressed charge and surrounds the bridge wire.

Such a composition is analyzed by completely transferring the dry charge to a filtering crucible, moistening it with alcohol to thoroughly wet it, and extracting the chlorate with water. The small amount of fulminate dissolved by the water is determined as HgS by precipitation with H_2S, and in the filtrate from the HgS the $KClO_3$ may be determined directly by double evaporation with nitric and sulphuric acids, weighing as K_2SO_4 the final residue after ignition.

No trinitrotoluene detonators are as yet manufactured in this country.

PRODUCTS OF COMBUSTION OF EXPLOSIVES.

The products resulting from explosion are determined by collecting the gases, solids, and liquids from the Bichel gage after 200 grams of the explosive in its original wrapper has been fired. The gage is evacuated before the firing, and electric ignition is used. One-half hour is allowed for the products to cool before they are collected.

The gases are sampled by collecting over mercury a differential sample representing practically the entire gas volume, and the solid and liquid residues are removed as completely as possible from the gage.

The analysis of the gas sample is made with the Hempel apparatus by the usual methods. For ordinary purposes, no correction is made for the gases resulting from the electric detonator, as their volumes would be practically negligible in comparison with the volume of gases from 200 grams of explosive. No attempt is made to obtain more than a qualitative analysis of the solid products because of the fact that they are largely contaminated by iron and copper salts resulting from the detonator legs, the shell, and the wire tripod upon which the cartridge is placed. Owing to the difficulty of collection, only about 95 per cent of the total products of explosion are recovered, the loss being largely solids and liquid.

No attempt has been made in this brief presentation of methods to even enumerate all of the many special details of interest, including the numerous difficulties arising in analysis.

REQUIREMENTS AND TESTS FOR EXPLOSIVES USED IN MINES AND QUARRIES.[a]

By CLARENCE HALL.

COAL-MINING EXPLOSIVES.

The methods followed at all stations that are testing coal-mining explosives were devised in order to classify explosives in regard to their suitability for use in coal mines. The testing galleries vary in size, form, and material. The cross section of some of the galleries is elliptical; that of others is circular. At all stations, except in Austria, steel cannons are used at the galleries for firing charges of explosives, the bore of the cannon varying from 40 to 55 mm.

Natural gas is used at some stations and artifical substitutes at others. Most of the Continental stations use coal dust. Either because of differences in the retardation of ignition of the gas mixtures, or because of other conditions at the different galleries, the limit charge established for a given explosive is by no means the same at different stations. In some cases there is even a marked dissimilarity of results.

The use of stemming at foreign stations and at the Pittsburgh testing station has been found to raise the limit charge considerably, especially of explosives of the ammonium nitrate class.

Air spacing inside the bore of the cannon, as practiced at most of the Continental stations, is said to extend the apparent border line of safety. However, it may be stated that the results obtained with different charging densities at the Pittsburgh testing station do not indicate that the border line of safety is necessarily extended with low charging densities; on the other hand some explosives have passed a limit charge of 680 grams when loaded with the charging density of a charge that would not pass with this amount when loaded air spaced.

Many of the ammonium nitrate explosives give low limit charges when tested by the suspended test as practiced in Austria, probably for the reason that as the explosive is not confined the flame temperature is more prolonged.

It has been shown by some of the German stations that many "permitted" explosives are less safe when tested with coal dust alone. It may be mentioned that coal-dust tests were subsequently repeated at other stations, but the results obtained at the German stations were not always reproduced, especially at the stations using pure artificial or natural gas. This may be explained, perhaps, by the fact that the pit gases used at many of the stations are not of uniform composition and contain more or less inert gases.

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[a] See Hall, Clarence, and Howell, S. P., The selection of explosives used in engineering and mining operations: Bull. 48, Bureau of Mines, 1913, 50 pp., 3 pls., 7 figs.; Tests of permissible explosives: Bull. 66, Bureau of Mines, 1913, 313 pp., 1 pl., 6 figs.

There is sometimes a small percentage of carbon dioxide and other inert gases present in the pit gases used abroad, the percentage depending on the efficiency and condition of the scrubbers at the time the gases are drawn through. Assuming that the percentage of these inert gases is kept within low limits, there may be at times sufficient inert gases present to have a retarding effect on the ignition. Tests made at the Pittsburgh testing station indicate that a mixture of gas and air containing 8.1 per cent of methane and ethane, 14.2 per cent of carbon dioxide, and 77.7 per cent of air is practically nonexplosive when ignition is attempted with an electric igniter containing 5 grains of black powder. It has also been found that the ignition of gas mixtures is materially retarded with a carbon dioxide content of 11.7 per cent, and even when ignited the mixtures burned at a slow rate, accompanied with little if any explosion. For this reason it would be fair to assume that a gas mixture containing a small percentage of carbon dioxide or other inert gases is proportionally less sensitive than a pure gas mixture.

APPARATUS AND METHODS USED AT THE PITTSBURGH TESTING STATION.

The apparatus used and methods followed at the European explosives-testing stations were carefully considered before the United States testing station was erected. Their experience particularly emphasized the importance of a suitable location for the United States testing station, one that would insure a uniform and constant supply of natural gas. Pittsburgh, Pa., was finally selected as a desirable location, and decision was reached to install there a large gallery and a cannon of practically the same size as that adopted by the Continental stations. In addition, a ballistic pendulum, similar to the one used at the English station, and the German type of apparatus for testing the physical and chemical characteristics of explosives were installed. It was decided that all explosives should be required to pass stemmed tests in the presence of highly inflammable gas mixtures, according to the English practice, and to pass supplemental tests made in the presence of coal dust alone; also, that all explosives should be required to pass such unstemmed tests as had been universally adopted by the Government testing stations on the Continent.

TESTS CONDUCTED.

After the gallery and other apparatus had been calibrated several types of so-called safety powders manufactured in the United States were tested in the gallery. Many of them passed when tests were made according to the English practice, but none of them passed the test of being fired without stemming in the presence of a mixture of gas and air containing 8 per cent of gas. Further tests were made by firing unstemmed shots separately with three explosives, namely, black blasting powder, one of the so-called safety powders of the nitroglycerin class, and one of the ammonium nitrate class, in the presence of mixtures of gas and air in different percentages.

The results of tests indicated that the highest limit charge that could be established with unstemmed shots when fired in the presence of a mixture of gas and air containing 8 per cent of methane and ethane was only 100 grams.

During a visit to the foreign stations the writer witnessed a test made in the presence of an 8 per cent mixture of methane and air, with a "permitted" explosive somewhat similar in composition to the explosive of the nitroglycerin class mentioned above. No ignition occurred when 880 grams of the explosive was fired without stemming, but under the conditions of testing at the Pittsburgh station an ignition occurred when 150 grams was used.

Four different samples, each representing a different class of explosive, were procured through one of the foreign testing stations for comparative tests. They were said to represent the best types of explosives used abroad in coal mines, one of them having the same composition as an explosive that had passed the Continental tests with a limit charge of 880 grams plus. Like the domestic explosives previously tested, they passed the test of being fired with stemming, but the highest limit charge established with unstemmed shots in the presence of mixtures of air and 8 per cent of methane and ethane was only 50 grams. It was found after repeated tests that this limit charge could not be increased even when loaded under special charging densities.

The tabulation following includes the results of tests made at the Pittsburgh testing station on the four explosives mentioned above and seven other foreign explosives recently received. The method of conducting the tests was approximately the same as that employed at the foreign testing stations.

Results of tests of 11 foreign explosives.

		Composition of explosive.														Gaseous products.								Results[b] of gallery tests.					
Designation of explosive.	Office key No.	Moisture.	Nitroglycerin.	Ammonium nitrate.	Nitrotoluene.	Sodium nitrate.	Potassium nitrate.	Wood pulp.	Starch.	Potassium chlorate.	Rosin.	Sulphur.	Miscellaneous components.[a]	Detonator used.	Heat test (min.).	Carbon dioxide.	Oxygen.	Hydrogen.	Methane.	Nitrogen.	Carbon monoxide.	Hydrogen sulphide.	Volume of CO from 680 grams of explosive.	Test No. 1.	Test No. 3.	Test No. 4.	Result of friction test.	Result of Woolwich test (mod.).	Result of Belgian test (mod.).
		Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per ct.*				*Per cent.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Liters*						
A	B-426	0.46	.....	.....	.....	.....	65.31	.....	8.73	.....	.....	2.63	Charcoal, 19.52; paraffin, 3.35.	Electric igniter.	....	13.9	0.0	23.6	2.8	16.8	41.8	1.1	[c]130.0	F.	P.	F.	....	P.	0
B	M-421	.17	.....	89.08	[g]6.56	.....	.....	.....	.....	.....	.....	....	Mononitronaphthalene, 4.19.	No. 7....	....	26.7	3.2	.0	.0	70.1	.0	.0	[d].0	P.	P.	F.	....	P.	0
C	D-422	.60	8.37	83.14	3.24	.....	.....	3.65	.....	.....	.....	....	Nitrocellulose, 1.00.......	No. 6....	....	30.8	1.5	.0	.0	67.7	.0	.0	[d].0	F.	P.	F.	....	P.	0
D	D-429	4.05	24.92	.....	.....	.....	25.37	34.60	6.64	.....	.....	....	Barium nitrate, 4.42.....	No. 6....	....	18.9	.0	29.1	5.8	9.9	36.3	.0	157.0	P.	P.	P.	....	P.	50
E	M-1404	.17	.....	....	[e] 17.85	....	.....	.....	.....	75.36	.....	....	Castor oil, 5.32; mononitronaphthalene, 1.3.	No. 6....	60	34.1	.0	15.6	.5	8.3	41.5	.0	99.1	P.	P.	F.	F.	P.	0
F	M-1405	.10	.....	.....	.....	.....	.....	.....	.....	75.27	24.63	....	Trace of starch and nitrostarch (not more than 0.2 per cent).	No. 6....	7	23.4	.0	23.9	.6	4.7	47.4	.0	127.3	P.	P.	[f]	F.	P.	0
G	D-1406	.52	8.13	82.11	3.57	.....	.....	4.30	.....	.....	.....	....	Castor oil, 0.81; nitrocellulose, 0.56.	No. 6....	60	28.7	.0	11.0	.8	53.6	5.9	.0	17.2	P.	P.	F.	P.	P.	0
H	M-1407	.18	.....	88.17	[g]8.13	.....	.....	.....	.....	.....	.....	....	Mononitronaphthalene, 3.52.	No. 7....	11	27.4	.0	9.1	.7	57.3	5.5	.0	16.0	F.	P.	[f]	P.	P.	0
I	M-1408	.06	.....	91.43	.....	.....	3.50	.....	.....	.....	5.01	....	....	No. 7....	21	25.5	.0	9.6	1.1	60.8	3.0	.0	8.1	P.	P.	P.	P.	P.	0
J	M-1409	.45	.....	90.50	[g]4.82	.....	.....	.....	.....	.....	.....	....	Flour, 4.23...............	No. 7....	26	30.0	.1	.0	.2	69.7	.0	.0	.0	P.	P.	P.	P.	P.	0
K	D-1263	2.69	39.63	.....	.....	10.63	....	[h] 20.84	21.57	....	.....	....	Barium nitrate, 3.30; sodium carbonate, 0.49; aluminum oxide, 0.85.	No. 6....	....	14.6	.0	24.9	8.6	10.3	41.6	.0	199.4	P.	P.	P.	P.	....	....

[a] Figures represent percentages.

[b] P signifies that explosive passed the test; F that it failed.

[c] Powder repacked and shot in tin-foil wrapper. The 130 liters of CO includes 3.3 liters of H_2S.

[d] Powder repacked and shot in tin-foil wrapper.

[e] Dinitrotoluene.

[f] Incomplete detonation.

[g] Trinitrotoluene.

[h] Includes crude fiber from corn meal.

The results of the tests indicate that all of the foreign explosives, except explosive H, which detonated incompletely, would pass the Woolwich tests but that all would fail to pass the Belgian tests. Only three of the explosives would pass the test requirements requisite for their being placed on the United States permissible list.

It is to be noted that two of the explosives failed in tests 1 and 4. Two explosives failed in test 4 only. One explosive failed in test 1 and detonated incompletely in test 4. Two explosives failed in the friction test and in test 4, and one explosive failed to meet the requirements for poisonous gases.

LIMIT CHARGES WITH VARIOUS GAS PERCENTAGES.

No stemming was used in any of the tests, but tests were made with different loading densities in the bore of the cannon. It is to be noted that the limit charge with all of the foreign explosives tested in the presence of 8, 8½, and 9 per cent gas mixtures was 0 gram in all tests, even though the explosive was loaded with different charging densities. (See figs. 1 to 4.)

When these explosives were fired in the presence of 10 per cent gas the highest limit charge established was 50 grams (explosive E). The conditions of loading in the cannon seemingly were not a factor in the limit charge established.

When the explosives were fired in the presence of 7 per cent gas the limit sharge established was 0 gram for samples G and J. The limit charge established for sample I was 0 gram when the explosive was loaded with a density of 1, and 150 grams when loaded with air spacing. Sample E also gave a higher limit charge (100 grams) when loaded with air spacing than when loaded with a density of 1 (50 grams).

When the explosives were fired in the presence of 6 per cent gas the limit charge established with explosive G was 0 gram, and 1,000 grams with explosive I, the results being the same irrespective of the charging density. Explosive J gave a limit charge of 50 grams when air spacing was used, and of 0 gram when loaded with a charging density of 1. The limit charge established for sample E was 150 grams when loaded with air spacing, and also when loaded with a charging density of 1.

When the explosives were fired in the presence of 5 per cent gas, 50 grams was established as the limit charge for explosive G, and 1,000 grams for explosive J. Different charging densities did not change the results. Six hundred grams was established as the limit charge for explosive E when loaded with a density of 1, and 400 grams when loaded with air spacing.

The results of the tests indicated that the natural gas used at the Pittsburgh testing station is more sensitive than the pit gas used

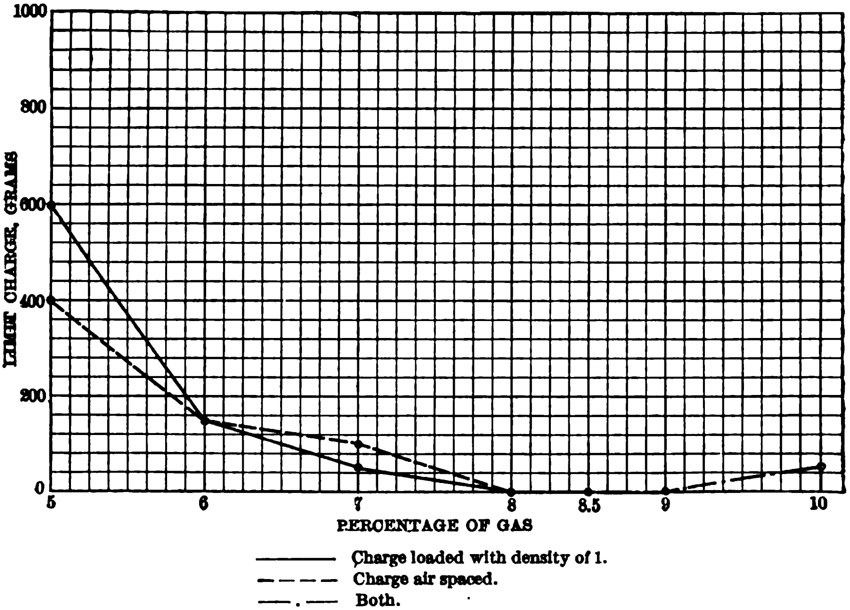

FIGURE 1.—Foreign explosive E. Limit charge established with various gas percentages. Composition of explosive: Moisture, 0.17; dinitrotoluene, 17.85; potassium chlorate, 75.36; castor oil, 5.32; mononitronaphthalene, 1.30; total 100.

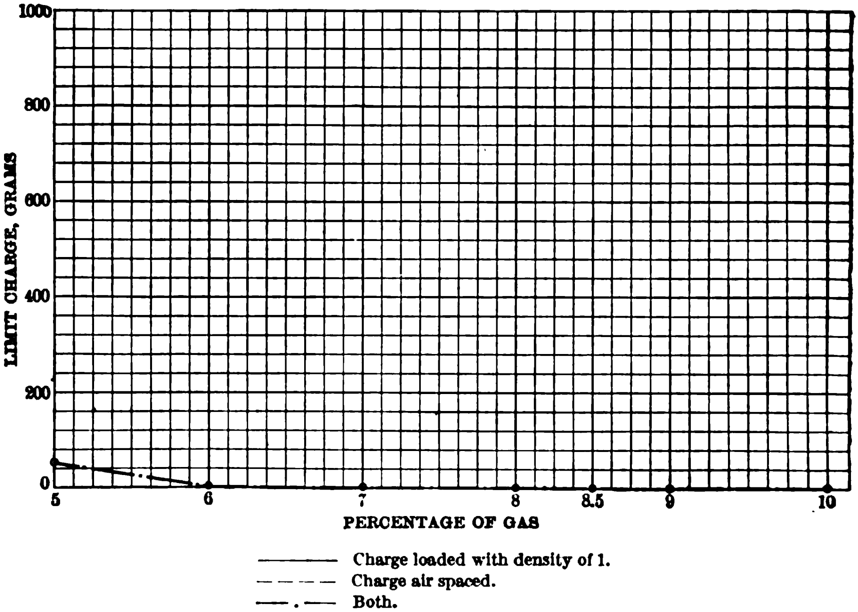

FIG. 2.—Foreign explosive G. Limit charge established with various gas percentages. Composition of explosive: Moisture, 0.52; nitroglycerin, 8.13; ammonium nitrate, 82.11; nitrotoluene, 3.57; wood pulp, 4.30; castor oil, 0.81; nitrocellulose, 0.56; total, 100.

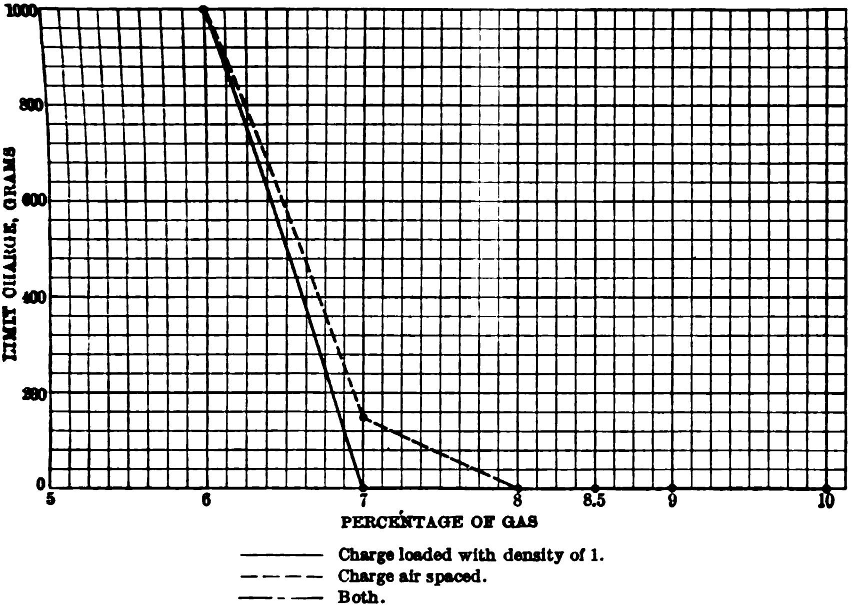

Figure 3.—Foreign explosive I. Limit charge established with various gas percentages. Composition of explosive: Moisture, 0.06; ammonium nitrate, 91.43; potassium nitrate, 3.50; rosin, 5.01; total, 100.

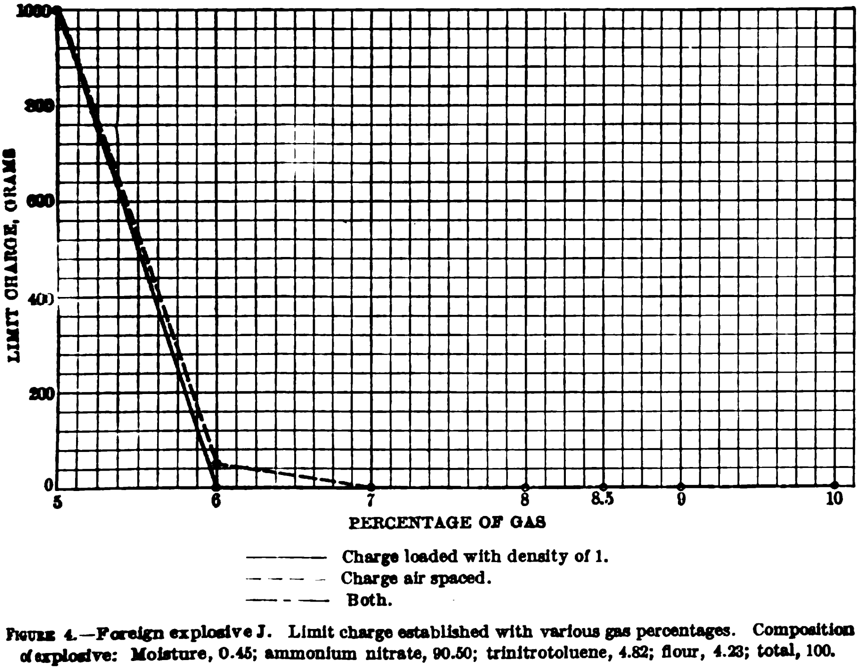

Figure 4.—Foreign explosive J. Limit charge established with various gas percentages. Composition of explosive: Moisture, 0.45; ammonium nitrate, 90.50; trinitrotoluene, 4.82; flour, 4.23; total, 100.

abroad and that it will be necessary to reduce the percentage of gas in order that the results of future tests may be comparable with results obtained in Europe. It was evident that the mixtures of gas and air that contained about 6 per cent of methane and ethane were equivalent to the 8 per cent mixtures abroad, and, further, that all explosives would be excluded if unstemmed shots were fired in the presence of air mixtures containing 8 per cent of methane and ethane. A conference of several of the leading coal operators was called at the Pittsburgh testing station, and the consensus of opinion was that a low percentage gas mixture with coal dust suspended in the air would more nearly represent the dangerous conditions that might exist in a coal mine, and that at least one of the tests should be made under these conditions with unstemmed shots and a charge of not less than 680 grams (1½ pounds).

STANDARDIZING APPARATUS AND METHODS.

The matter of a universal standard of methods and apparatus for testing coal-mining explosives is still an open one. Obviously, cooperation on the part of all stations aiming to unify or standardize apparatus and methods of testing explosives would be beneficial to all concerned, and it is hoped that such will eventually be the outcome.

It would seem that the design and the construction of all testing apparatus could be established on a uniform basis. In fact, the tendency at present to adopt the cylindrical type of steel gallery and steel cannons having a bore 55 mm. in diameter is universal. The unifying of methods of testing does not offer a ready solution.

In Belgium the mines are unusually gaseous and all explosives are required to pass a severe gas test. In recent years explosives have been tested in the presence of coal dust, but the dust used has had a relatively low percentage of volatile matter as compared with that used in the United States. It is contended that, as all the coals in Belgium with a high percentage of volatile matter have been practically exhausted, coal dust similar to that which exists in mines under actual working conditions should be used in the gallery.

In France great stress is put on the requirements as to the gaseous products of combustion of explosives. The nitroglycerin class of explosives is practically eliminated from use in French coal mines for the reason that they produce carbon monoxide on detonation. The writer's experience with permissible explosives of this class is that when they are properly used in coal mines of the United States in quantities less than 1½ pounds per shot the amount of poisonous gases produced is so small that it is of little, if any, consequence, especially when the ventilation current is brought directly to the working face.

In England stemmed charges only are used for determining the permissibility of explosives for use in coal mines. This requirement would not always represent the conditions of mining in the United States. Miners have been known to fire permissible explosives in a drill hole with little, if any, stemming, although the practice is reprehensible. For this reason the Bureau of Mines requires an explosive to show satisfactory performance in an unstemmed shot before the explosive will be designated permissible. The bureau requires all explosives to meet practically all the European requirements in order to be placed on the permissible list; that is, satisfactory performance in shots in the presence of gas alone and of coal dust alone, and in unstemmed shots in mixtures of gas and coal dust.

PROPOSED REQUIREMENTS FOR PERMISSIBLE EXPLOSIVES IN COAL MINES.

SENSITIVENESS TO DETONATION.

It has been observed that many permissible explosives when stored lose sensitiveness to detonation and in some cases fail to detonate when fired by a proper detonator. This may be explained, perhaps, by the fact that the nitroglycerin contained in the explosive becomes more completely absorbed by the other ingredients; also by the absorption of water by the hygroscopic salts.

It was with this in view that one of the conditions specified by the Bureau of Mines for testing explosives was that the manufacturer be required to deliver to the experiment station three weeks prior to the date set for test 100 pounds of each explosive that he desired to have tested.

It is believed that several explosives have failed to pass the test requirements of the bureau that otherwise would have passed if they had not been 4 to 6 weeks old when tested. Supplemental tests have been made with all explosives on the permissible list, and it has been found that many of the ammonium-nitrate class of explosives after six months' storage at the Pittsburgh experiment station have either failed to detonate or detonated incompletely when retested. A few of the ammonium-nitrate explosives on being retested have shown signs of deterioration and have detonated incompletely after storage for four months. It is worthy of note that 2 of the 11 foreign explosives that were tested by the bureau failed to detonate completely when tested four months after they had been received.

It has been suggested that the bureau should consider the elimination from the permissible list of those explosives that become insensitive to detonation when stored under favorable conditions for a period of four months. I would like to hear from the visiting members of this conference as to whether any such requirement is in force abroad, and whether it is, in their opinion, advisable to have a requirement of this kind.

GASEOUS PRODUCTS.

The list of permissible explosives issued by the bureau January 1, 1911, contained a few explosives that on detonation in the Bichel gage developed large quantities of carbon monoxide. These explosives were later tested in coal mines. When the usual charge was fired in rooms, no appreciable amount of carbon monoxide was detected in samples of the mine air taken at the face immediately after the shot. However, when tests were made in narrow entries, ahead of the air, all mine-air samples taken immediately after charges of 680 grams had been fired showed the presence of carbon monoxide, one sample containing as much as 0.11 per cent. It was believed that these explosives, used under such conditions, would so vitiate the air near the face as to be harmful to the health of those directly connected with the work.

Other permissible explosives tested on detonation evolved less quantities of carbon monoxide in the gage tests. It was found, as a result of tests made in coal mines, that an explosive that on detonation in the gage evolved less than 158 liters of permanent poisonous gases from a charge of 680 grams would not, when used in coal mines, vitiate the mine air in narrow entries even when there was no circulation of air near the face. The highest percentage of carbon monoxide found in any of the mine-air samples taken immediately after the shot was 0.02 per cent.

When the later permissible list was issued (January 1, 1912) a few explosives were eliminated and a new requirement was adopted, namely, all explosives that, on detonation of a 680-gram charge, evolved 158 liters or more of permanent poisonous gases in gage tests shall not be considered permissible.

It is the opinion of the writer that the limitations that have been placed on the amount of poisonous gases evolved on detonation of explosives for use in coal mines is adequate, but I would like to hear an expression of opinion from the visiting members concerning this subject.

METAL-MINING EXPLOSIVES.

For driving tunnels and sinking shafts, and for metalliferous mining in this country, gelatin dynamites have been found well adapted and are used to a large extent. They produce on detonation smaller quantities of poisonous gases than any other class of explosives tested by the bureau. In work of this kind, the ventilation is usually defective; accordingly the gaseous product evolved on detonation of explosives is one of the most important considerations. The investigations that have been made by the bureau show that it is possible to manufacture gelatin dynamite having a slight oxygen excess sufficient to insure the complete oxidation of all combustible materials, including paper wrappers.

Some of the manufacturers of explosives have taken advantage of the suggestions made by the bureau and are supplying the trade with the improved gelatin dynamites. In order to insure the use of better grades of explosives in metal mines, it will be necessary to publish, for the information of users of explosives, the nature and character of the many explosives offered for sale. Now that the bureau has been authorized by law to investigate the mining and the treatment of ores and other mineral substances, it will be possible to determine the permissibility of explosives for use in all mining and quarrying operations, and to publish a list of those explosives that pass the requirements satisfactorily.

The following is a tentative outline of the proposed requirements and I will appreciate any criticism or suggestion offered by this conference. The recommendation of specifications by this conference will be of material assistance to the director of the bureau in considering this subject.

The conditions under which explosives for use in metal mines and quarries will be tested in respect to priority of testing, method of deliveries, etc., will doubtless be the same as those under which explosives are tested for their permissibility in coal mines.

Proposed requirements of explosives for metal mines.

1. The explosives must be in a stable condition and all chemical and physical tests must not show any unfavorable results.

2. The explosives, on detonation, must not evolve carbon monoxide, hydrogen sulphide, nitrogen oxides, or other permanent poisonous gases, in quantities that may be considered harmful to the health of miners, when fired in 200-gram charges in the pressure gage.

3. Electric detonators will be used in the tests as recommended by the manufacturers, provided the grade recommended completely detonates or explodes the charge.

4. The explosives must pass pendulum friction tests in which the steel shoe is faced with wood fiber.

5. The propulsive and disruptive effect developed by each explosive will be determined and the results published, in order to serve as a useful guide in judging the efficiency of the explosive.

6. After an explosive shall have passed the above requirements it will be considered permissible for use only when used under the following conditions:

(a) That the explosive is in all respects similar to the sample submitted by the manufacturer for test.

(b) That detonators, or electric detonators, are used of not less efficiency than those prescribed, namely, those consisting by weight of 90 parts of mercury fulminate and 10 parts of potassium chlorate (or their equivalents).

(c) That the explosive, if frozen, shall be thoroughly thawed in a safe and suitable manner before use.

PROPOSED REQUIREMENTS OF EXPLOSIVES FOR QUARRIES.

Practically every class and grade of explosives are used in the United States in open work, such as quarries and railroad excavations. In work of this kind the efficiency of explosives is the most important consideration. The amount of poisonous gases produced on detonation of an explosive is of little importance, and the liability to cause ignition of gas or dust does not need consideration.

Tests will be made to determine the stability of each explosive and other tests that concern safety in the handling and use of such explosives. Information will also be published that will show the practical value of these explosives as well as offering the best means for selecting explosives suitable to meet the varying conditions of quarrying, etc.

PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF MINING.

Limited editions of the following Bureau of Mines publications are available for free distribution. Copies may be obtained by applying to the Director, Bureau of Mines, Washington, D. C. Requests for all publications can not be granted.

BULLETIN 10. The use of permissible explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls., 4 figs.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls., 28 figs. Reprint of United States Geological Survey Bulletin 425.

BULLETIN 42. The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs.

BULLETIN 44. First national mine-safety demonstration, Pittsburgh, Pa., October 30 and 31, 1911, by H. M. Wilson and A. H. Fay, with a chapter on the explosion at the experimental mine, by G. S. Rice. 1912. 75 pp., 7 pls., 4 figs.

BULLETIN 45. Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 46. An investigation of explosion-proof mine motors, by H. H. Clark. 1912. 44 pp., 6 pls., 14 figs.

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Bulletin 83

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, Director

THE HUMIDITY OF MINE AIR

WITH ESPECIAL REFERENCE TO COAL MINES IN ILLINOIS

BY

R. Y. WILLIAMS

ILLINOIS COAL-MINING INVESTIGATIONS
COOPERATIVE AGREEMENT

[This report was prepared under a cooperative agreement with the Illinois State Geological Survey and the department of mining engineering of the University of Illinois]

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THE HUMIDITY OF MINE AIR.

By R. Y. Williams.

INTRODUCTION.

The investigative work described in this report was undertaken by the Bureau of Mines under a cooperative agreement with the department of mining engineering of the University of Illinois and the Illinois State Geological Survey.[a] The purpose of this agreement was to provide for a comprehensive investigation of coal-mining conditions in the State, with especial reference to the possibility of lessening waste in the mining and preparation of coal and of reducing the dangers to which miners are exposed. The Bureau of Mines paid particular attention to those phases of the investigation that were of general interest from their bearing on safety and efficiency in the coal-mining industry as a whole. Thus, it studied the occurrence of gas (methane) in the mines of the southern part of the State,[b] tested the inflammability of the coal dust from many mines, examined the efficiency of the control of ventilating currents, and investigated the factors affecting the humidity of mine air with respect to their bearing on the occurrence of mine explosions.

Following the establishment of the fact that under certain conditions dry bituminous coal dust is inflammable, there arose the problem of finding efficient methods of preventing or limiting coal-dust explosions.

Experiments in the gas-and-dust gallery at the Pittsburgh testing station of the Bureau of Mines verified the fact that certain dusts that explode violently when dry are rendered inert by proper humidification of the atmosphere within the gallery, and the importance of humidity as a factor in limiting the inflammability of coal dust in a mine is demonstrated by the fact that there has never been in Illinois a dust explosion during the summer months, whereas the records show a number of dust explosions during cold weather. It is easy to see that the warm moisture-laden air that enters a mine in the summer must deposit moisture on cooling, whereas the cold and relatively dry air that enters in winter must take up moisture as it becomes heated in traveling through the mine.

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[a] For a detailed account of the terms of this agreement, see Preliminary report on organization and method of investigations: Illinois Coal-Mining Investigations, Cooperative Agreement, 1913, 71 pp.

[b] Darton, N. H., Occurrence of explosive gases in coal mines: Bull. 72, Bureau of Mines, 1914. In press.

A study of humidity in the coal mines of the State was begun in January, 1912, by the Illinois coal-mining investigations previously mentioned, in cooperation with the United States Weather Bureau and the operators of 20 typical mines in the State. In this study records were made that show the humidity changes both underground and on the surface during the seasonal changes of a complete year. These records afford a basis for generalizations as to humidity. The records, with pertinent discussion, are presented in this report for the purpose of supplying information that will assist operators and miners in making mines safe from coal-dust explosions.

A separate report prepared in connection with the cooperative investigation will discuss the character of the coal dusts encountered in the different mining districts in Illinois, and which dusts are the most dangerous.

In the collection and compilation of the data presented on the following pages the author has received suggestions and helpful criticism from his associates, to whom, and especially to George S. Rice, chief mining engineer of the Bureau of Mines, and L. A. Scholl, jr., junior chemist, he hereby makes acknowledgment of his indebtedness. Thanks are due S. T. Wallage for the calculations necessary in preparing Table 6.

The work of Carl Scholz, H. N. Eavenson, and Frank Haas has been reviewed in pages 14 to 16. The author also wishes to express his indebtedness for the hearty cooperation of the United States Weather Bureau and the operators of the 20 selected mines.

FUNDAMENTAL CONSIDERATIONS.

DEFINITION OF TERMS.

To insure a clear understanding of the discussion and records in this report, definitions of certain terms used are presented below:

The temperature of mine air or of the outside atmosphere signifies the degree of sensible heat or cold.

Absolute humidity signifies the total quantity of water vapor present in a given volume of air.

Relative humidity is the ratio of the quantity of vapor present to the quantity necessary to saturate, at the given temperature and pressure, the space occupied by the air.

Dew point is that temperature of the air at which the water vapor present begins to condense into visible moisture—that is, the temperature of the air when the relative humidity is 100 per cent.

Evaporation is the transformation of water into aqueous vapor.

Latent heat of vaporization is the quantity of heat absorbed during evaporation. This quantity varies slightly with the temperature at which evaporation takes place.

The specific heat of a substance is the ratio of the heat required to raise the temperature of a certain weight of the substance 1 degree to that required to raise the temperature of the same weight of water 1 degree (from 62° to 63° F.).

A thermometer is an instrument for measuring temperature and is based on the fact that different substances expand, or contract, differently under the same changes in temperature. For example, mercury within a glass tube expands more on a rise of temperature than does the glass tube, and the difference of expansion is utilized as the measure of temperature. There are several scales or manners of graduating thermometers; but the one adopted in this report is the Fahrenheit, in which the freezing point of water is at 32° and the boiling point at 212°.

A thermograph is an instrument that automatically records variations in atmospheric temperature. An arm carrying a pen is connected with a spiral lamina that expands or contracts as the temperature rises or falls. The pen, fed with special ink, makes a mark on a chart that is revolved by clockwork. Each chart provides a record for one week.

A psychrometer or hygrometer is an instrument for measuring the quantity of aqueous vapor in the air. One type involves a determination of the temperature of evaporation and consists of two similar thermometers, one called the dry bulb and the other the wet bulb. A thin muslin sack is fitted over the wet-bulb thermometer, and in order to make an observation the sack is thoroughly moistened and both thermometers are placed in a current of air that has a velocity of 15 feet or more per second. If the relative humidity of the air is less than 100 per cent, some of the water on the muslin sack is evaporated, a certain amount of heat is absorbed by evaporation, and the temperature of the wet-bulb thermometer is reduced to a point at which the amount of heat absorbed by evaporation is just equal to that received from the surrounding air. From the readings of the two thermometers, by reference to suitable tables, the relative humidity and the quantity of aqueous vapor per cubic foot of the air may be determined.

There are two kinds of instruments using wet-bulb and dry-bulb thermometers for determining the relative humidity of air—the sling psychrometer and the stationary hygrometer.

The sling psychrometer is used in places where the velocity of the air is low. The instrument must be rapidly whirled through the air in order that the velocity against the wet-bulb thermometer shall approximate 15 feet per second. The standard sling psychrometer designed by the United States Weather Bureau has two thermometers mounted side by side, the wet-bulb thermometer extending below the dry bulb, which are attached by a link to a handle. By means of the handle the instrument can be whirled to obtain a velocity of the wet bulb of about 3 feet per second. This type of psychrometer is unprotected, and when it is swung in a mine it is likely to be broken by striking against the side, roof, or a timber. To guard against such breakage the Bureau of Mines has designed a special form (Pl. I), in which the thermometers are attached to an aluminum frame, with a pivoted handle stiffly connected. Carried in a leather case with shoulder strap this is a convenient instrument for mine use, and if used with ordinary care it is not liable to be broken.

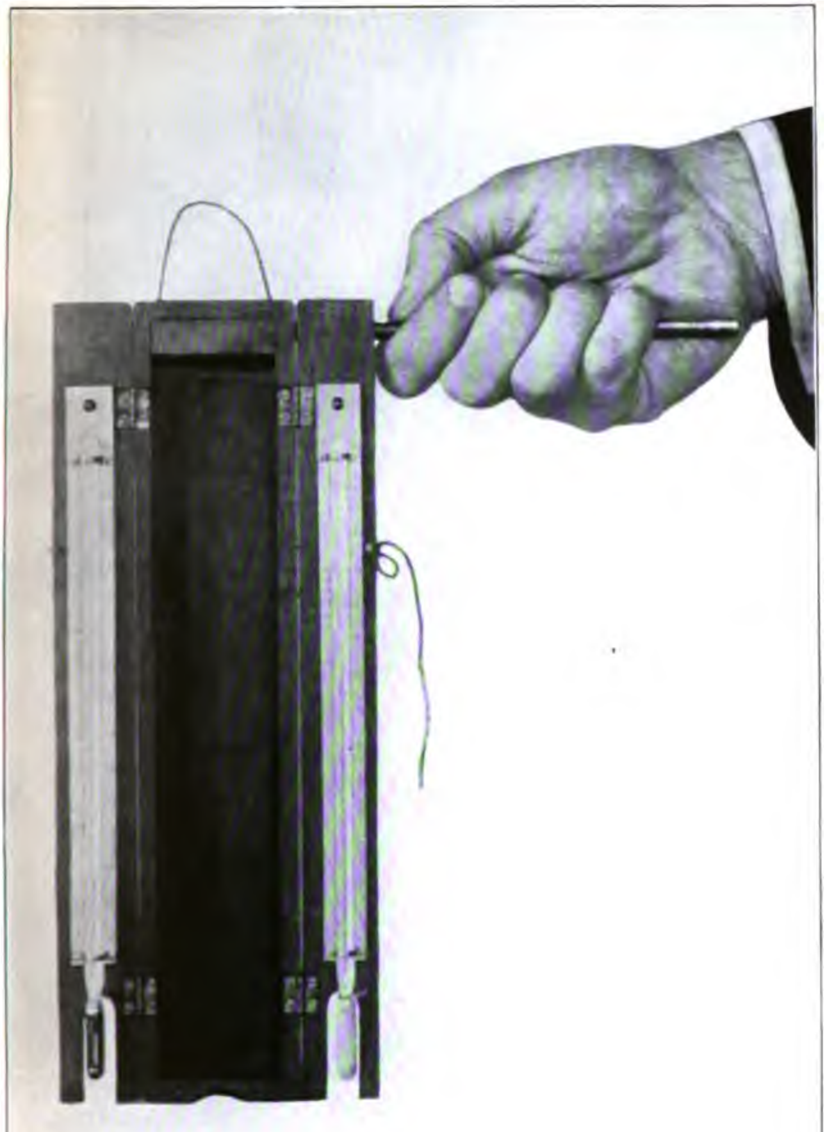
The stationary hygrometer may be used in places where the air is moving at the required velocity of 15 feet or more per second. This form of instrument was used in the cooperative investigations for the underground readings at the mines. The hygrometer (Pl. II) consists of two exposed thermometers 12 inches long, accurately calibrated and graduated to the Fahrenheit scale, and mounted on a silvered brass back which is fastened by a bent support to a wooden frame. The thermometer bulbs extend 2 inches below the mounting to allow free circulation of air around the bulbs and the stems. The wet bulb is covered with a fine muslin bag to which is attached the wicking that extends into a nickeled water cup. Water is thus brought by capillary attraction to the muslin bag. In order to protect the hygrometers from breakage, a shelter (Pl. II) of 24-gage galvanized iron was made. It consisted of two 7-inch cylinders intersecting at right angles. To allow perfect circulation of air around the thermometer bulbs, the horizontal cylinder is open at both ends, each end being protected by a screen of hardware cloth of ¼-inch mesh. In the vertical cylinder there is a similarly protected opening to allow convenient reading of the thermometers.

A hygrograph is an instrument by which the variations in atmospheric humidity are automatically recorded. In appearance, this instrument resembles a thermograph except that an animal hair is substituted for the spiral lamina. The hair expands and contracts with changes in relative humidity, and the movement is recorded on the revolving chart with a pen.

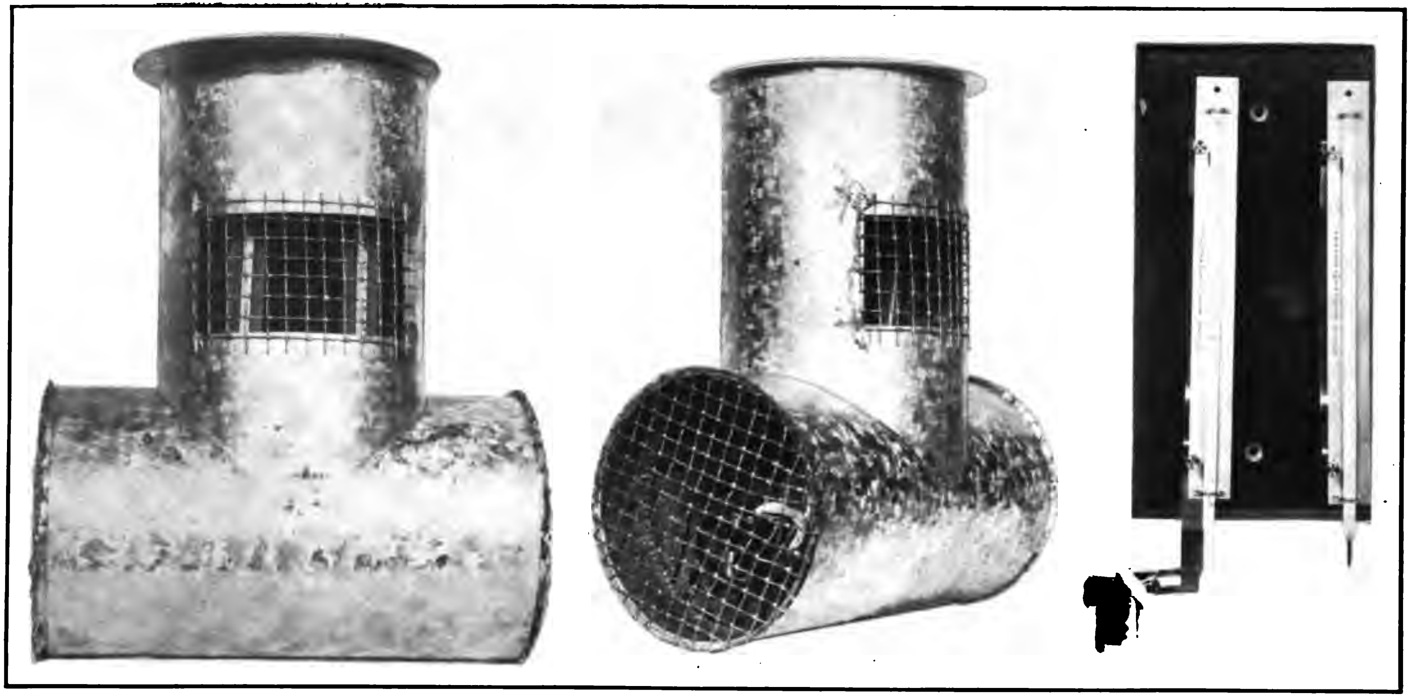
PHYSICAL LAWS CONTROLLING THE HUMIDITY OF MINE AIR.

THE KINETIC THEORY OF FLUIDS.

According to the kinetic theory, gases and liquids are composed of extremely small particles, called molecules. These particles dart about in straight lines until deflected by encounters with other molecules or with the walls of the containing vessel. The average velocity of the molecules increases with the temperature.



PSYCHROMETER DESIGNED BY BUREAU OF MINES.



MINE HYGROMETER AND TWO VIEWS OF HYGROMETER SHELTER.

When liquid water and water vapor are in contact some of the darting molecules of the liquid escape from the surface of the liquid and enter the vapor. Also, some of the molecules of the vapor enter the liquid. The measure of the tendency of a substance to become a gas, or vapor, is its vapor pressure. Different substances have different vapor pressures, and these differences may be shown by the fall of the mercury column in a barometer tube when a small quantity of a liquid is inserted so that its vapor can enter the space above the mercury in the tube.

If the vapor pressure of the liquid be greater than the pressure of the vapor, more molecules leave the liquid than return to it, and evaporation takes place. If the vapor pressure of the liquid be less than that of the vapor, more molecules enter the liquid than leave it, and there is condensation.

For any given temperature there is a definite vapor pressure at which the number of molecules leaving the liquid equals the number of those entering it. This state is defined as physical equilibrium and is independent of the amounts, whether relative or absolute, of either the liquid or the vapor.

To illustrate the effects of varying the relative temperature and humidity of water and water vapor in juxtaposition, the following tabulation has been prepared:

TABLE 1.—*Effects of varying the relative temperature and humidity of water and water vapor in juxtaposition.*

Temperature of water.	Temperature of vapor.	Relative humidity of vapor.	Vapor pressure of water.	Pressure of vapor.	Result.
°F.	*°F.*	*Per cent.*	*Inches of mercury.*	*Inches of mercury.*	
60	30	100	0.517	0.164	Evaporation.
60	60	60	.517	.310	Do.
60	90	85	.517	1.197	Condensation.
60	80	50.6	.517	.517	Neither evaporation nor condensation.

Table 2, below, gives the values of the vapor pressure of water, in inches of mercury, for temperatures ranging from 0° to 212° F.

TABLE 2.—*Vapor pressure of water for temperatures of 0° to 212° F.*[a]

Temperature.	Vapor pressure of water.	Temperature.	Vapor pressure of water.
° F.	*Inches of mercury.*	*° F.*	*Inches of mercury.*
0	0.0383	70	.732
10	.0631	80	1.022
20	.103	90	1.408
30	.164	100	1.916
40	.247	150	7.552
50	.360	200	23.45
60	.517	212	29.92

[a] Marvin, C. F., Psychrometric tables: U. S. Weather Bureau Bulletin 235. 1900.

RATE OF EVAPORATION AND CONDENSATION.

The rate of evaporation and the rate of condensation depend mainly on the difference between the pressure of the vapor and the vapor pressure of the liquid. Unfortunately, there are available no empirical data showing the quantity of water that will change to a unit volume of vapor under a given difference of pressure acting for a unit of time.

The following factors, however, determine in a relative way the rates of evaporation and of condensation, and even the minor factors assume importance in studies of mine air, because of the volume of the ventilating currents and the amount of moisture they carry.

Temperature.—The vapor pressure increases with temperature, hence temperature is the most important factor influencing the rates of evaporation and condensation.

Relative humidity.—The values for the vapor pressure of water given in Table 2 are correct only for a relative humidity of 100 per cent. For any other relative humidity the pressure of the vapor will vary directly with the percentage of saturation. For example, if the temperature is 60° F., and the relative humidity is 80 per cent, the pressure of the vapor will be 0.517 multiplied by 80 per cent, or 0.414 inches.

Volume.—The rate of evaporation is ordinarily independent of the amount, whether relative or absolute, of the water. When water particles are exceedingly small, however, as in exhaust steam, their surface area is great, and the rate of evaporation is correspondingly increased.

Absorption.—All coals absorb moisture. The quantity absorbed depends on the nature of the coal and on local conditions of temperature and relative humidity.

Adsorption.—Solids have the property of condensing and holding moisture on their surfaces. Coal possesses this property to a remarkable degree. If water be brought in contact with a pile of coal dust, a film of water forms on the outer layer of the pile, and it is exceedingly difficult to dampen the dust beneath this surface.

Coal dust.—Fine coal-dust particles provide nuclei on which moisture is readily deposited. In places where there is a total absence of dust, aqueous vapor may become supersaturated; that is, more than normal quantities may be present.

Convection.—Convection implies the transfer of heat by currents. In a mine convection is responsible for the movement of heat from warmer to the cooler portions of the air. It causes an intimate mixing of the air, and thereby facilitates evaporation and condensation.

Conduction.—Conduction implies the transfer of heat by molecular movement. By it heat flows from the zone of condensation or to the zone of evaporation.

WATER VAPOR AS A CATALYTIC AGENT.

Some gases do not explode in an absolutely dry atmosphere,[a] but can be made to explode by a suitable means of ignition if small quantities of water vapor, which acts as a catalytic agent, be added. This fact, however, is relatively unimportant in the study of mine-air humidity, because all mine air carries at least a trace of water vapor.

WATER VAPOR AS AN INERT GAS.

Water vapor, if present in large quantities, retards ignition and the propagation of an explosion by diluting the explosive mixture. Furthermore, before an explosion can occur, the vapor must be raised to the ignition temperature of the gas and on account of the high specific heat of water, considerable heat is thus absorbed.

WEIGHT OF AIR.

The weight of a volume of air varies directly with the pressure and inversely with the temperature. At a pressure of 14.7 pounds per square inch, and at a temperature of 32° F., a cubic foot of pure dry air weighs 0.080728 pound. Table 3 shows the weight of 100,000 cubic feet of pure dry air at temperatures of 0° to 100° F. and with a uniform pressure of 14.7 pounds per square inch.

MOISTURE IN AIR.

Absolute humidity may be expressed as grains of water per cubic foot of air, but because of the large amounts of both air and water involved in studies of mine air it is more convenient to express absolute humidity as gallons of water per 100,000 cubic feet of air.

The amount of moisture in the form of aqueous vapor that may be present in a space occupied by air depends mainly on the temperature of the air. Table 3 gives the weight of saturated aqueous vapor in pounds and the quantity of water in gallons that will exist in a space occupied by 100,000 cubic feet of air when the relative humidity is 100 per cent and the temperature varies from 0° to 100° F.

Table 4 gives the changes in relative humidity that accompany increases in temperature from 0° to 100° F. if the quantity of water vapor remains constant.

The interrelation of the temperature and the relative humidity of mine air is of great importance. Tables 3 and 4 indicate (*a*) that it is not always sufficient for mine safety to have a relative humidity of 100 per cent at the intake of a mine, because if the entering air had a low temperature the aqueous vapor present would suffice for only a low relative humidity at the return; and (*b*) that to insure a proper condition of the intake air both temperature and moisture must be added during the greater part of the year in order to prevent the extraction of large quantities of water from the mine by the ventilating current.

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[a] See Mellor, J. W., Chemical Statics and Dynamics, p. 471.

TABLE 3.—*Physical changes in air due to temperature.*

[Relative humidity, 100 per cent; pressure, 14.7 pounds per square inch.]

Temperature of the air.	Weight of 100,000 cubic feet of pure air.	Weight of water in 100,000 cubic feet of saturated aqueous vapor.	Quantity of water per 100,000 cubic feet of air.
° *F.*	*Pounds.*	*Pounds.*	*Gallons.*
0	8,635.4	6.9	0.823
10	8,459.4	11.1	1.329
20	8,275.5	17.6	2.114
30	8,106.3	27.6	3.312
40	7,943.9	40.7	4.878
50	7,787.9	58.2	6.979
60	7,637.9	82.1	9.848
70	7,493.5	114.0	13.686
80	7,354.6	156.2	18.776
90	7,220.6	211.3	25.439
100	7,091.4	282.4	34.058

TABLE 4.—*Changes in relative humidity due to temperature.*

[Quantity of water assumed to be constant.]

Temperature of air.	Quantity of water per 100,000 cubic feet of air.	Relative humidity.	Temperature of air.	Quantity of water per 100,000 cubic feet of air.	Relative humidity.
° *F.*	*Gallons.*	*Per cent.*	° *F.*	*Gallons.*	*Per cent.*
0	1.329	Supersaturated.	60	1.329	14
10	1.329	100	70	1.329	10
20	1.329	63	80	1.329	8
30	1.329	40	90	1.329	7
40	1.329	27	100	1.329	6
50	1.329	19			

RÉSUMÉ OF MINE-HUMIDITY INVESTIGATIONS IN UNITED STATES.

CARL SCHOLZ.

Scholz, in October, 1908, drew attention to the fact that dust explosions in Oklahoma occurred only during the dry periods of the year.[a] He concluded from a series of observations that "the hygrometric condition of the atmosphere has the greatest effect upon the cause of explosions; and since this is a matter which can be controlled without much expense, it is believed that the application of a vaporizing system in mines which are dusty and generate fire damp will prove a remedy."

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[a] Scholz, Carl, Effect of humidity on mine explosions: Trans. Am. Inst. Min. Eng., vol. 39, 1908, pp. 328-336.

H. N. EAVENSON.

Eavenson, in November, 1908, prepared tables that show the average monthly weight of aqueous vapor and the average monthly temperatures in the outside atmosphere in the Appalachian coal fields.[a] He also presented the results of humidity observations in five mines in McDowell and Logan Counties, W. Va., where the quantity of air delivered by the fans ranged from 102,000 to 200,000 cubic feet per minute. He concluded from these figures that "during the winter months water is removed from the mine by the ventilating current at rates of 21 to 69 tons per 24 hours, and that in the summer months it is deposited in the same mines at rates of 12 to 23 tons in 24 hours. Can anyone doubt that the removal of such an amount of moisture from the mine air of mines apt to be dusty may convert a mine ordinarily safe into one liable to cause a terrific explosion, if anything should happen to raise the temperature to the explosion point of the dust?"

FRANK HAAS.

Haas[b] has given valuable information based on tests extending over a full year (June 1, 1908, to May 31, 1909) in certain mines near Fairmont, W. Va. The results of his investigation may be summarized as follows:

(*a*) Coal dust, in an atmosphere of 100 per cent relative humidity, absorbs a small amount of water, the exact amount depending on the nature of the dust and also on the length of time the dust is exposed.

(*b*) The suggestion is made that the evaporation of sufficient water will cool the temperature of an explosion below the ignition point of the coal dust.

(*c*) Temperature and humidity readings taken of the outside air and of the return air in the mines in question after being plotted on charts form the basis for the conclusion that an ordinary mine will show a large accumulated deficit of water per year. "Two months during the year the quantity of water taken into the mine by the air is in excess of that taken out. This period is approximately from the middle of June to the middle of August. Hence it appears that [in that district] artificial means of watering is necessary during the greater part of the year—from August 15 to June 15."

(*d*) Radiation from the side walls of an entry has considerable effect in raising the temperature of the air. In 4,000 feet of entry, where the average velocity was 453 feet per minute and the cross section 147 square feet, "over 30,000 British thermal units had been radiated per minute, which would be equivalent to heat derived from burning over a ton of coal per day." In this particular test, the temperature of the air at the intake was about 24° F. and at the return 51° F.

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[a] Eavenson, H. N., Effect of humidity on mine explosions: Trans. Am. Inst. Min. Eng., vol. 40, 1909, pp. 35-46.

[b] Haas, Frank, Exhaust steam as a preventive of dust explosions: Bull 20, Bureau of Mines, 1911, pp. 166-179.

(*e*) The quantity of water that can be carried mechanically in the form of a fog depends on the velocity of the air current. In order to prevent the fog line from advancing to the working face, three methods are suggested, as follows: First, reducing the volume and thereby the velocity of the air current; second, increasing the number or area of the airways; and, third, so changing the air courses that the distance to the working face is increased.

BUREAU OF MINES INVESTIGATORS.

The Bureau of Mines, at the Pittsburgh laboratories during September, 1909, carried on some interesting experiments to test the explosibility of coal dust under varying humidity conditions.[a] These tests were conducted in a 100-foot explosion gallery. All shots were made with 2½ pounds of black powder tamped with clay. The amount of dust was 120 pounds, of which 20 pounds were placed on a horse in front of the shot and 100 pounds distributed on side shelves running along the full length of the gallery, 1 pound per linear foot. A definite quantity of water was mechanically mixed with pure, high-volatile, 100-mesh coal dust, and the air current entering in the gallery was heated and humidified by compressed-air water sprays.

The results of these experiments indicated that slight dampness of the dust was not sufficient to prevent flame propagation by an explosion, but that there could be no propagation when the total moisture content of the dust approached 30 per cent. This amount of moisture did not make the 100-mesh dust exceedingly wet, as the dust could be formed into balls in the hand without any of the water being squeezed out. If more moisture is added, the critical point is reached, and with about 40 per cent the mixture becomes a coal mud.

The conclusion is drawn that "these experiments, though they must be accepted as preliminary, give a definite figure for the percentage of moisture that makes coal dust relatively safe under ordinary conditions. As a pure 100-mesh coal dust was used in the experiments, a larger amount of moisture probably was needed to render it inert than if it had contained larger particles, or had been less pure, or had been mixed with shale dust. The tests probably represented extreme conditions."

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[a] Rice, G. S., The explosibility of coal dust, with chapters by J. C. W. Fraser, Axel Larsen, Frank Haas, and Carl Scholz. Bull. 20, Bureau of Mines, 1911, 204 pp.

MINE-HUMIDITY INVESTIGATIONS IN ILLINOIS.

STATEMENT OF PROBLEMS.

A minimum quantity (usually 100 cubic feet) of air per man per minute is required by State law for the ventilation of coal mines. This air must be delivered to the places where the miners are working. In an average mine in Illinois it may be assumed that 100,000 cubic feet of air per minute is put into circulation, a quantity of air that will weigh 4 tons and may be represented by the volume of an entry 10 feet wide by 6 feet high by 1,667 feet long. As the ventilating machinery is kept in continuous operation, the volume of air passing through the mine in 24 hours will weigh 5,760 tons and may be represented by the volume of still air contained in an entry 10 feet by 6 feet by 453 miles long.

The maximum amount of aqueous vapor that can exist in any given space varies directly with the temperature. For example: In winter when the outside temperature is only 20° F. there is little moisture in the air. When this air enters the mine and is heated to a temperature of 65° F. it is able to carry much more moisture, and in order to obtain the moisture it evaporates water from the coal dust and the mine surfaces and thus renders the dust more inflammable.

The general method adopted to obtain information on the changes in temperature and relative humidity that air undergoes in passing through coal mines, and also to learn the effect of seasonal variations on the humidity of mine air, was to install hygrographs and thermographs at central points for continuous records of the outside atmosphere and to install hygrometers near the intake and return-air shafts of a number of representative mines to obtain records of the mine air.

RECORDS OF OUTSIDE ATMOSPHERE.

The United States Weather Bureau cooperated with the three cooperators doing the work herein reported and installed self-registering recorders of temperature (thermographs) and of humidity (hygrographs) at La Salle and Springfield, Ill., and at St. Louis, Mo.

Similar instruments were already in operation at the University station at Urbana.

At Carbondale, Ill., a thermograph and a hygrograph were installed by the investigators in connection with the equipment being used at the local weather bureau.

The charts received from these stations give a continuous record of the temperature and relative humidity of the outside air for the period February, 1912, to April, 1913.

The situation of these stations is indicated in the tabulation following, and in figure 1.

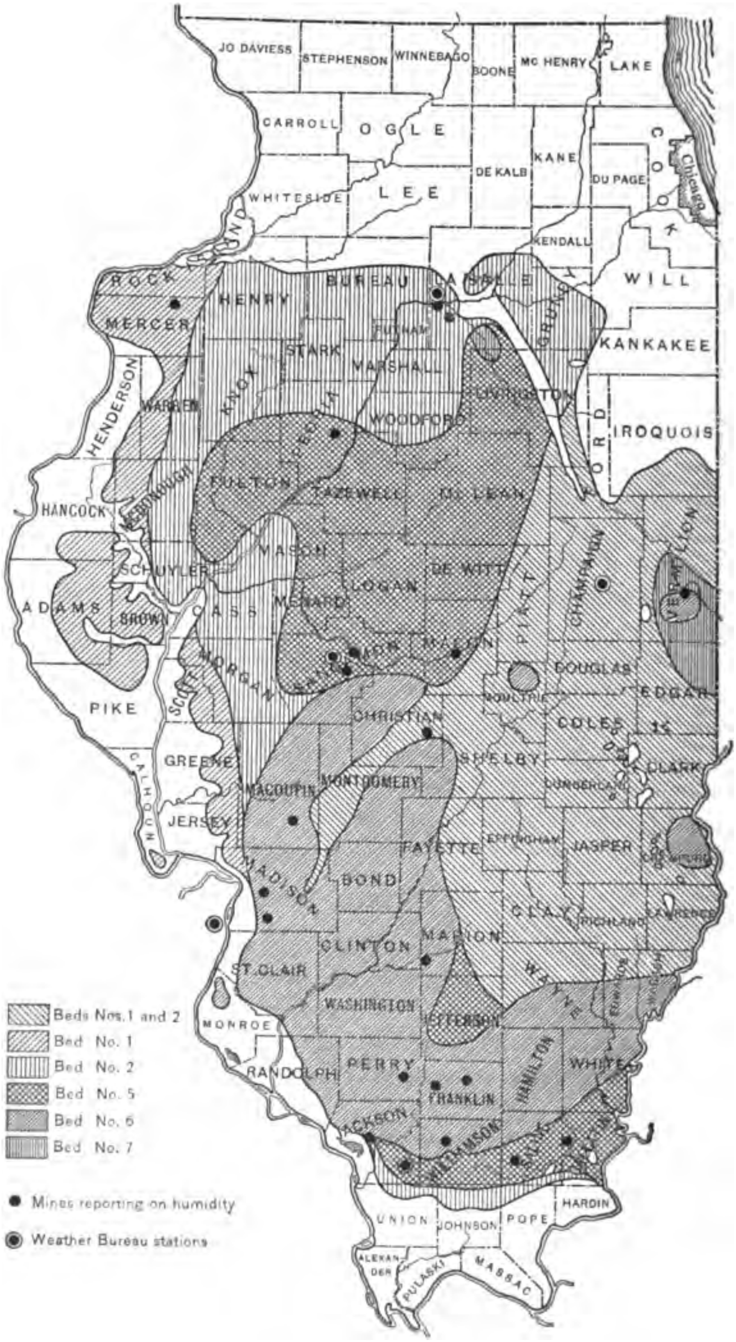

FIGURE 1.—Approximate coal areas (Bement) showing mines and United States Weather Bureau hygrograph stations that recorded humidity.

Stations reading weather conditions.

Name of station.	Town.	County.	Part of State.
U. S. Weather Bureau	La Salle	La Salle	Northern.
Do	Springfield	Sangamon (near)	Central.
Do	St. Louis, Mo.	St. Clair (near)	Southwestern.
University of Illinois	Urbana	Champaign	Eastern.
Mining investigation and local observer	Carbondale	Jackson	Southern.

RECORDS OF MINE AIR.

To learn the moisture content of the mine air, 20 mines were selected to represent the average conditions in the State. In each of these mines two hygrometers were installed—one in the intake air course and one in the return air course. In order that the proper rate of evaporation from the surface of the wet bulb might be obtained, the hygrometers were placed at points where the velocity of the air was at least 600 feet per minute.

The intake hygrometer, which in each mine was installed within 300 feet of the downcast shaft, gave records of the average daily amount of moisture carried by the ventilating current as it entered the underground workings. These readings therefore supplemented the Weather Bureau observations by showing the changes in the temperature and relative humidity of the air due to variations in the shaft conditions, such as depth, amount of falling water, and presence of steam and water pipes. The return hygrometer was placed at a point within 500 feet of the upcast shaft, and showed the moisture content of the air leaving the mine. The readings of both hygrometers were made by local officials of each mine three to six times for each 24 hours, and the results were forwarded to Urbana for tabulation.

The names of the mines that furnished hygrometric data are given in Table 5, following:

TABLE 5.—*Mines furnishing hygrometric data, hours of reading, and names of readers.*

Name of mine.	Company.	Town.	County.	Hours of reading.	Name of reader.
Oglesby	Oglesby Coal Co.	Oglesby	La Salle	7 a. m., 12 m., 4 p. m.	D. W. Kelly.
La Salle shaft	La Salle County Carbon Coal Co.	La Salle	do	7 a. m., 12 m., 3.30 p. m., 9.30 p. m.	J. G. Millhouse.
No. 9	Big Muddy Coal & Iron Co.	Murphysboro.	Jackson	do	J. M. Mitchell.
No. 2	Coal Valley Mining Co.	Sherrard	Mercer	7 a. m., 12 m., 4 p. m.	R. E. Lee.
No. 1	Assumption Coal Mining Co.	Assumption.	Christian	8 a. m., 12 m., 4 p. m.	T. J. Connolly.
Empire No. 2	Clark Coal & Coke Co.	Peoria	Peoria	8 a. m., 11 a. m., 2 p. m.	R. U. Tyson.
Peerless No. 1	Jones & Adams Coal Co.	Springfield	Sangamon	8 a. m., 12 m, 4 p. m.	P.W.MacMurdo.
Woodside	Woodside Coal Co.	do	do	do	Robert Wilson.
Longwall M. & C.	Manufacturers & Consumers Coal Co.	Decatur	Macon	7.30 a. m., 11.30 a. m., 3.30 p. m.	Louis White.
No. 3	Saline County Coal Co.	Harrisburg	Saline	7 a. m., 12 m., 4 p. m.	Walter Scott.

TABLE 5.—*Mines furnishing hygrometric data, hours of reading, and names of readers*—Continued.

Name of mine.	Company.	Town.	County.	Hours of reading.	Name of reader.
Equality No. 1..	Gallatin Coal & Coke Co.	Equality....	Gallatin.....	7 a. m., 12 m., 4 p. m.	J. B. Smith.
Hart-Williams...	Hart-Williams Coal Co.	Benton.....	Franklin....	8 a. m., 12.30 p. m., 4.30 p. m., 9 p. m., 12 a. m., 4 a. m.	C. O. Wellington, Robt. Waugh.
Paradise.........	Paradise Coal & Coke Co.	Duquoin....	Perry.......	7 a. m., 12 m., 4 p. m.	J. H. Lester, Hugh McGonaghie.
East Mine.......	United Coal Mining Co.	Christopher.	Franklin....	.....do............	Walter Stephens.
Big Muddy......	Chicago & Big Muddy Coal Co.	Marion......	Williamson.	8 a. m., 12 m., 3 p. m.	A. B. McClaren; J. L. Moses.
No. 3............	Superior Coal Co..	Gillespie....	Macoupin...	7 a. m., 12 m., 4 p. m.	Philip White, D. E. Reese.
Madison No. 2...	Madison Coal Corporation.	Glen Carbon	Madison.....	.....do............	William Turton.
Lumaghi No. 2..	Lumaghi Coal Co..	Collinsville..	.....do......	8 a. m., 12 m., 4 p. m., 6 p. m.	W. T. Scully.
No. 5............	Centralia Coal Co..	Centralia....	Marion......	8 a. m., 1 p. m., 3 p. m.	J. H. Hoye.
Little Vermilion.	Bunsen Coal Co...	Westville...	Vermilion...	7 a. m., 12 m., 4 p. m.	W. B. Wiley.

CALCULATION AND TABULATION OF RESULTS.

The monthly reports from the Weather Bureau stations contained for each two-hour interval the average temperature and relative humidity of the outside atmosphere. The daily-report cards from the 20 mines contained records of the quantity of air in circulation underground and at least three readings of the wet-bulb and of the dry-bulb thermometers of the intake and return hygrometers. From these records the average daily readings were obtained and the results recorded in the weekly humidity reports. By further combination the average weekly readings were computed and calculations were made of the quantity of water per 100,000 cubic feet of air and also of the quantity of water deposited or extracted per 24 hours. A specimen daily-report card and a specimen weekly humidity report are shown below.

Specimen daily report card of mine-humidity reading.

DAILY REPORT. DATE: 4-1-1912.
Mine, Paradise. Address, Duquoin, Ill.
Total air in mine, 66,500 cu. ft. per min.

Return air:	Time.	Dry bulb.	Wet bulb.	Intake air:	Time.	Dry bulb.	Wet bulb.
Velocity	7 a. m.	60½	59½	Velocity	7 a. m.	31	30
520				1,478			
Area	12 m.	60½	59½	Area	12 m.	34	30
96				45			
	4 p. m.	60½	60		4 p. m.	36	33

(Signed) JAS. H. LISTER,
Mine manager.

Specimen weekly humidity report.

WEEKLY HUMIDITY REPORT.

No. 17.
Mine name, Sherrard No. 2.
Operator, Coal Valley Mining Co. Address, Sherrard.
Total air current, 52,000 cu. ft. per minute.

Week ending May 12, 1912.

Average readings.	Monday.	Tuesday.	Wednesday.	Thursday.	Friday.	Saturday.	Sunday.
OUTSIDE AIR.							
Dry bulb..................(t)..	68¼	67½	62	61	66⅓	58	46¼
Wet bulb..................(t′)..	....	....	....	....	....	....	....
Depression..............(t−t′)..	....	....	....	....	....	....	....
Rel. humidity............(%)..	64	66	55	56	59	89	83
INTAKE AIR.							
Dry bulb..................(t)..	66⅛	63¼	58⅔	58½	59⅔	63¼	63¼
Wet bulb..................(t′)..	64½	61 7/12	57 1/12	57	57¾	61⅙	61⅙
Depression..............(t−t′)..	1⅝	1⅔	1 7/12	1½	1 11/12	1 1/12	1 1/12
Rel. humidity............(%)..	90	90	91	91	89	92	92
RETURN AIR.							
Dry bulb..................(t)..	67¾	68	67⅝	67½	67¾	67¾	67½
Wet bulb..................(t′)..	67⅙	66¾	66⅜	66½	66¾	66½	66½
Depression..............(t−t′)..	7/12	1¼	1¼	1	1	1¼	1
Rel. humidity............(%)..	97	94	94	95	95	94	95

Computations from average readings.	Dry bulb (t) ° F.	Depression of wet bulb (t−t′).	Relative humidity %.	Gals. water per 100,000 cu. ft. air.	Gals. water per total air 24 hours.	Gals. artificially introduced per 24 hrs.	Gals. of water, extracted, (−), deposited (+), per 24 hrs.
Outside air.........................	61⅓	....	67	6.85	5,147	....	....
Intake air shaft....................	61¾	1½	91	9.65	7,414	....	....
Return air shaft...................	67⅝	1	95	12.08	9,032	....	....

After the investigation had continued for one year, tabulations (Table 6), from the weekly averages were made showing the temperature, relative humidity, and actual moisture content (gallons of water per 100,000 cubic feet of air) of the outside atmosphere and of the intake and the return air in each of the 20 mines. Typical results of these tabulations are plotted in figures 2 and 3.

METHOD OF USING PSYCHROMETRIC TABLES.[a]

In order to show the method of using psychrometric tables, a determination is given in full below:

1. Information given and to be obtained:

Reading.	Mine intake.	Mine return.
Area....................................	......	10 by 16 feet.
Velocity..................................	......	625 feet per minute.
Barometer..............................	[b] 30	[b] 30
Dry bulb (*t*)............................	10°	60°
Wet bulb (*t′*)...........................	8°	59½°

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[a] Marvin, C. F., Psychrometric tables for obtaining the vapor pressure, relative humidity, temperature of the dew point, and grains of water per cubic foot of air: U. S. Weather Bureau, Bull. 235, 1900.

[b] The normal barometer reading for any mine in Illinois may be assumed at 30 inches. Any error resulting from this assumption is exceedingly small.

2. **Use of the tables:**

(*a*) **Quantity of air passing through mine—**

Area×velocity=160×625=100,000 cubic feet per minute.

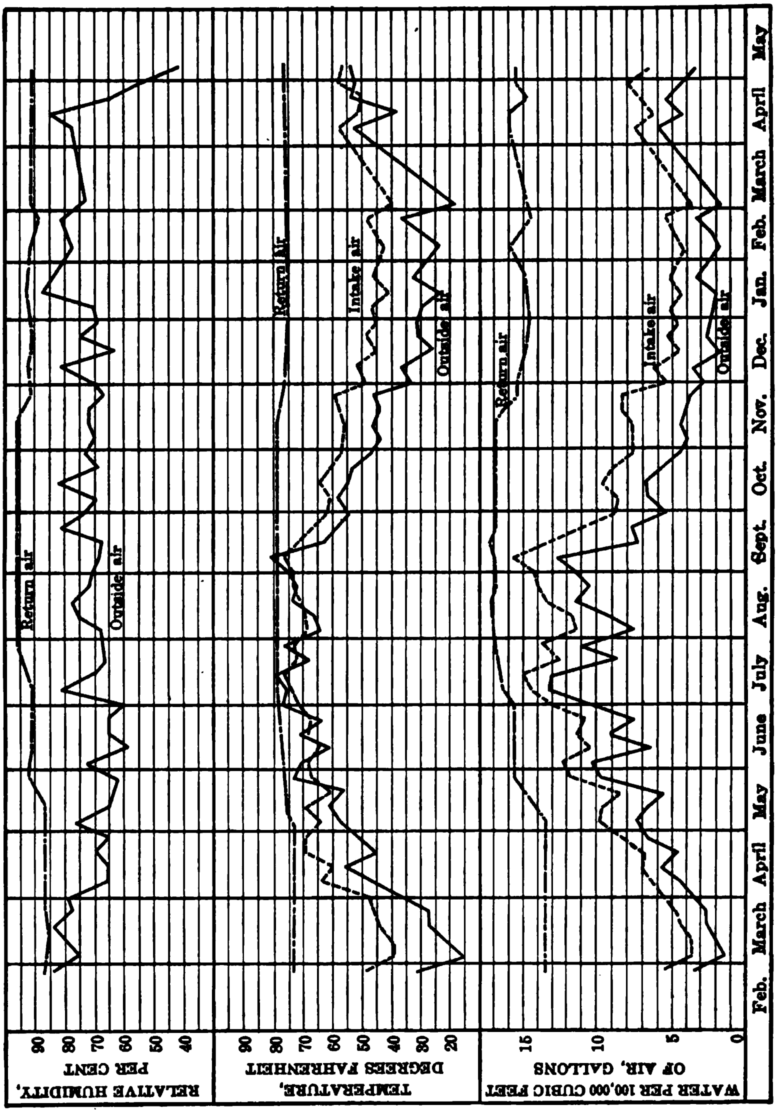

FIGURE 2.—Curves plotted from hygrometric records of a typical long-wall mine in La Salle County, Ill.

(*b*) Depression of wet-bulb thermometer—

Intake air: $(t-t')=(10-8\ )=2°$.

Return air: $(t-t')=(60-59\frac{1}{2})=\frac{1}{2}°$.

(*c*) Relative humidity (U. S. Weather Bureau Bull. 235, Table 7)—

Relative humidity of intake air, with air temperature =10° and depression of wet bulb=2°=56 per cent.

Relative humidity of return air, with air temperature=60° and depression of wet bulb=½°=97 per cent.

(*d*) Quantity of water per cubic foot of air (Bull. 235, Table XII):

Quantity of water in intake air, with air temperature=10° and with the relative humidity=56 per cent=0.435 grains per cubic foot of air.

Quantity of water in return air, with air temperature=60° and with the relative humidity=97 per cent=5.573 grains per cubic foot of air.

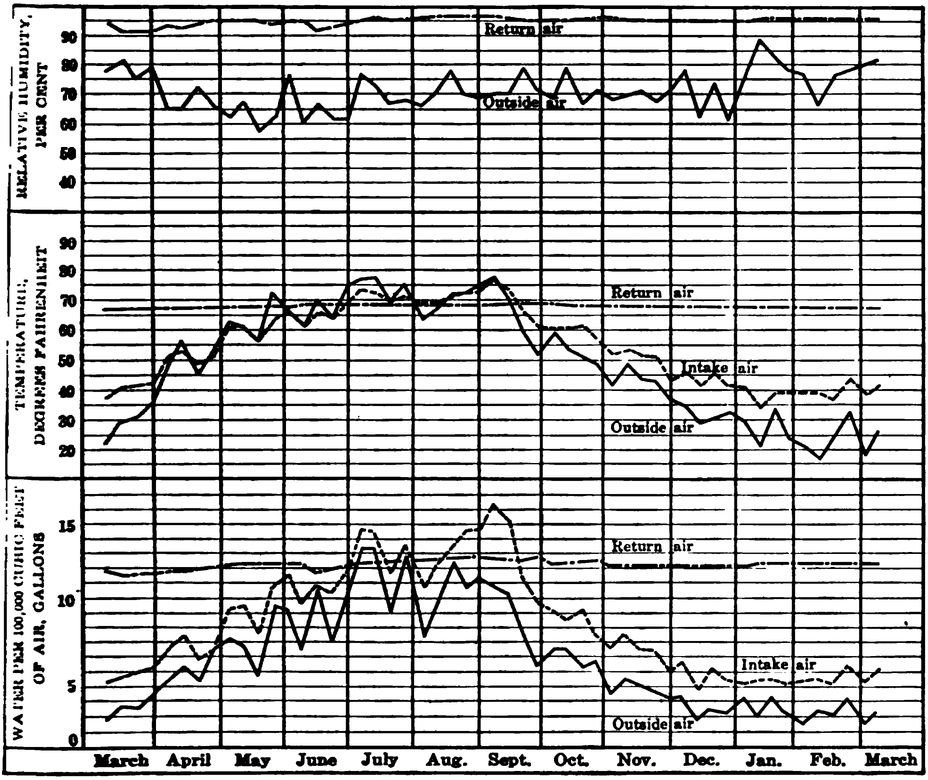

FIGURE 3.—Curves plotted from hygrometric records of a typical room-and-pillar mine in Mercer County, Ill.

(*e*) Gallons of water per 100,000 cubic feet of air—

If 1 pound avoirdupois=7,000 grains and 1 gallon of water at ordinary temperatures weighs 8.35 pounds, then:

Gallons per 100,000 cubic feet of intake air—

$$\frac{0.435\times100{,}000}{7{,}000\times8.35}=0.744 \text{ gallon.}$$

Gallons per 100,000 cubic feet of return air—

$$\frac{5.573\times100{,}000}{7{,}000\times8.35}=9.535 \text{ gallons.}$$

(*f*) Extraction or deposition of moisture—

The amount of air passing through the mine is 100,000 cubic feet per minute; therefore every minute 9.535−0.744=8.791 gallons of water are extracted from the mine. Reducing this amount to 24 hours, we find that under the assumed conditions an enormous quantity is taken daily from the mine, namely, 8.777 gallons×60 minutes×24 hours=12,659 gallons=52.85 tons of water a day.

HYGROMETRIC RECORDS.

Table 6, giving the weekly averages of the hygrometric records made, is presented below:

TABLE 6.—*Weekly averages of hygrometric records made in Illinois cooperative investigations.*

OGLESBY MINE, OGLESBY COAL CO., LA SALLE COUNTY.

Week ending—	Temperature.			Relative humidity.[a]			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1912.	°*F.*	°*F.*	°*F.*	*Per cent.*	*Per cent.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gallons.*
Mar. 3	15.00	28.44	71.17	74.00	97.00	89.67	1.25	2.34	12.74
10	21.00	29.92	71.17	83.00	95.50	95.93	1.75	3.15	13.55
17	27.50	34.56	71.00	83.00	83.67	97.00	2.46	3.33	13.71
24	28.67	36.77	71.83	77.00	89.00	96.50	2.41	3.85	14.01
31	36.33	39.00	71.60	77.00	89.60	94.00	3.28	4.21	13.54
June 30	76.00	70.33	75.00	61.00	85.00	91.00	10.11	11.76	14.61
July 7	75.92	73.67	75.73	79.00	90.00	96.00	13.06	13.95	15.77
14	77.42	73.82	76.13	75.00	86.00	99.67	12.20	13.30	16.58
21	69.42	68.13	76.67	64.00	82.50	100.00	8.60	10.63	16.92
28	76.60	73.60	76.60	69.00	86.60	100.00	11.65	13.30	16.89
Aug. 4	65.58	66.67	76.50	67.00	80.00	100.00	7.94	9.83	16.83
11	67.00	67.50	76.33	80.00	84.17	100.00	9.93	10.62	16.74
18	70.67	70.13	76.22	79.00	90.33	100.00	11.05	12.42	16.69
25	72.25	71.33	76.83	72.00	89.00	99.83	10.59	12.71	16.98
Sept. 1	74.00	72.67	77.00	72.00	86.00	100.00	11.20	12.82	17.10
8	80.15	76.47	77.00	70.00	85.00	100.00	13.20	14.30	17.10
15	62.83	71.33	77.00	67.50	82.00	100.00	7.19	11.71	17.10
22	59.33	63.56	76.50	82.00	84.00	100.00	7.89	9.31	16.83
29	54.13	58.33	76.00	73.00	79.00	100.00	5.88	7.35	16.57
Oct. 6	58.33	58.23	75.00	69.00	80.00	100.00	6.42	7.42	16.05
13	55.00	60.67	75.00	82.00	85.00	95.00	6.81	8.56	15.25
20	53.33	58.08	75.00	68.00	80.50	95.00	5.33	7.43	15.25
27	47.00	51.67	75.00	73.00	80.00	86.00	4.58	5.92	13.81
Nov. 3	44.50	50.83	74.83	70.00	79.00	88.00	4.02	5.68	14.05
10	46.33	52.50	73.83	72.00	80.00	90.67	4.42	6.09	14.48
17	43.33	50.39	74.33	72.50	81.67	84.00	3.99	5.78	13.21
24	45.33	48.94	73.67	66.00	80.00	96.67	3.90	5.38	14.91
Dec. 1	34.00	40.67	73.07	70.00	80.80	99.80	2.73	4.04	15.08
8	35.30	43.13	73.00	80.00	85.00	99.80	3.28	4.65	15.04
15	26.00	35.22	72.67	62.00	80.00	99.00	1.72	3.27	14.76
22	31.67	39.44	72.44	75.00	85.00	99.83	2.67	4.07	14.78
29	31.60	37.20	71.80	69.00	86.00	100.00	2.45	3.78	14.50

LA SALLE MINE, LA SALLE COUNTY CARBON COAL CO., LA SALLE COUNTY.

1912.									
Feb. 25	32.83	49.33	74.00	83.00	86.33	86.00	3.10	5.89	13.38
Mar. 3	15.00	39.67	74.13	74.00	82.33	85.50	1.25	3.95	13.35
10	21.00	39.25	74.17	79.00	82.00	85.00	1.75	3.89	13.29
17	27.50	43.00	74.00	83.00	79.00	85.33	2.47	4.30	13.33
24	28.67	45.50	74.00	77.00	82.00	86.00	2.41	4.86	13.38
May 5	59.67	47.50	74.00	77.00	86.00	86.00	7.36	5.41	13.38
12	61.75	64.25	76.00	65.00	84.25	87.00	6.80	9.56	14.38
19	56.00	58.90	76.00	64.00	86.00	90.00	5.50	8.15	14.91
26	74.50	68.67	76.25	61.00	89.00	93.00	9.64	11.66	15.53
June 2	70.25	69.88	76.50	77.00	90.00	92.75	10.55	12.27	15.61
9	61.66	64.33	77.00	59.00	90.00	91.00	6.14	10.24	15.56
16	71.00	68.00	77.00	67.00	90.00	91.00	9.47	11.54	15.56
23	64.00	66.20	77.00	67.00	89.00	91.00	7.54	10.76	15.56
30	77.50	72.50	77.00	59.50	88.00	91.00	10.34	13.05	15.56
July 7	76.33	74.67	78.00	80.00	92.00	91.33	13.40	14.61	16.11
14	78.50	76.33	78.00	72.00	90.50	92.50	12.91	15.15	16.32
21	68.00	71.75	78.00	66.50	85.00	94.50	8.53	12.26	16.67
28	76.00	73.25	78.00	67.00	91.33	96.00	11.11	13.87	16.94
Aug. 4	64.50	68.50	78.00	68.00	85.00	96.00	7.78	11.08	16.94
11	66.50	68.87	78.25	75.00	88.00	96.00	9.16	11.61	17.01
18	73.20	71.75	78.40	77.00	93.00	96.00	11.68	13.47	17.15
25	72.25	73.00	78.00	72.00	93.00	96.00	10.60	14.01	16.94

[a] The 100 per cent relative humidity of the return air, as reported from July 14 to Oct. 6, is too high, the error being due to the failure of the wicking to conduct water from the cup to the wet-bulb thermometer. This failure was caused by the clogging of the wick by impurities in the water and by dust settling from the air currents.

TABLE 6.—*Weekly averages of hygrometric records made in Illinois cooperati investigations*—Continued.

LA SALLE MINE, LA SALLE COUNTY CARBON COAL CO., LA SALLE COUNTY—Co

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic f		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Re a
	°F.	°F.	°F.	*Per cent.*	*Per cent.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gal*
1912.									
Sept. 1	74.33	73.67	78.00	71.00	93.00	96.00	11.16	14.31	
8	80.00	77.00	78.25	68.00	93.00	96.00	12.77	15.90	
15	62.33	72.75	78.50	67.50	87.33	96.00	7.22	13.08	
22	59.33	67.00	78.00	82.00	87.00	96.00	7.89	10.80	
29	54.13	62.50	78.00	73.00	84.00	96.00	5.88	8.99	
Oct. 6	58.33	61.00	78.00	69.00	85.50	96.00	6.42	8.70	
13	55.00	64.67	78.00	82.00	84.50	96.00	6.81	9.72	
20	53.67	61.00	78.00	68.00	88.00	96.00	5.40	8.96	
27	47.00	57.25	78.00	73.00	86.67	96.00	4.58	7.77	
Nov. 3	44.50	56.87	78.00	70.00	86.00	96.00	4.02	7.63	
10	46.33	56.00	78.00	72.00	89.00	96.00	4.42	7.65	
17	43.33	57.67	77.67	72.50	90.87	94.00	3.99	8.25	
24	45.33	58.13	77.00	66.00	88.00	91.00	3.90	8.13	
Dec. 1	34.00	49.40	76.80	70.00	81.60	91.00	2.73	5.58	
8	36.00	52.50	76.17	81.50	83.83	91.83	3.42	6.39	
15	26.00	45.00	75.83	62.00	79.33	90.33	1.72	4.64	
22	31.67	49.00	75.67	75.00	83.00	90.33	2.67	5.59	
29	31.60	45.25	75.00	69.00	78.00	91.00	2.45	4.77	
1912									
5	29.90	47.00	75.00	70.40	82.00	91.00	2.40	5.15	
12	22.17	42.00	75.00	87.50	81.00	92.00	2.03	4.25	
19	33.65	46.67	75.17	84.17	82.00	91.00	3.22	5.09	
26	28.47	45.33	75.50	80.00	80.00	91.00	2.48	4.76	
Feb. 2	23.38	43.50	75.75	77.00	79.75	91.00	1.90	4.42	
23	37.66	49.00	75.33	80.17	83.67	88.00	3.59	5.64	
Mar. 2	18.47	40.00	75.00	74.00	76.00	91.00	1.46	3.71	
Apr. 6	53.83	58.00	76.00	78.67	84.67	91.00	6.27	7.79	
13	39.25	52.64	76.00	85.75	79.50	91.00	4.06	6.09	
20	54.47	50.50	76.75	65.50	83.00	91.75	5.34	5.90	
27	53.33	58.75	77.00	57.00	84.75	91.00	4.51	8.00	
May 4	54.25	57.67	77.00	40.00	82.00	91.00	3.24	7.46	

MINE NO. 9, BIG MUDDY COAL & IRON CO., MURPHYSBORO, JACKSON COUN

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic f		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Re a
1912.									
Feb. 11	....	42.88	48.90	....	70.00	100.00	....	3.79	
18	....	45.50	52.50	....	74.50	100.00	....	4.43	
25	....	46.25	56.00	....	73.00	100.00	....	4.46	
Mar. 3	....	44.00	55.90	....	74.00	100.00	....	4.17	
10	....	43.50	56.67	....	77.50	100.00	....	4.29	
17	....	45.50	57.00	....	82.00	100.00	....	4.88	
24	....	51.00	57.50	....	85.00	100.00	....	6.14	
31	....	51.50	57.67	....	87.00	100.00	....	6.40	
Apr. 7	63.00	55.50	58.00	67.00	85.00	100.00	7.29	7.18	
14	62.00	56.50	58.00	68.00	86.00	100.00	7.16	7.52	
21	....	56.25	28.25	....	82.00	99.75	....	7.13	
May 28	60.50	56.20	58.25	61.00	84.00	99.85	6.11	7.27	
5	69.00	60.13	59.00	72.00	93.00	100.00	9.54	9.20	
12	70.20	62.80	59.00	70.00	93.50	100.00	9.64	10.11	
19	61.75	57.88	58.50	69.00	82.25	100.00	7.26	8.65	
26	76.80	67.00	59.50	65.00	93.00	100.00	11.04	11.21	
2	71.00	65.67	60.00	72.00	94.00	100.00	10.18	11.17	
9	71.00	65.00	60.00	63.00	87.60	100.00	8.90	10.18	
16	71.50	66.50	60.00	74.00	90.00	100.00	10.63	10.99	
23	62.00	64.50	60.00	85.00	95.67	100.00	8.95	10.94	
30	68.00	66.90	60.00	85.00	96.33	100.00	10.91	11.92	
7	73.60	71.80	60.33	89.00	99.80	100.00	13.66	14.47	
14	77.00	73.50	61.25	83.00	98.67	100.00	14.19	15.10	
21	75.00	72.75	61.50	83.00	98.40	100.00	13.32	14.71	
28	78.00	73.25	61.50	77.00	99.00	100.00	13.15	15.04	
4	71.00	69.33	61.08	75.00	94.00	100.00	9.59	12.59	
11	67.60	67.90	61.00	76.00	90.80	100.00	9.62	11.61	
18	75.20	72.83	61.00	85.00	99.00	100.00	13.73	14.83	
25	71.17	73.00	61.67	84.00	95.00	100.00	13.13	14.31	
1	76.33	73.50	62.00	79.00	96.50	100.00	13.22	14.77	
8	78.75	75.00	62.00	75.00	91.00	100.00	13.54	14.61	
15	70.00	71.50	62.00	71.00	90.00	100.00	9.72	12.93	
22	64.20	67.67	61.50	86.00	93.00	99.00	6.74	11.80	
29	54.00	62.50	61.00	81.00	80.67	100.00	6.50	8.63	

TABLE 6.—*Weekly averages of hygrometric records made in Illinois cooperative investigations*—Continued.

MINE NO. 9, BIG MUDDY COAL & IRON CO., MURPHYSBORO, JACKSON COUNTY—Con.

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1912.	*°F.*	*°F.*	*°F.*	*Per cent.*	*Per cent.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gallons.*
Oct. 6	55.00	61.00	60.00	81.00	82.50	100.00	6.73	8.40	9.84
13	65.00	65.50	60.00	72.00	90.00	100.00	8.37	10.64	9.84
20	52.80	60.00	60.00	85.00	87.00	100.00	6.55	8.56	9.84
27	49.40	56.60	59.60	76.50	84.00	100.00	5.23	7.37	9.71
Nov. 3	50.67	54.00	59.10	76.50	81.00	100.00	5.60	6.50	9.55
10	48.67	55.00	57.00	65.00	77.00	100.00	4.33	6.40	8.89
17	39.33	54.33	59.00	80.33	73.00	100.00	3.83	5.93	9.52
24	45.75	53.60	58.33	61.00	71.50	100.00	3.66	5.66	9.31
Dec. 1	34.50	48.00	58.00	64.50	74.00	100.00	2.67	4.82	9.20
8	41.00	50.50	58.00	77.60	77.50	100.00	3.93	5.51	9.20
15	25.50	43.30	57.75	65.50	66.00	99.50	1.77	3.63	9.07
22	31.00	46.33	57.67	78.00	72.00	100.00	2.69	4.41	9.09
29	30.00	45.17	57.00	75.00	76.67	100.00	2.48	4.51	8.89
1913.									
Jan. 5	33.50	46.67	57.17	71.00	75.00	100.00	2.72	4.65	8.94
12	29.20	45.33	57.17	92.00	81.00	100.00	2.94	4.79	8.94
19	41.00	48.60	56.00	86.67	86.00	100.00	4.38	5.72	8.59
26	35.25	48.25	57.67	88.00	85.00	100.00	3.60	5.58	9.09
Feb. 2	30.25	44.25	57.25	70.00	71.00	100.00	2.35	4.04	8.97
9	16.50	42.83	51.50	80.00	81.00	100.00	1.45	4.38	7.36
16	27.00	43.25	55.00	64.00	78.00	100.00	1.86	4.59	8.31
23	52.00	51.67	57.00	78.00	84.33	100.00	5.84	6.98	8.89
SHERRARD NO. 2 MINE, COAL VALLEY MINING CO., MERCER COUNTY.									
1912.									
Mar. 10	21.92	36.50	67.17	77.33	99.00	94.00	1.78	4.24	11.74
17	29.42	40.33	67.42	81.00	97.00	91.00	2.63	4.79	11.46
24	30.14	41.33	67.50	75.00	96.00	91.00	2.48	4.92	11.51
31	37.58	43.86	67.50	79.00	92.00	91.00	3.51	5.16	11.51
Apr. 7	48.50	51.08	67.42	65.00	96.00	94.00	4.30	6.96	11.83
14	57.50	53.25	67.67	65.00	93.00	93.00	5.97	7.46	11.80
21	45.83	48.00	67.75	72.00	91.00	94.00	4.36	5.92	11.96
28	52.58	51.57	67.58	66.00	90.00	95.00	6.60	6.64	12.02
May 5	63.75	60.25	67.63	63.00	93.00	95.00	7.02	9.22	12.04
12	61.33	61.75	67.89	67.00	91.00	95.00	6.85	9.50	12.14
19	55.88	55.67	67.50	57.00	89.00	95.33	4.89	7.56	12.13
26	73.71	64.83	68.00	63.00	92.00	94.00	9.71	10.63	12.05
June 2	66.17	66.58	67.92	76.00	93.71	95.00	9.13	11.42	12.15
9	62.70	61.75	68.00	60.00	92.00	95.00	6.42	9.60	12.18
16	70.00	67.25	68.00	67.00	86.00	92.00	10.54	10.77	11.80
23	64.00	64.75	68.00	63.00	87.57	93.00	7.08	10.10	11.93
30	75.83	69.33	68.00	63.00	85.00	94.00	10.43	11.76	12.05
July 7	77.17	74.50	68.25	77.00	92.00	95.71	13.16	14.54	12.37
14	78.33	73.92	68.25	74.00	93.00	96.00	13.19	14.28	12.41
21	69.67	69.92	68.50	67.00	86.43	95.71	9.17	11.80	12.48
28	75.58	72.50	68.50	68.00	90.14	96.00	12.76	13.37	12.51
Aug. 4	64.10	68.42	68.50	66.00	83.57	96.29	7.42	10.86	12.55
11	67.00	69.83	68.50	80.00	90.00	97.00	9.93	12.32	12.64
18	73.83	72.00	68.50	78.00	93.29	97.00	12.13	13.62	12.64
25	73.58	73.50	68.50	70.00	95.57	97.00	10.71	14.63	12.64
Sept. 1	75.83	73.83	68.50	69.00	95.29	97.00	11.13	14.71	12.64
8	77.92	76.50	68.50	70.00	96.71	97.00	12.30	16.28	12.64
15	71.67	74.92	68.83	70.00	95.00	96.33	10.17	15.21	12.63
22	59.92	66.58	69.00	79.00	92.50	95.00	7.78	11.33	12.59
29	53.17	62.17	69.00	72.00	91.00	95.00	5.58	9.64	12.59
Oct. 6	60.20	60.75	69.00	68.00	91.00	95.00	6.69	9.19	12.59
13	54.33	60.58	68.25	79.00	87.00	95.00	6.41	8.73	12.28
20	52.83	61.92	68.25	67.00	88.00	96.00	5.19	9.23	12.41
27	48.58	55.17	68.00	72.00	89.00	97.00	5.64	7.43	12.44
Nov. 3	43.83	53.00	68.00	69.00	89.00	97.00	3.75	6.90	12.44
10	48.00	54.00	68.00	70.00	93.43	95.00	4.55	7.50	12.18
17	44.25	52.14	68.00	72.00	88.43	95.00	4.10	6.65	12.18
24	43.60	51.71	68.00	67.00	88.71	95.00	3.87	6.57	12.18
Dec. 1	36.50	44.50	68.00	71.00	87.00	95.00	3.04	5.00	12.18
8	34.33	45.71	68.00	79.00	97.00	95.00	3.12	5.82	12.18
15	28.00	41.67	68.00	63.57	77.00	95.00	1.94	3.99	12.18
22	30.50	45.14	68.00	74.00	86.00	95.00	2.51	5.29	12.18
29	33.33	42.17	68.00	62.00	85.00	95.00	2.39	4.49	12.18

TABLE 6.—*Weekly averages of hygrometric records made in Illinois cooperative investigations*—Continued.

SHERRARD NO. 2 MINE, COAL VALLEY MINING CO., MERCER COUNTY—Continued

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Retur air.
	°F.	*°F.*	*°F.*	*Per cent.*	*Per cent.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gallon*
1913. Jan. 5	29.92	41.83	68.00	77.00	82.00	95.00	3.30	4.27	12.
12	21.60	35.58	68.00	88.00	90.60	96.00	2.00	4.48	12.
19	34.29	39.86	68.00	83.86	95.00	96.00	3.31	4.61	12.
26	24.67	39.57	68.00	79.43	91.00	95.00	2.08	4.37	12.
Feb. 2	21.88	39.33	68.00	76.14	95.00	96.00	1.69	4.52	12.
9	16.58	39.50	68.00	67.29	98.00	96.00	2.22	4.69	12.
16	25.58	37.00	68.00	76.00	98.00	96.00	2.06	4.28	12.
23	33.29	44.50	68.00	79.50	94.00	96.00	3.02	5.36	12.
Mar. 2	18.14	38.33	68.00	80.43	94.00	96.00	1.56	4.31	12.
9	26.67	42.00	68.00	82.33	96.00	96.00	2.35	5.04	12.
ASSUMPTION NO. 1 MINE, ASSUMPTION COAL & MINING CO., CHRISTIAN COUNT									
1912. Feb. 18	36.25	41.50	62.50	81.00	86.00	84.00	3.44	4.43	8.
25	31.25	41.67	63.25	76.00	88.00	88.00	2.67	4.56	9.
Mar. 3	22.00	40.83	61.33	72.50	88.00	89.00	1.68	4.42	9.
10	24.33	43.50	59.50	77.00	88.50	92.00	1.98	4.90	8.
17	31.50	44.67	66.67	83.00	91.00	95.00	2.94	5.25	11.
24	34.00	47.75	68.00	80.00	93.00	95.00	3.14	6.30	12.
31	39.60	48.00	68.67	80.00	93.50	95.00	3.85	6.09	12.
Apr. 21	48.00	55.00	69.83	70.00	93.00	93.67	4.56	7.72	12.
28	57.50	56.50	70.00	60.00	94.00	95.00	5.43	8.23	13.
May 5	63.00	59.75	70.00	75.00	95.00	95.00	8.16	9.27	13.
12	67.50	64.67	69.00	65.00	97.00	97.67	8.20	11.16	12.
19	54.50	58.50	68.67	67.00	94.00	94.00	5.47	8.79	12.
26	74.67	66.83	70.17	59.00	95.67	95.00	9.37	11.81	13.
June 2	68.00	67.83	71.17	75.00	96.40	95.00	9.62	12.03	13.
9	65.50	64.00	71.83	55.50	95.67	97.00	6.56	10.76	14.
16	72.75	66.00	72.50	62.50	97.67	96.00	9.35	11.61	14.
23	64.00	65.50	72.67	70.00	97.00	97.00	7.87	11.47	14.
30	75.33	68.67	73.00	66.00	96.33	97.70	10.71	12.63	14.
July 7	77.00	73.75	74.00	78.00	97.00	96.50	13.34	14.97	15.
14	78.67	74.50	73.67	76.00	98.00	98.00	13.69	15.50	15.
21	74.50	72.25	74.17	65.50	97.00	98.00	10.36	14.27	15.
28	78.50	72.50	74.50	61.50	98.00	98.00	11.02	14.53	15.
Aug. 4	70.33	69.00	74.50	62.00	97.50	98.00	8.57	12.92	15.
11	69.00	67.75	74.50	74.00	97.67	98.00	9.80	12.42	15.
18	73.83	72.17	74.50	80.00	95.00	87.00	12.38	13.94	13.
25	76.83	72.33	75.00	69.00	95.00	84.00	11.72	14.01	13.
Sept. 1	77.83	72.83	75.00	70.00	97.00	83.00	12.29	14.54	13.
8	82.50	75.25	75.00	64.00	98.00	84.40	13.03	15.86	13.
15	74.50	70.83	75.17	66.00	98.00	86.00	10.44	13.78	13.
22	61.00	66.25	75.50	78.00	97.00	87.00	7.94	11.75	14.
29	57.67	62.50	75.50	64.00	97.00	89.67	5.73	10.38	14.
Oct. 6	56.17	48.00	75.50	61.00	97.33	91.00	5.28	8.95	14.
13	62.00	63.00	75.50	71.00	97.00	92.00	7.48	10.55	14.
20	56.50	58.33	75.50	64.50	97.00	89.00	5.38	9.03	14.
27	50.67	55.00	75.50	67.50	96.83	89.00	4.83	8.04	14.
Nov. 3	46.17	54.17	75.50	69.00	96.83	91.67	4.37	8.09	14.
10	48.50	53.00	75.50	68.00	96.80	93.60	4.51	7.50	15.
17	46.25	53.50	75.50	68.00	97.00	94.00	4.16	8.01	15.
24	46.00	49.50	75.50	62.50	96.67	96.00	4.09	6.63	15.
Dec. 1	35.20	43.83	75.50	64.00	96.00	96.00	2.61	8.06	15.
8	38.67	48.67	75.50	73.00	95.00	94.00	3.39	6.33	15.
15	24.75	50.83	75.50	56.00	97.00	94.00	1.47	6.47	15.
22	34.25	53.17	75.50	66.00	96.83	95.50	2.60	7.55	15.
29	33.00	55.67	75.50	65.50	98.20	95.00	2.46	8.35	15.
1913. Jan. 5	31.80	44.20	75.50	61.00	92.33	92.00	2.20	5.24	14.
12	25.33	45.00	75.50	91.00	93.00	91.00	2.46	5.44	14.
19	37.33	48.50	75.50	84.00	97.33	96.00	3.71	6.45	15.
26	30.67	46.00	75.50	84.00	97.50	97.33	2.87	5.91	15.
Feb. 2	26.33	43.00	75.33	71.00	97.00	98.50	2.00	5.28	15.
9	15.00	40.67	75.50	74.50	97.33	95.67	1.26	4.86	15.
16	25.83	40.00	75.50	69.00	96.00	97.00	1.85	4.68	15.
23	42.00	45.83	75.50	72.00	96.00	98.00	3.78	5.78	15.
Apr. 6	55.00	54.50	75.50	69.00	97.00	77.75	5.74	7.92	12.
13	43.00	50.00	75.50	88.67	95.83	79.83	4.82	6.69	13.
20	59.37	52.22	75.50	53.42	93.67	81.50	5.15	7.07	13.
27	59.83	56.16	75.50	60.00	96.83	83.83	5.87	8.38	13.

TABLE 6.—*Weekly averages of hygrometric records made in Illinois cooperative investigations*—Continued.

ASSUMPTION NO. 1 MINE, ASSUMPTION COAL & MINING CO., CHRISTIAN COUNTY—Continued.

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1913.	*°F.*	*°F.*	*°F.*	*Per cent.*	*Per cent.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gallons.*
May 4	64.37	56.00	75.50	46.67	97.17	87.50	5.30	8.32	14.27
11	58.00	57.67	75.50	53.00	97.00	89.33	4.88	8.80	14.57
18	67.17	60.22	76.00	61.50	97.00	93.50	7.68	9.55	15.74
25	62.33	61.01	76.00	63.50	95.33	93.17	6.75	9.71	15.43
June 1	74.00	62.33	76.00	67.10	96.40	95.40	10.42	10.37	15.81

EMPIRE NO. 2 MINE, CLARKE COAL & COKE CO., PEORIA, PEORIA COUNTY.

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1912.									
Feb. 11	4.75	38.17	....	74.50	87.00	....	0.77	3.97	....
25	37.00	46.25	63.50	74.00	88.00	97.00	3.23	4.77	10.73
Mar. 3	15.00	32.33	63.50	74.00	88.00	97.00	1.25	3.22	10.73
10	17.33	33.50	63.33	77.00	93.33	97.00	1.36	3.37	10.64
17	27.90	36.25	63.50	84.00	91.00	97.00	2.54	3.87	10.73
24	30.50	34.00	63.50	77.00	97.50	97.00	2.61	3.80	10.73
31	36.00	40.80	63.50	77.50	92.00	97.00	3.26	4.61	10.73
Sept. 29	50.00	54.75	65.33	75.00	74.50	97.00	5.23	6.13	11.40
Oct. 6	56.25	56.25	64.83	65.00	77.00	97.00	5.63	6.73	11.22
13	56.00	59.67	65.00	81.00	94.00	97.00	6.96	9.15	11.29
20	54.40	56.25	64.50	67.00	86.00	97.00	5.45	7.45	11.10
27	48.33	48.33	64.50	72.00	92.67	97.00	4.73	6.10	11.10
Nov. 3	44.50	48.00	64.00	70.00	94.00	97.00	4.02	6.12	10.92
10	45.60	47.33	63.89	68.00	96.00	97.00	4.06	6.04	10.88
17	42.00	44.62	63.83	72.00	96.00	97.00	3.78	5.54	10.86
24	43.75	42.67	63.33	63.00	96.00	97.00	3.52	5.35	10.64
Dec. 8	36.00	39.17	63.50	81.50	94.00	97.00	3.43	4.64	10.73
15	27.25	47.00	63.50	63.00	94.00	97.00	1.85	5.90	10.73
22	29.67	41.83	63.50	72.00	92.00	97.00	2.34	4.79	10.73
29	31.60	44.00	63.50	69.00	91.00	97.00	2.43	5.13	10.73
1913.									
Jan. 5	28.25	42.00	63.50	71.75	89.00	97.00	2.18	4.67	10.73
19	34.80	45.80	63.50	85.30	90.00	97.00	3.41	5.41	10.73
26	28.43	43.50	63.50	80.00	89.00	97.00	2.48	4.93	10.73
Feb. 2	25.40	40.20	63.50	76.60	89.00	98.00	1.75	4.37	10.84
9	15.88	36.50	63.50	70.00	93.00	97.00	1.23	3.99	10.73
Apr. 6	52.25	53.00	63.50	73.50	78.75	97.00	5.54	6.04	10.73
13	41.20	41.65	63.22	80.00	88.00	96.00	4.08	4.56	10.52
20	55.00	54.64	63.33	61.83	78.33	97.00	5.14	6.43	10.64
27	57.00	59.75	63.43	59.75	83.50	97.00	5.30	8.82	10.74
May 4	54.25	51.83	63.00	40.00	72.00	97.00	3.24	5.36	10.55

PEERLESS MINE, JONES & ADAMS COAL CO., SPRINGFIELD, SANGAMON COUNTY.

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1912.									
Mar. 10	23.00	29.17	57.17	62.50	96.00	92.00	1.52	3.07	8.23
17	33.50	32.00	58.67	81.00	93.00	92.50	3.07	3.36	8.73
24	37.71	38.00	58.67	78.00	88.00	90.00	3.49	3.99	8.47
31	42.57	42.25	60.25	78.50	93.33	92.00	4.20	4.94	9.13
Apr. 7	50.00	45.50	59.00	65.00	93.00	94.00	4.54	5.54	8.94
14	49.00	44.33	59.33	94.00	94.00	95.00	6.34	5.36	9.14
21	51.33	55.50	60.33	67.00	94.00	99.00	4.90	7.94	9.85
28	57.50	59.67	60.50	60.00	94.00	97.00	5.43	9.15	9.71
July 28	76.00	74.67	70.33	57.00	94.50	98.00	9.44	15.01	13.56
Aug. 4	74.67	79.00	66.88	69.50	84.00	96.50	11.04	15.29	11.93
Sept. 1	74.50	74.25	67.12	67.00	94.50	95.00	10.50	14.82	11.85
Oct. 6	61.67	61.00	64.25	83.00	92.17	96.00	8.64	9.38	10.89
13	55.50	60.00	65.25	58.00	89.00	96.00	4.90	8.67	11.25
27	53.50	54.25	63.50	65.00	99.50	99.00	5.13	8.05	10.95
Nov. 3	60.25	60.00	62.50	59.00	100.00	100.00	5.86	9.84	10.90
Dec. 8	45.25	47.00	61.50	79.00	100.00	100.00	4.66	6.28	10.35
22	46.00	45.00	60.00	69.00	81.00	89.00	2.90	4.73	8.76
1913.									
Jan. 5	26.00	24.37	60.25	71.50	90.00	94.00	1.99	2.32	9.33
12	26.33	18.67	61.00	93.00	98.00	88.00	2.62	1.95	8.96
Feb. 16	28.00	36.00	63.00	65.00	84.00	83.50	1.97	3.53	9.08
23	45.25	46.75	61.50	74.50	87.00	93.00	4.39	5.42	9.59
Mar. 2	19.00	20.50	59.50	76.25	100.00	94.00	1.54	2.16	9.10
9	23.75	21.00	60.75	60.00	100.00	94.00	1.50	2.22	9.49

TABLE 6.—*Weekly averages of hygrometric records made in Illinois cooperative investigations*—Continued.

WOODSIDE MINE, WOODSIDE COAL CO., SPRINGFIELD, SANGAMON COUNTY.

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1912.	°F.	°F.	°F.	*Per cent.*	*Per cent.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gallons.*
Feb. 25	28.00	44.37	59.37	76.50	78.00	80.50	2.33	4.46	7.76
Mar. 10	23.33	41.00	60.00	76.00	89.00	87.50	1.87	4.50	8.61
17	31.50	....	60.00	83.00	....	84.00	2.94	....	8.27
24	34.00	....	60.50	80.50	....	85.00	3.14	....	8.52
31	39.40	....	61.00	78.50	....	84.00	3.75	....	8.55
Sept. 29	52.50	54.50	66.67	62.50	87.00	91.33	4.76	7.10	11.21
Oct. 6	56.17	54.67	66.00	61.00	88.00	94.00	5.27	7.23	11.29
13	61.00	59.33	66.00	68.00	89.00	95.00	6.92	8.56	11.41
20	56.50	55.00	66.00	61.50	84.00	95.33	5.38	6.98	11.45
27	50.67	50.67	65.50	67.50	82.00	95.17	4.82	5.86	11.25
Nov. 3	46.17	49.17	65.00	69.00	88.67	95.25	4.21	6.01	11.07
10	48.83	49.50	64.50	72.00	90.00	91.33	4.82	6.17	10.45
17	43.67	47.20	64.25	66.33	86.20	89.80	3.53	5.45	10.19
24	46.00	46.33	64.00	62.50	86.17	90.00	4.00	5.28	10.12
Dec. 1	35.20	37.80	63.20	64.00	81.00	90.00	2.61	3.65	9.86
8	38.67	42.25	63.00	73.00	87.00	90.00	3.39	4.46	9.79
15	26.50	32.20	62.20	51.00	87.00	90.00	1.45	3.17	9.66
22	35.20	37.40	62.00	67.00	83.00	90.20	2.73	3.68	9.49
29	35.00	36.50	62.00	62.00	81.00	92.00	2.51	3.47	9.68
1913.									
Jan. 5	31.80	36.33	62.00	61.00	83.40	93.00	2.19	3.55	9.79
12	25.33	32.50	61.50	91.00	88.17	94.00	2.45	3.25	9.73
19	37.33	37.17	61.75	84.00	91.00	93.00	3.71	3.99	9.61
26	30.67	35.17	61.83	84.00	88.00	94.00	2.87	3.59	9.84
Feb. 2	30.50	32.40	62.00	73.00	76.00	92.00	2.47	2.79	9.68
9	15.00	23.17	61.50	74.50	98.00	93.00	1.26	2.40	9.63
16	23.00	28.00	61.00	69.50	97.00	92.00	1.69	2.94	9.36
22	....	41.33	61.00	----------	92.00	94.00	....	4.71	9.57

M. & C. MINE, MANUFACTURERS AND CONSUMERS COAL CO., DECATUR, MACON COUNTY.

1912.									
Feb. 18	40.75	42.75	62.50	72.00	92.00	85.00	3.62	4.96	9.11
25	34.00	42.50	62.60	79.00	88.00	84.25	3.08	4.70	9.04
Mar. 3	21.00	34.67	62.50	69.50	89.50	87.33	1.16	3.58	9.34
10	26.00	35.87	62.25	80.50	93.00	89.00	2.24	3.89	9.44
17	31.50	41.17	63.00	83.00	90.67	92.00	2.94	4.62	10.01
24	36.60	43.25	63.25	81.00	92.00	93.50	3.41	5.05	10.26
31	40.25	44.25	63.25	80.00	94.00	95.00	3.95	5.35	10.42
Oct. 13	61.50	65.00	68.00	72.50	91.50	88.50	7.51	10.64	11.37
20	56.50	56.83	66.83	61.50	91.00	89.00	5.38	8.04	10.98
27	52.00	54.00	66.25	67.00	84.00	91.67	6.51	6.74	11.11
Nov. 3	46.17	53.00	66.33	69.00	81.50	88.00	4.20	6.32	10.69
10	48.50	50.60	66.33	68.00	82.00	89.83	4.50	5.84	10.91
17	46.25	50.00	66.17	68.00	82.83	93.00	4.16	5.78	11.24
24	46.00	28.33	65.83	62.50	78.00	91.33	4.00	5.14	10.91
Dec. 1	35.20	42.00	65.00	64.00	82.33	91.00	2.61	4.32	10.58
8	38.67	47.50	65.00	73.00	86.00	89.67	3.33	5.50	10.42
15	31.67	40.00	65.00	54.00	82.00	95.00	1.93	4.00	11.04
22	34.25	42.50	65.00	66.00	87.00	95.00	2.60	4.65	11.04
29	33.00	41.00	65.00	65.50	84.00	95.00	2.46	4.25	11.04
1913.									
Jan. 5	31.80	42.50	65.00	61.00	85.00	95.00	2.19	4.54	11.04
12	26.40	39.50	65.00	91.00	89.00	95.00	3.75	4.26	11.04
19	37.33	41.25	65.00	83.50	89.33	95.00	3.68	4.56	11.04
26	23.00	38.00	65.00	83.50	89.50	95.00	2.03	4.05	11.04

TABLE 6.—*Weekly averages of hygrometric records made in Illinois cooperative investigations*—Continued.

MINE NO. 3, SALINE COUNTY COAL CO., HARRISBURG, SALINE COUNTY.

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1912.	°F.	°F.	°F.	*Per cent.*	*Per cent.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gallons.*
Feb. 18	....	33.83	51.58	....	93.00	87.83	....	3.61	5.74
25	....	37.75	52.75	....	93.66	94.00	....	4.20	7.22
Mar. 3	....	33.50	52.33	....	94.00	94.16	....	3.60	7.13
10	....	33.50	51.53	....	98.50	93.16	....	3.77	6.81
17	....	38.16	52.50	....	95.66	97.00	....	4.36	7.39
24	....	46.00	54.50	....	92.33	97.33	....	5.59	7.95
31	....	49.00	54.67	....	92.33	95.50	....	6.22	7.82
Apr. 7	64.50	58.56	55.83	59.00	90.33	98.00	6.75	8.75	8.37
14	61.30	57.06	56.33	60.00	61.00	96.80	6.17	8.11	8.41
21	....	49.67	56.50	....	86.00	91.00	....	5.87	7.96
28	60.75	58.08	57.22	61.00	84.00	94.50	6.16	7.61	8.47
May 5	65.75	63.89	57.92	81.00	89.17	96.17	9.65	9.99	8.82
12	70.50	68.44	59.11	71.00	83.00	97.00	9.97	10.80	9.26
19	62.80	60.83	59.00	69.00	86.20	99.20	7.46	8.72	9.44
26	76.70	71.60	59.90	61.00	85.20	99.20	10.33	12.28	9.73
June 2	72.88	73.67	61.17	73.00	86.00	99.50	10.96	13.06	10.23
9	71.70	69.00	61.00	65.00	84.00	98.80	9.40	11.09	10.06
16	71.83	71.58	60.83	68.00	80.33	99.00	9.85	12.00	10.02
23	65.25	67.17	62.33	86.00	91.67	99.00	10.08	11.45	10.54
30	68.00	70.67	62.17	85.00	93.00	99.50	10.91	13.01	10.51
July 7	74.62	75.50	63.75	88.00	96.00	99.50	13.96	15.66	11.10
14	77.50	77.50	64.83	83.00	95.50	100.00	14.42	16.59	11.56
21	75.17	77.56	66.00	82.00	96.00	99.00	13.22	16.70	11.89
28	78.08	78.83	66.11	77.00	86.50	99.00	13.60	15.67	11.92
Aug. 4	71.88	72.08	65.63	74.00	86.00	99.50	10.76	12.58	11.81
11	68.21	71.33	64.07	78.00	92.00	98.29	10.06	13.14	11.08
18	75.50	65.11	65.50	85.00	94.00	99.67	13.91	10.97	11.78
25	74.17	75.14	66.78	84.00	92.67	100.00	13.13	15.54	12.32
Sept. 1	75.50	78.60	66.93	79.00	90.00	100.00	12.89	16.18	12.39
8	78.75	78.42	67.83	75.00	89.00	98.00	13.54	15.91	12.50
15	71.45	74.40	67.30	73.00	89.60	99.60	10.46	14.12	12.49
22	62.88	66.67	64.83	83.00	90.50	99.00	8.99	11.12	11.44
29	49.58	58.33	63.44	79.00	83.67	99.33	6.42	8.88	10.97
Oct. 6	54.92	61.50	63.61	80.00	85.33	97.00	6.63	8.83	10.77
13	66.69	70.58	63.08	76.00	78.75	99.00	9.34	10.78	10.80
20	44.38	59.17	63.00	78.00	89.00	100.00	4.46	8.52	10.88
27	49.40	55.30	63.00	78.00	86.00	98.00	5.33	7.21	10.65
Nov. 3	47.13	54.17	63.08	77.00	84.00	98.33	4.85	6.78	10.72
10	49.19	52.50	62.88	70.00	82.50	98.00	4.75	6.29	10.62
17	35.13	45.50	60.42	68.50	70.00	96.00	2.79	4.17	9.58
24	45.75	54.83	60.50	61.00	94.33	94.83	3.66	7.77	9.34
Dec. 1	36.62	46.58	59.83	67.00	94.00	93.00	2.88	5.87	9.11
8	41.50	49.80	60.00	80.00	91.00	95.00	4.12	6.31	9.35
15	27.75	38.50	59.89	63.00	86.50	94.00	1.89	3.99	9.22
22	34.50	42.28	60.00	76.00	81.50	94.00	3.02	4.32	9.25
29	29.83	38.78	59.00	72.33	84.00	94.00	2.39	3.92	8.94

EQUALITY MINE, GALLATIN COAL & COKE CO., GALLATIN COUNTY.

1912.									
Sept. 29	53.50	64.50	64.50	81.00	90.40	95.40	6.39	10.34	10.91
Oct. 6	54.91	64.67	64.08	80.00	90.67	96.67	6.63	10.43	10.91
13	64.91	68.00	65.08	72.00	89.00	96.00	8.35	11.41	11.19
20	52.75	63.00	63.00	78.50	88.50	96.00	5.93	9.63	10.45
27	47.63	59.25	64.00	79.00	89.00	99.00	5.06	8.52	11.14
Nov. 3	47.13	57.25	63.75	77.00	97.00	97.00	4.84	8.70	10.82
10	47.91	57.30	63.83	68.00	98.00	97.00	4.41	8.80	10.85
17	44.25	56.50	64.00	77.00	88.67	97.67	4.38	7.75	10.99
24	45.75	55.40	63.61	61.00	78.00	96.00	3.66	6.57	10.65
Dec. 1	35.90	49.80	62.83	66.00	76.00	95.60	2.76	5.27	10.34
15	27.75	45.00	60.92	63.00	76.00	98.00	1.89	4.44	9.95
22	34.50	48.50	62.08	76.00	79.00	98.00	3.02	5.23	10.34
29	31.05	47.30	61.40	73.00	79.00	98.00	2.53	5.01	10.11

TABLE 6.—*Weekly averages of hygrometric records made in Illinois cooperative investigations*—Continued.

EQUALITY MINE, GALLATIN COAL & COKE CO., GALLATIN COUNTY—Continued.

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1913.	*°F.*	*°F.*	*°F.*	*Per cent.*	*Per cent.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gallons.*
Jan. 5	34.91	48.75	61.50	70.00	83.00	95.00	2.83	5.54	9.83
12	30.10	46.08	62.60	91.00	87.00	97.00	3.01	5.29	10.42
19	40.91	50.83	62.56	84.67	88.00	96.00	4.37	6.32	10.30
26	37.08	51.91	61.83	83.00	93.00	97.00	3.63	6.94	10.15
Feb. 2	31.17	50.78	61.39	71.00	89.00	96.00	2.48	6.38	9.90
9	18.00	44.33	60.50	66.67	85.00	94.00	1.28	4.85	9.41
16	30.33	46.33	58.94	56.33	87.00	95.00	1.89	5.33	9.02
23	44.67	54.67	61.22	77.33	91.00	95.67	4.47	7.47	9.82
Mar. 2	31.40	47.43	58.47	86.00	87.00	96.00	3.03	5.55	8.97
9	34.50	49.67	59.22	73.50	82.00	95.33	2.92	5.66	9.14
16	45.50	55.45	59.67	75.00	90.00	96.00	4.47	7.59	9.35
23	45.75	55.83	58.83	75.33	90.00	94.50	4.51	7.69	8.94
30	55.50	54.83	58.75	95.50	89.50	93.50	8.07	7.39	8.68

HART & WILLIAMS NO. 1 MINE, HART & WILLIAMS COAL CO., FRANKLIN COUNTY.

1912.									
Feb. 11	....	26.00	60.33	....	97.00	85.00	....	2.70	8.46
18	....	34.50	60.83	....	91.50	90.00	....	3.64	9.11
25	....	38.50	61.50	....	98.00	92.00	....	4.52	9.52
Mar. 3	....	33.00	61.00	....	95.00	90.00	....	3.57	9.16
10	....	33.50	60.50	....	99.00	91.00	....	3.79	9.11
17	....	40.00	61.00	....	100.00	92.00	....	4.88	9.36
24	....	44.00	61.50	....	99.00	92.00	....	5.58	9.52
31	....	48.67	62.00	....	99.50	89.33	....	6.63	9.40
June 9	74.67	65.83	66.00	63.00	98.33	99.50	10.01	11.75	11.95
16	71.50	76.33	66.50	74.00	99.00	99.33	10.63	16.58	12.12
23	62.00	69.33	66.33	85.00	99.00	99.75	9.00	13.26	12.11
30	68.00	74.13	66.63	85.00	93.33	100.00	10.91	15.51	12.27
July 7	74.67	79.50	67.75	86.00	100.00	100.00	13.66	18.49	12.72
14	77.40	81.50	69.00	84.00	100.00	100.00	14.54	19.97	13.25
21	75.60	80.20	69.20	80.00	99.20	100.00	13.09	18.73	13.34
28	78.33	82.63	68.75	76.00	100.00	100.00	13.54	20.36	13.38
Aug. 4	71.75	75.40	69.00	72.00	99.33	100.00	10.42	16.15	13.25
11	67.75	75.25	68.50	83.00	100.00	100.00	10.47	16.18	13.04
18	76.75	83.25	70.13	86.50	100.00	100.00	14.67	20.75	13.74
Sept. 22	62.67	68.00	68.67	84.00	99.33	100.00	9.04	12.74	13.11
29	51.75	60.14	68.00	81.00	100.00	100.00	6.01	9.89	12.82
Oct. 6	56.33	60.56	66.25	79.00	99.28	99.56	6.86	9.96	12.06
13	64.50	64.50	67.50	70.00	99.67	99.50	8.00	11.40	12.56
20	52.67	57.00	67.00	79.00	100.00	100.00	6.04	8.89	12.42
27	47.50	57.33	66.67	78.00	99.83	100.00	4.99	8.98	12.28
Nov. 3	44.50	50.00	66.00	78.00	100.00	100.00	4.48	6.98	12.02
10	53.83	62.00	65.83	76.00	100.00	100.00	6.06	10.53	11.95
17	38.00	53.33	65.50	70.00	96.00	95.67	3.17	7.53	11.31
24	45.75	54.25	65.50	61.00	97.00	98.50	3.66	7.83	11.64
Dec. 1	36.00	42.67	63.33	66.00	100.00	99.00	2.78	5.37	10.89
8	41.75	45.50	63.75	85.00	100.00	100.00	4.42	5.95	11.16
15	33.00	37.80	63.33	62.00	99.33	100.00	2.33	4.47	11.00
22	33.00	41.47	64.00	75.00	100.00	100.00	2.82	5.15	11.25
29	33.00	38.00	63.50	77.00	100.00	99.00	2.89	4.53	10.95
1913.									
Jan. 5	34.00	38.75	64.83	76.00	98.67	99.25	2.97	4.60	11.47
12	33.00	36.00	63.00	89.00	100.00	100.00	3.34	4.21	10.88
19	41.50	46.50	63.50	86.33	100.00	100.00	4.45	6.17	11.07
26	37.33	41.00	64.00	85.00	100.00	99.40	3.76	5.06	11.18
Feb. 2	31.17	37.67	63.50	71.00	99.50	99.67	2.48	4.45	10.97
9	12.75	25.50	61.00	73.50	100.00	89.00	1.12	2.72	9.06
Apr. 20	58.14	57.78	64.22	60.28	89.42	97.50	5.58	8.18	11.24
27	60.33	63.33	65.13	67.00	97.33	96.33	6.67	10.71	11.25
May 4	63.00	62.33	65.17	58.33	94.50	99.17	6.35	10.06	11.59
11	58.71	60.07	64.95	58.56	99.70	99.70	5.45	9.84	11.58
18	68.80	70.30	66.44	74.00	94.00	100.00	9.74	12.99	12.21
25	61.17	52.43	66.72	68.50	100.00	99.67	7.01	7.55	12.28
June 1	74.08	67.08	67.13	71.83	99.00	99.67	11.20	12.30	12.38

TABLE 6.—*Weekly averages of hygrometric records made in Illinois cooperative investigations*—Continued.

BIG MUDDY MINE, CHICAGO & BIG MUDDY COAL CO., WILLIAMSON COUNTY.

Week ending—	Temperature.[a]			Relative humidity.[a]			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1912.	°*F.*	°*F.*	°*F.*	*Per cent.*	*Per cent.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gallons.*
Oct. 6	63.17	71.17	73.50	77.00	75.00	76.00	8.31	10.66	11.63
13	64.92	67.00	77.75	72.00	86.67	89.67	8.35	10.76	15.70
20	52.75	57.17	78.08	78.08	91.00	96.67	5.93	8.14	17.10
27	47.62	52.42	77.83	79.00	93.50	96.67	5.06	7.09	16.96
Nov. 3	47.12	56.67	77.67	77.00	95.33	96.67	4.84	8.38	16.88
10	47.88	51.42	77.50	68.00	100.00	100.00	4.39	7.34	17.37
17	44.25	49.67	77.58	77.00	94.67	100.00	4.38	6.53	17.41
24	45.75	47.00	76.75	61.00	100.00	100.00	3.66	6.28	16.95
Dec. 1	35.90	39.40	76.25	66.00	100.00	100.00	2.76	4.77	16.70
8	44.83	46.00	76.50	74.67	96.00	100.00	4.46	5.82	16.83
15	27.75	31.67	75.50	63.00	100.00	100.00	1.89	3.57	16.31
22	34.50	38.75	76.00	76.00	85.00	100.00	3.02	3.89	16.57
29	30.10	36.00	75.00	75.40	87.00	95.00	2.51	3.66	15.25
1913.									
Jan. 5	33.45	40.40	76.75	71.00	86.00	97.00	2.71	4.32	16.46
12	30.00	36.13	75.75	91.00	91.00	97.00	2.99	3.84	15.94
19	40.92	41.17	76.08	84.67	92.00	97.50	4.37	4.68	16.19
26	37.08	41.50	75.75	83.00	93.00	98.00	3.63	4.79	16.31
Feb. 2	31.17	37.17	76.00	71.00	92.00	98.00	2.48	4.04	16.24
9	18.00	27.00	75.00	66.67	91.00	97.00	1.28	2.64	15.57
16	26.90	30.75	75.00	66.80	85.00	98.00	1.93	2.91	15.73
Mar. 2	25.50	31.50	76.00	87.00	89.00	99.00	2.36	3.15	16.40
9	33.10	35.35	76.80	76.80	92.00	91.80	2.90	3.78	15.60
16	45.50	46.13	76.46	75.00	87.00	97.00	4.47	5.29	15.89
23	45.75	43.33	77.42	75.33	90.33	96.33	4.51	4.96	16.69
Apr. 6	56.38	47.50	79.25	45.25	87.50	95.50	3.94	5.59	17.52
13	49.75	40.25	78.50	84.33	93.67	98.17	5.83	6.37	17.59
20	59.08	52.11	78.50	60.50	93.33	100.00	5.97	7.52	17.92
27	60.33	54.92	78.50	67.00	96.00	99.00	6.67	7.96	17.25
May 4	55.17	51.67	79.33	60.33	95.67	100.00	5.04	7.04	18.39

MINE NO. 3, SUPERIOR COAL CO., MACOUPIN COUNTY.

1912.									
Feb. 18	44.00	39.83	50.17	67.50	95.50	98.50	3.81	4.63	6.56
25	34.86	35.00	51.72	77.72	92.00	94.00	3.13	3.73	6.97
Mar. 3	29.33	32.67	52.55	73.00	92.00	94.00	2.35	3.41	7.17
17	31.00	35.33	51.33	95.00	91.00	94.00	3.29	3.73	6.88
July 7	80.00	75.50	74.00	78.00	96.00	98.00	14.64	15.66	15.24
14	79.50	81.33	67.25	73.25	95.00	95.75	13.54	18.59	11.99
Oct. 6	69.67	68.00	64.33	58.00	99.00	91.67	7.85	12.70	10.43
13	67.60	68.80	65.33	66.00	100.00	95.33	8.41	13.16	11.20
20	62.75	65.42	64.56	60.50	95.00	94.00	6.53	11.20	10.77
27	52.50	57.37	64.00	58.25	95.50	84.00	4.44	8.60	9.45
Nov. 3	40.33	48.89	62.17	73.33	98.00	85.67	4.06	6.81	9.07
10	50.00	60.42	63.17	75.00	98.50	90.50	5.23	9.83	9.90
17	46.00	50.00	63.00	74.00	100.00	85.00	4.48	6.98	9.25
24	55.67	53.67	61.83	49.00	99.33	89.33	4.12	8.16	9.35
1913.									
Jan. 5	40.50	47.50	57.96	50.00	92.00	93.00	2.48	5.83	8.48
12	26.00	46.00	56.00	89.67	98.00	92.67	2.49	5.63	7.96
19	36.00	44.33	56.83	75.50	91.50	92.00	3.18	5.22	8.13
26	37.33	44.67	57.00	81.33	94.00	94.00	3.60	5.48	8.36
Feb. 2	20.00	41.33	58.33	58.00	91.00	89.00	1.23	4.66	8.28
Apr. 6	57.33	60.67	58.92	62.83	96.00	94.33	5.65	9.66	8.95
13	46.00	48.00	59.57	83.00	99.43	94.28	5.03	6.47	9.15
20	59.86	60.73	58.81	52.57	99.43	97.00	5.15	10.03	9.17
27	67.00	62.22	59.67	49.67	98.67	98.00	6.10	10.46	9.54

MINE NO. 2, MADISON COAL CORPORATION, GLEN CARBON, MADISON COUNTY.

1912.									
Mar. 3	26.00	32.53	65.50	64.50	84.00	92.00	1.79	3.10	10.87
10	27.40	33.00	65.50	78.20	85.00	92.00	2.32	3.19	10.87
17	33.67	35.78	65.46	80.00	87.00	93.00	3.08	3.63	10.97
24	39.33	39.67	65.39	80.00	88.00	93.00	3.83	4.27	10.95
31	42.80	42.60	65.42	79.40	88.00	93.00	4.29	4.72	10.96

[a] In the Big Muddy mine, the temperature and relative humidity at the return shaft was higher than normal because of heat radiating from uncovered steam pipes along the main entry and because of exhaust steam from pumps.

TABLE 6.—Weekly averages of hygrometric records made in Illinois cooperative investigations—Continued.

MINE NO. 2, MADISON COAL CORPORATION, GLEN CARBON, MADISON COUNTY—Cont

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1912.	°F.	°F.	°F.	Per cent.	Per cent.	Per cent.	Gallons.	Gallons.	Gallons
May 19	57.67	62.00	68.67	64.33	70.00	94.33	5.85	7.37	12.3
26	74.84	77.17	68.67	60.33	65.00	94.33	9.63	11.17	12.3
June 2	70.60	73.07	68.93	68.00	78.00	94.00	9.50	11.77	12.4
9	68.20	72.50	68.50	62.00	67.00	95.00	7.90	10.16	12.3
16	73.50	76.00	68.67	64.50	76.67	94.50	9.88	12.70	12.3
23	64.67	68.58	68.83	76.00	78.00	94.00	7.59	10.19	12.3
30	76.00	77.33	69.08	68.50	70.00	94.00	11.35	12.10	12.4
July 7	75.33	76.67	69.00	83.67	77.67	95.67	13.48	13.11	12.
28	81.60	82.67	68.67	68.60	72.00	95.00	12.53	14.48	12.4
Aug. 4	75.40	77.60	69.10	57.60	64.00	95.00	9.37	11.33	12.
11	71.17	73.45	69.28	72.50	73.50	95.00	10.30	11.24	12.7
18	77.33	78.22	69.83	77.00	83.00	95.67	13.01	14.74	13.1
25	79.00	80.67	70.50	66.67	75.00	95.33	12.13	14.38	13.2
Sept. 1	80.33	84.00	70.56	69.33	70.00	94.00	13.06	14.86	13.1
8	85.40	84.60	71.00	57.40	68.00	95.60	12.72	14.70	13.5
15	78.80	79.13	70.87	60.60	66.60	95.00	10.88	12.17	13.3
29	60.25	62.25	70.33	62.00	72.00	95.00	6.15	7.64	13.1
Oct. 6	65.60	66.93	69.90	56.80	67.00	95.60	6.73	8.30	13.0
13	68.40	70.10	70.10	66.40	75.00	96.67	8.63	10.30	13.2
20	61.00	62.47	69.78	56.67	72.67	98.00	5.73	8.20	13.3
27	54.00	55.19	69.28	59.50	77.33	94.75	4.78	6.46	12.6
Nov. 3	47.00	52.83	68.83	64.17	75.33	94.33	4.03	5.81	12.4
10	52.60	52.20	68.50	68.60	84.60	94.40	5.24	6.38	12.3
17	50.33	53.50	68.67	72.00	71.50	94.50	5.09	5.64	12.3
24	52.17	52.00	68.25	56.33	71.00	94.67	4.24	5.32	12.2
Dec. 1	38.60	44.60	67.17	61.18	80.00	95.40	2.86	4.61	11.9
8	42.33	43.83	67.50	65.67	85.00	95.50	3.48	4.76	12.0
15	32.17	37.02	66.89	50.17	76.00	95.00	1.86	3.32	11.7
22	38.33	39.75	66.67	62.17	80.00	95.00	2.83	3.87	11.6
29	38.00	39.69	66.07	61.00	79.40	95.60	2.76	3.82	11.5
1913.									
Jan. 5	40.00	40.50	66.60	50.20	87.00	95.00	2.46	4.32	11.6
12	26.00	35.42	65.83	89.67	91.00	96.00	2.49	3.73	11.4
19	43.50	43.22	66.11	77.00	87.50	95.50	4.27	4.80	11.5
26	35.50	37.67	66.00	81.33	91.00	95.00	3.36	4.07	11.4
Feb. 2	30.33	35.92	66.00	65.67	89.00	95.00	2.21	3.72	11.4
9	21.33	25.32	65.33	68.67	85.00	95.00	1.54	2.35	11.1
16	31.67	33.06	65.67	64.00	88.00	94.00	2.28	3.31	11.1
23	49.33	48.25	66.17	66.67	87.00	95.00	4.53	5.71	11.4
Mar. 2	31.25	35.33	65.67	62.17	88.50	95.25	2.17	3.75	11.2
Apr. 6	60.50	61.33	67.00	62.25	83.30	96.25	6.23	8.91	11.9
13	46.33	50.11	67.17	83.67	88.67	94.83	5.13	6.21	11.8
20	60.83	60.67	67.45	53.83	81.50	94.83	5.45	8.21	11.9
27	61.00	63.28	68.39	64.17	80.67	94.83	6.53	8.86	12.3
May 4	66.17	67.39	68.25	48.50	66.67	95.00	5.86	8.37	12.2
11	58.80	63.07	68.67	51.00	76.20	95.00	4.84	8.31	12.4
18	68.00	68.08	68.92	50.50	82.50	95.00	6.44	10.61	12.5

LUMAGHI NO. 2 MINE, LUMAGHI COAL CO., COLLINSVILLE, MADISON COUNTY.

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1912.									
Apr. 14	58.00	62.00	63.00	48.00	79.00	83.00	4.42	8.31	9.
21	49.00	53.40	62.00	73.00	92.40	91.40	4.77	7.23	9.
28	61.17	56.50	62.00	67.67	86.00	90.67	7.26	7.52	9.5
May 5	65.67	60.58	62.83	75.33	90.00	92.00	8.96	9.06	9.9
12	73.00	67.00	63.00	70.50	93.00	93.50	10.62	11.55	10.1
19	57.67	59.00	63.70	64.33	88.00	95.00	5.85	8.37	10.5
26	74.60	71.90	64.70	63.40	91.00	96.00	10.05	13.23	11.
June 2	67.17	69.50	65.92	70.00	91.00	95.33	11.71	12.25	11.4
9	66.67	68.00	65.58	60.33	89.50	96.33	7.41	11.48	11.4
16	73.00	71.00	65.30	67.60	92.00	96.60	10.18	13.00	11.7
23	64.67	68.33	66.67	76.00	94.50	95.33	7.59	12.25	11.5
30	75.60	72.40	66.40	67.60	93.00	95.00	11.06	13.74	11.5
July 7	79.00	76.75	64.75	78.25	93.00	96.50	14.24	15.77	11.1
14	79.67	78.25	68.83	73.17	95.00	96.00	13.57	16.90	12.
21	75.67	77.92	69.25	77.50	93.67	98.33	12.80	16.48	13.
28	80.67	79.50	69.42	67.92	94.33	98.00	13.53	17.42	13.1
Aug. 4	73.33	74.50	68.75	58.67	95.67	98.00	8.94	15.12	12.
11	71.17	74.00	68.00	72.50	94.33	98.00	10.30	14.67	12.5
18	77.33	76.00	68.75	77.00	96.33	98.50	13.01	15.96	12.9
25	79.00	78.25	70.00	66.67	94.00	98.00	12.13	16.72	13.4

TABLE 6.—*Weekly averages of hygrometric records made in Illinois cooperative investigations*—Continued.

LUMAGHI NO. 2 MINE, LUMAGHI COAL CO., COLLINSVILLE, MADISON COUNTY—Contd.

Week ending—	Temperature.			Relative humidity.			Water in 100,000 cubic feet.		
	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.	Outside air.	Intake air.	Return air.
1912.	*°F.*	*°F.*	*°F.*	*Per cent.*	*Per cent.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gallons.*
Sept. 1	80.33	79.25	69.67	69.33	96.00	98.00	13.06	17.61	13.27
8	85.40	77.20	74.40	57.40	96.80	95.40	12.72	16.66	15.03
15	77.67	77.00	70.00	60.00	91.00	97.67	10.48	15.56	13.38
22	66.25	76.25	69.38	78.25	92.75	96.00	9.48	15.49	12.88
29	56.33	62.00	67.00	56.00	90.00	98.00	4.87	9.47	12.17
Oct. 6	63.50	63.25	67.00	56.33	90.50	96.33	6.73	9.93	11.96
13	66.33	65.42	66.42	65.33	94.00	96.50	7.94	11.06	11.75
20	57.50	62.00	66.00	46.00	90.00	95.00	4.18	9.47	11.41
27	58.00	59.00	64.50	56.00	91.00	95.00	5.15	8.66	10.87
Nov. 24	54.00	56.00	62.50	62.00	91.00	86.00	4.97	7.82	9.20
Dec. 1	43.00	48.00	62.00	66.00	86.00	89.00	3.59	5.48	9.37
8	42.33	51.25	64.00	65.67	91.00	90.00	3.48	6.55	10.12
15	32.17	39.20	60.50	50.17	88.00	95.00	1.86	4.17	9.51
22	38.33	45.00	60.58	62.17	88.00	94.00	2.83	5.14	9.44
29	38.00	43.75	60.00	57.50	86.50	94.00	2.58	4.83	9.25
1913.									
Jan. 5	39.25	42.75	60.00	50.75	86.00	94.00	2.41	4.63	9.25
12	27.20	39.40	59.60	91.60	90.00	95.00	2.68	4.29	9.23
19	43.50	45.25	59.50	77.00	93.00	96.00	4.27	5.49	9.29
26	42.75	42.50	61.00	85.25	91.00	94.00	4.59	4.95	9.57
Feb. 2	30.33	42.33	60.00	65.67	91.00	94.00	2.21	4.83	9.25
9	21.33	30.50	58.50	38.67	96.00	94.00	1.54	3.25	8.80
16	31.67	36.67	58.00	64.00	90.00	94.00	2.28	3.88	8.65
23	49.60	37.70	59.00	63.40	91.00	94.00	4.36	4.06	8.95
Mar. 2	28.17	38.67	58.75	77.00	89.00	95.50	2.36	4.13	9.01
9	37.67	41.25	58.17	60.67	90.50	94.50	2.71	4.63	8.44
16	48.40	50.50	59.30	72.40	93.40	93.40	4.60	6.60	8.98
23	28.17	46.75	59.42	82.67	87.33	94.00	2.53	5.43	9.07
30	44.25	48.25	60.00	86.50	90.25	94.00	4.88	5.93	9.25
Apr. 6	57.20	55.70	60.70	62.00	89.00	92.40	5.55	7.56	9.31
13	46.40	53.00	60.80	85.80	91.20	90.00	5.27	7.07	9.10
20	60.83	58.10	61.30	53.67	89.60	89.00	5.45	8.22	9.15
27	61.00	61.75	61.83	64.17	90.17	91.50	6.53	9.41	9.57
May 4	68.00	62.50	62.87	49.00	89.00	89.75	6.28	9.52	9.72
11	62.60	63.40	63.40	52.20	86.20	89.80	5.61	9.50	9.90

MINE NO. 5, CENTRALIA COAL CO., MARION COUNTY.

1912.									
Feb. 18	46.00	46.00	61.67	63.00	79.00	89.00	3.81	4.79	9.26
25	34.33	39.00	57.92	74.83	89.00	91.00	2.96	4.18	8.34
Mar. 3	25.83	34.00	59.22	70.83	95.00	90.00	1.94	3.61	8.84
10	26.67	36.00	57.64	79.50	93.00	88.00	2.27	3.91	8.00
17	33.67	38.67	60.89	80.00	91.50	88.00	3.06	4.25	8.93
24	39.33	44.39	62.30	80.33	87.00	86.00	3.87	4.98	9.14
31	44.50	48.75	63.00	78.67	90.67	86.00	4.52	6.46	9.36
May 12	65.00	69.00	68.00	76.50	96.00	85.00	8.89	12.72	10.91
19	56.00	60.00	67.35	65.40	83.00	85.00	5.62	8.17	10.67
26	74.83	76.00	68.25	60.33	64.00	87.00	9.63	10.60	11.25
June 2	70.60	74.66	68.66	68.00	80.00	88.60	9.50	12.71	11.60
9	66.75	74.00	68.25	63.50	63.75	88.75	7.72	9.92	11.48
16	73.80	78.17	68.60	68.40	62.00	87.00	10.58	11.00	11.38
23	64.67	70.00	68.78	76.00	75.00	89.00	7.59	10.26	11.82
30	75.80	77.50	68.70	68.00	65.00	86.00	11.19	11.29	11.41
July 21	75.00	81.00	69.20	78.40	76.00	95.00	12.58	14.84	12.67
28	81.60	83.33	69.25	68.60	69.00	96.00	13.53	14.36	12.82
Aug. 4	73.33	76.00	68.75	58.67	68.00	94.00	8.94	11.27	12.20
11	71.17	75.33	68.35	72.50	76.00	93.00	10.30	12.33	12.06
18	77.33	80.33	69.00	77.00	92.00	94.00	13.01	17.45	12.46
25	79.00	81.00	69.55	66.67	79.67	95.00	12.13	15.42	12.81
Sept. 1	80.33	84.17	69.33	69.33	78.00	94.83	13.06	16.66	12.70
8	85.40	87.40	69.80	57.40	80.60	95.00	12.72	18.97	12.91
15	77.67	82.11	69.50	60.00	91.00	94.00	10.48	18.24	12.66
22	63.83	70.50	69.00	80.00	88.67	93.00	8.95	12.33	12.32
29	59.67	65.17	68.05	59.50	81.00	90.50	5.79	9.47	11.62
Oct. 6	63.50	67.42	67.42	56.33	75.50	88.00	6.73	9.50	11.08
13	66.33	72.33	68.17	65.33	75.00	87.67	7.94	11.06	11.54
20	61.00	65.56	67.92	56.33	73.00	87.00	5.67	8.64	11.13

DISCUSSION OF HYGROMETRIC RECORDS.

Comments on the records contained in Table 6 and deductions from them are given below.

TEMPERATURE AND RELATIVE HUMIDITY OF OUTSIDE AIR DURING PERIOD OF TESTS.

During the year February 1, 1912, to January 31, 1913, the total range of temperature and the maximum and minimum temperatures at two Illinois weather-bureau stations were as shown below:

Annual temperature range at two stations.

Weather-bureau station.	Maximum temperature.		Minimum temperature.		Total annual range.
	° F.	Date.	° F.	Date.	° F.
LaSalle	97	Sept. 8, 1912	−7	Feb. 4, 1912	104
Springfield	96	.....do.......	−9	.....do.......	105

The mean monthly temperatures for the same period were as given below:

Mean monthly temperatures during period of tests.

Date.	Temperature at LaSalle weather-bureau station.	Temperature at Springfield weather-bureau station.	Date.	Temperature at LaSalle weather-bureau station.	Temperature at Springfield weather-bureau station.
1912.	° *F.*	° *F.*	1912.	° *F.*	° *F.*
February	20.4	24.6	September	66.0	68.2
March	28.0	32.1	October	53.7	57.0
April	50.7	54.3	November	41.8	43.4
May	64.1	66.4	December	32.6	34.6
June	68.1	69.8			
July	74.6	77.1	1913.		
August	70.8	73.9	January	27.6	30.8

The mean annual temperature during 1912 in each county in Illinois that produces coal is given below:

Mean annual temperature of counties producing coal.

Name and County.	Mean annual temperature for 1912.	Name of County.	Mean annual temperature for 1912.
	° *F.*		° *F.*
Bureau, Grundy, Henry, LaSalle, Rock Island, Will	48	Calhoun, Christian, Green, Macoupin, Montgomery, Scott, Shelby	52
Livingston, Marshall, Mercer, Peoria, Putnam, Stark, Warren	49	Bond, Jefferson, Jersey, Marion	53
Fulton, Hancock, McDonough, McLean, Tazewell, Vermilion, Woodford	50	Clinton, Franklin, Madison, St. Clair, Washington, Williamson	54
Brown, Edgar, Logan, Macon, Menard, Moultrie, Sangamon, Schuyler	51	Jackson, Perry, Randolph, Saline	55
		Gallatin	56

The average maximum and minimum daily temperatures and the average daily range in temperature at the La Salle weather bureau station for the 12 months beginning February, 1912, are below:

Average daily temperature, by months, at La Salle station.

Month.	Average daily temperature.			Month.	Average daily temperature.		
	Maximum.	Minimum.	Range.		Maximum.	Minimum.	Range.
1912.	°*F.*	°*F.*	°*F.*		°*F.*	°*F.*	°*F.*
February	26.9	13.6	13.3	October	65.3	42.7	22.6
March	35.8	20.0	15.8	November	50.7	33.0	17.7
April	60.6	40.7	19.9	December	40.5	24.4	16.1
May	74.8	53.1	21.7	1913.			
June	79.7	56.7	23.0	January	35.5	19.0	16.5
July	85.5	63.4	22.1	Average			18.9
August	80.8	61.4	18.4				
September	76.6	56.4	20.2				

It is significant that the greatest diurnal range in temperature occurred in the spring and autumn seasons and coincides with the period of the greatest frequency of roof falls in the mines.

In a tabulation of the linear expansion of solids at ordinary temperatures, prepared mostly by the British Board of Trade, the increase in unit length for a temperature change of 1° F. is given as follows:

Slate	0.00000577
Sandstone	.00000652

In an entry 10 feet wide, therefore, if the lower layer of the roof assumes the temperature of the air, temperature changes will cause expansion as follows:

Mine-roof expansion due to temperature changes.

Kind of mine roof.	Lineal expansion due to temperature changes.	
	Diurnal change 23°.	Annual change 105°.
	Inches per 10 feet.	*Inches per 10 feet.*
Slate	0.0159	0.0727
Sandstone	.0180	.0822

If, therefore, this daily shrinkage and swelling of the exposed roof layers takes place as a result of temperature changes in addition to the effect caused by changes in the moisture content of the air, there need be little wonder that roof falls are particularly prevalent in the intake air courses.

In many mines in which the roof is friable, the application of the method for maintaining a uniform temperature of the intake air suggested on pages 52 to 61 would be especially desirable.

Table 7 following gives for each two-hour interval the mean monthly temperatures at the LaSalle weather-bureau station for the 16 months, February, 1912, to May, 1913.

Table 8 gives for each two-hour interval the mean monthly relative humidity at the same station during the same period.

TEMPERATURE AND RELATIVE HUMIDITY OF THE RETURN MINE AIR DURING THE TESTS.

The temperature and relative humidity of the upcast at any mine should be approximately constant throughout the year. In the return air at the Sherrard mine, during 52 weeks, the maximum fluctuation in temperature was 1¾° and in relative humidity 6 per cent. This draws attention to the fact that (except in new mines) the efficiency of the main entry stoppings may be measured in terms of the seasonal changes in the temperature of the upcast.

Table 9 shows the temperature and relative humidity of the return air at three long-wall and five room-and-pillar mines in which three observations were made daily during a period of a year.

Table 10 has been compiled from the data in Tables 7, 8, and 9, in order to show the amount of moisture extracted from the average long-wall and room-and-pillar mine by the ventilating currents, when the quantity of air per minute is 100,000 cubic feet. For any other volume of ventilation, the figures in Table 10 will vary directly.

TABLE 7.—*Temperature of the outside air at the LaSalle weather-bureau station, by months, from February, 1912, to May, 1913.*

Month.	2 a. m.	4 a. m.	6 a. m.	8 a. m.	10 a. m.	Noon.	2 p. m.	4 p. m.	6 p. m.	8 p. m.	10 p. m.	Night.	Average.
1912.													
February	18.9	17.7	16.8	16.9	19.1	22.6	24.5	24.8	22.9	21.3	20.5	19.2	20.4
March	24.1	23.2	22.7	23.9	28.2	31.3	33.4	33.6	31.8	29.6	28.0	26.2	28.0
April	45.1	43.7	43.3	47.3	52.7	56.3	58.5	58.3	56.3	51.6	48.5	47.0	50.7
May	57.5	55.7	56.0	61.7	66.2	69.0	72.5	72.3	70.3	65.7	62.4	60.1	64.1
June	59.9	58.2	59.2	66.4	72.4	76.0	77.9	77.2	74.0	68.9	65.1	62.3	68.1
July	67.6	65.7	66.3	72.0	77.9	81.3	83.6	83.2	80.9	75.4	71.5	69.3	74.6
August	64.1	63.1	63.0	68.5	74.3	77.1	78.6	78.7	76.3	71.4	68.6	66.1	70.8
September	60.6	59.2	58.3	63.7	70.6	73.6	74.4	73.4	69.6	65.5	62.8	60.6	66.0
October	46.9	45.9	45.0	49.1	57.6	62.0	64.0	62.9	57.6	53.9	50.8	48.7	53.7
November	38.4	37.2	35.9	36.7	42.4	46.1	48.5	48.1	44.8	42.3	41.1	39.8	41.8
December	31.2	30.1	28.8	28.4	31.4	35.1	37.3	36.9	34.6	33.5	32.4	31.2	32.6
1913.													
January	25.7	25.2	24.9	24.7	27.4	30.4	31.7	31.3	29.6	27.8	26.5	25.5	27.6
February	20.8	19.4	18.4	18.6	22.9	26.8	29.1	29.2	26.7	24.6	23.3	22.4	23.5
March	31.0	30.2	29.1	29.5	33.4	36.2	38.5	38.8	37.2	35.4	34.6	33.6	34.0
April	44.7	42.9	42.2	46.7	53.1	56.7	59.2	59.9	57.9	53.0	49.2	46.9	51.0
May	54.3	53.0	53.2	58.8	63.4	66.0	68.0	68.3	65.4	60.6	57.9	56.3	60.4
Averages for 16 months	43.2	41.9	41.4	44.6	49.6	52.9	55.0	54.8	52.2	48.8	46.5	44.7	48.0
Number of degrees below high temperature	11.8	13.1	13.5	10.4	5.4	2.1	High.	.2	2.8	6.2	8.5	10.3	7.0

TABLE 8.—*Relative humidity of the outside air at the LaSalle weather-bureau station, by months, from February, 1912, to May, 1913.*

Month.	2 a. m.	4 a. m.	6 a. m.	8 a. m.	10 a. m.	Noon.	2 p. m.	4 p. m.	6 p. m.	8 p. m.	10 p. m.	Night.	Average.
1912.													
February	83.4	84.3	86.0	84.1	78.8	71.7	68.0	69.0	74.9	79.3	80.1	82.5	78.5
March	84.0	85.0	85.7	83.4	74.3	68.9	65.9	66.3	72.2	78.5	81.3	83.1	77.4
April	76.6	79.5	80.7	74.4	63.6	57.6	54.2	54.2	56.7	65.8	71.8	74.3	67.5
May	81.4	84.0	84.1	72.7	64.0	58.0	52.2	51.6	56.4	64.4	74.1	77.8	68.4
June	81.0	84.5	82.0	68.6	55.4	48.1	44.2	45.8	51.6	60.2	70.2	76.0	64.0
July	85.6	87.9	87.3	77.3	64.6	57.6	53.2	52.1	56.5	66.7	77.9	83.0	70.8
August	86.1	86.7	87.0	78.9	65.5	58.8	56.6	56.0	61.2	72.4	79.9	84.1	74.4
September	85.6	87.2	87.9	78.5	63.1	54.7	53.3	53.7	61.6	73.2	79.8	83.9	71.9
October	83.3	85.2	85.5	81.6	66.5	56.2	51.7	53.5	63.1	72.5	77.8	80.6	71.5
November	75.1	77.2	79.2	78.7	69.1	60.6	55.7	56.1	63.2	68.2	71.0	73.1	68.9
December	74.5	76.4	78.2	78.2	72.6	66.9	63.1	64.3	68.4	70.0	71.7	74.1	71.5
1913.													
January	83.7	84.4	84.6	85.3	80.7	74.7	73.0	73.9	77.5	80.8	82.5	83.4	80.4
February	80.2	80.5	81.0	80.9	73.2	67.9	65.8	67.1	73.0	77.5	80.2	80.7	75.7
March	80.8	80.1	81.8	80.9	74.4	70.2	67.4	68.7	71.5	75.9	76.2	77.6	75.5
April	75.0	77.6	78.4	71.6	58.4	50.4	45.6	44.3	48.7	56.7	65.7	70.7	61.9
May	78.9	80.6	80.0	68.5	61.1	58.0	56.5	54.0	57.5	67.8	73.8	76.5	67.8
Average	81.0	82.6	83.1	77.7	67.8	61.3	57.9	58.2	63.4	70.6	75.9	78.8	71.6

TABLE 9.—*Temperature and relative humidity of the return mine air at various mines during period of tests.*

Name of mine.	System of mining.	Number of weeks of daily readings.	Data on return air.		
			Temperature.	Relative humidity.	Water per 100,000 cubic feet of air.
			° F.	Per cent.	Gallons.
Oglesby	Long-wall	38	74.08	96.65	
LaSalle	do	52	76.62	92.02	
Assumption	do	52	74.32	93.36	
Average for long-wall mines			75.01	94.01	15.098
Murphysboro	Room-and-pillar	54	58.56	99.97	
Sherrard	do	52	68.07	95.10	
Harrisburg	do	46	60.66	97.15	
Glen Carbon	do	51	68.01	94.81	
Collinsville	do	52	63.93	94.19	
Average for room-and-pillar mines.			63.85	96.24	10.77

TABLE 10.—*Amount of moisture extracted from average long-wall and room-and-pillar mine by ventilating currents during the tests.*

Month.	Data on the outside atmosphere, LaSalle weather-bureau station.			Water extracted by ventilating currents.					
				Average long-wall mine.[a]			Average room-and-pillar mine.[b]		
	Temperature.	Relative humidity.	Water per 100,000 cubic feet of air.	Per 100,000 cubic feet of air.	Per day at 100,000 cubic feet of air per minute.	Per month at same daily rate.	Per 100,000 cubic feet of air.	Per day at 100,000 cubic feet of air per minute.	Per month at same daily rate.
1912.	° *F.*	*Per cent.*	*Gallons.*	*Gallons.*	*Gallons.*	*Gallons.*	*Gallons.*	*Gallons.*	*Gallons.*
February..	20.4	78.5	1.691	13.407	19,306	540,568	9.082	13,078	366,184
March.....	28.0	77.4	2.350	12.748	18,357	569,067	8.423	12,129	375,999
April......	50.7	67.5	4.829	10.269	14,787	443,610	5.944	8,559	256,770
May.......	64.1	68.4	7.720	7.378	10,624	329,344	3.053	4,396	136,276
June.......	68.1	64.0	8.234	6.864	9,881	296,520	2.539	3,656	109,680
July.......	74.6	70.8	11.225	3.873	5,577	172,887	[c] .452	[c] 651	[c] 20,181
August....	70.8	74.4	10.448	4.650	6,696	207,576	[c] .325	[c] 468	[c] 14,508
September.	66.0	71.9	8.639	6.459	9,301	279,030	2.134	3,073	92,190
October....	53.7	71.5	5.717	9.381	13,509	418,779	5.056	7,281	225,711
November.	41.8	68.9	3.589	11.509	16,573	497,190	7.184	10,345	310,350
December.	32.6	71.5	2.646	12.452	17,931	555,861	8.127	11,703	362,793
1913.									
January...	27.6	80.4	2.398	12.700	18,288	566,928	8.375	12,060	373,860
February..	23.5	75.7	1.881	13.217	19,032	532,896	8.892	12,804	358,512
March.....	34.0	75.5	2.946	12.152	17,499	542,469	7.827	11,271	349,401
April......	51.0	61.9	4.474	10.624	15,299	458,970	6.299	9,071	272,130
May.......	60.4	67.8	6.776	8.322	11,984	371,504	3.997	5,756	178,436

[a] 4,877,380 gallons per annum. [b] 2,604,140 gallons per annum. [c] Deposited in the mine.

EFFECT OF HUMIDITY CHANGES ON RIB-DUST SAMPLES.

In a preliminary report of the cooperative investigations[a] the following statement was made on the collection of rib-dust samples:

> Rib-dust samples have been taken at three points in each mine visited—namely, on the main haulage road, on the subsidiary haulage roads, and inside the last crosscut near the face of an entry or a room. These samples were taken according to the following uniform method:
>
> A chalk line was drawn from the mine roof to the floor. On one side of this line an area of rib was cleaned with a 1-inch varnish brush from the top to the bottom of the coal seam. This area was made of sufficient width for the dust to fill a 40-gram glass bottle. The material was collected on a sheet of celluloid or a piece of clean paper and transferred to the bottle, which was then sealed with a rubber cork. The height and width of the area of rib brushed was measured. Note was also made of the temperature, the relative humidity, and the distance that the air had traveled from the intake to the point of sampling.

In Table 11 following, figures are given showing the effect of the humidity and temperature changes on the rib dusts collected in 100 mines during the year 1912.

HUMIDITY CHANGES BETWEEN INTAKE AND UNDERGROUND STATIONS.

Records of humidity changes between intake and underground stations are presented in Table 11 following:

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[a] Preliminary report on organization and method of investigations: Illinois Coal-Mining Investigations, Cooperative Agreement. 1913, p. 44.

TABLE 11.—*Humidity changes between the intake and underground stations.*

No. of mine.	Place of sampling.[a]	Depth of shaft.	Air-drying loss of face sample.	Air-drying loss of rib-dust sample.	Intake.		Mine station.				Changes between intake and mine station.				
					Temperature.	Relative humidity.	Temperature.	Relative humidity.	Water per 100,000 cubic feet of air (gals.).	Vapor pressure.	Distance.	In vapor pressure.	Amount of water per 100,000 cubic feet of air.	Heat given by (+) or to (−) the mine ribs per 100,000 cubic feet of air.	Air-drying loss of the rib dust as compared with that of the face sample.
		Feet.	*Per cent.*	*Per cent.*	*° F.*		*° F.*		*Gallons.*		*Feet.*	*Inches.*	*Gallons.*	*B. t. u.*	
5	a	135	9.92	18.85	44	93	60	89	8.760	0.460	2,000	+0.193	+3.516	+57,888	+ 8.93
6	b	100	12.66		84	68	66	95	11.415	.606	1,800	− .185	−3.022	−52,225	
	c						79	79	14.380	.781	300	− .010	− .057	− 9,105	
7	a	200	9.56	17.14	52	63	65	85	9.883	.524	2,200	+ .280	+5.166	+62,657	+ 7.58
	b			5.73			65	85	8.883	.524	2,500	+ .280	+5.166	+62,657	− 3.83
9	c		12.61	3.02	45	58	56	57	6.616	.345	1,400	+ .172	+3.224	+46,452	− 9.59
14	a	135	4.52	17.64	66	45	64	90	10.124	.536	1,200	+ .249	+4.717	+30,292	+13.12
15	a	160	4.37	15.89	64	85	64	95	10.686	.565	6,000	+ .059	+1.125	+ 8,150	+11.52
	c			12.55			65	90	10.463	.554	560	+ .048	+ .902	+ 8,308	+ 8.18
21	b	1,004	6.87	b 42.96	86	51	74	70	10.889	.587	750	− .046	− .619	−25,301	b +36.09
	c			b 4.21			77	83	14.192	.769		+ .136	+2.684	+ 886	b − 2.66
25	a	185	10.75	21.64	76	83	62	89	9.368	.494	4,200	− .250	−4.384	−55,834	+10.89
	c						67	95	11.795	.628	9,000	− .116	−1.957	−29,427	
26	a	196	10.83	2.96	88	61	64	90	10.124	.536	10,000	− .270	−4.494	−72,600	− 7.87
	b			12.08			65	90	10.463	.554	10,000	− .252	−4.155	−68,393	+ 1.25
	c			28.43			69	83	10.996	.587	1,700	− .219	−3.622	−57,043	+17.60
27	a	170	10.17	19.65	84½	47	64½	87	9.950	.527	1,500	− .029	− .190	−37,224	+ 9.48
	b			23.01			64	90	10.124	.536	1,900	− .020	− .016	−37,006	+12.84
	c			11.06			65½	95	11.230	.596	3,500	+ .040	+1.090	−26,949	+ .89
29	b	185	11.50	8.68	78	64	70	77	10.538	.564	250	− .048	− .754	−19,261	− 2.82
	c			9.87			69	77	10.202	.544	1,000	− .068	−1.090	−23,337	− 1.63
31	b	60	11.86	8.72	70	64	67	85	10.553	.562	100	+ .094	+1.693	+ 6,449	− 3.14
32	a	60	10.79	2.12	93	51	63	84	9.139	.483	3,800	− .305	−5.039	−86,496	− 8.67
	b			12.35			67	90	11.174	.595	4,000	− .193	−3.004	−65,489	+ 1.56
	c			9.03			76	87	14.415	.780	8,000	− .008	+ .237	−28,487	− 1.76
33	a	285	10.50	21.72	77	67	66	80	9.612	.510	4,000	− .110	−1.844	−32,271	+11.22
	b			6.69			64	90	10.124	.536	4,000	− .084	−1.332	−32,453	− 3.81
36	b	238	11.83	22.45	87	74	66	90	10.814	.574	12,000	− .374	−6.394	−79,297	+10.62
37	b	235	6.27	8.50	70	77	67	85	10.553	.562	3,800	− .002	+ .015	− 5,266	+ 2.23
	c			10.57			68	90	11.542	.616	5,000	+ .052	+1.004	+ 3,415	+ 4.30
38	a	245	7.30	19.49	79½	68	66	92	11.054	.587	5,500	− .097	−1.518	−34,425	+12.19
	b			11.11			67	95	11.795	.628	3,400	− .056	− .777	−27,583	+ 3.81
39	a	204	6.98	14.79	74	82	64	92	10.349	.547	3,000	− .140	−2.407	−34,731	+ 7.81
	b			11.71			64½	92	10.524	.558	3,500	− .129	−2.232	−32,564	+ 4.78
	c			3.95			68	90	11.542	.616	8,000	− .071	−1.214	−19,018	− 3.03

40	a	270	10.65	22.49	70	59	66	92	11.054	.587	5,400	+ .008	+ .315	− 21,184	+11.84
	b			13.38			67½	85	10.727	.571	5,600	− .013	− .012	− 20,642	+ 2.78
	c			7.49			70	90	12.317	.659	1,500	+ .075	+1.578	− 5,559	− 3.16
41	c	365	9.39	6.24	79	64	69	90	11.923	.636	800	+ .003	+ .274	− 16.005	− 3.15
42	c	570	7.14	8.31	80	64	70	100	13.686	.732	1,200	+ .078	+1.669	− 6,785	+ 1.17
44	a	270	4.01	15.20	88	61	66	100	12.015	.638	3,000	− .168	−2.603	− 56,321	+11.19
	b			16.35			66	100	12.015	.638	3,000	− .168	−2.603	− 56,321	+12.34
45	a	320	4.31	1.50	85	80	65	95	11.044	.585	2,000	− .376	−6.466	− 78,615	− 2.81
	b			8.81			65	95	11.044	.585	2.000	− .376	−6.466	− 78,615	+ 4.50
	c			11.20			67	90	11.174	.595	4,500	− .366	−6.336	− 73,670	+ 6.89
46	a	450	4.34	11.02	79	57	68	95	12.183	.650		+ .067	+1.808	− 7,565	+ 9.68
	b			2.39			68	95	12.183	.650	4,000	+ .067	+1.808	− 7,565	+ 1.95
	c			12.81			67	95	11.795	.628	10,000	+ .065	+1.420	− 11,942	+ 8.47
49	a	337	3.05	10.04	87	51	70	90	12.317	.659		+ .006	+ .457	− 27,337	+ 6.99
	b			7.48			70	90	12.317	.659	1,500	+ .006	+ .457	− 27,337	+ 4.43
	c			8.40			70	90	12.317	.659	10,000	+ .006	+ .457	− 27,337	+ 5.35
50	a	726	3.72	6.42	75	87	69	95	12.585	.672	3,000	− .081	−1.382	− 20,062	+ 2.70
	b			12.22			60	90	11.923	.636	2,000	− .117	−2.044	− 24,548	+ 8.50
	c			5.72			70	95	13.001	.695	2,500	− .058	− .966	− 15.420	+ 2.00
51	a	494	5.89	16.00	51	100	63	84	9.139	.483	2,000	+ .110	+1.910	+ 36,769	+10.11
	b			13.59			61	81	8.245	.434	1,700	+ .061	+1.016	+ 26,295	+ 7.70
	c			11.78			62	89	9.368	.494	3,500	+ .121	+2.139	+ 36,740	+ 5.89
52	a	640	4.09	8.17	34	81	70	81	11.086	.593	8,000	+ .435	+7.925	+131,740	+ 4.08
	b			3.69			60½	78	7.808	.411	4,000	+ .253	+4.647	+ 88,833	− .40
	c			4.76			60	78	7.677	.403	16,000	+ .245	+4.516	+ 86,863	+ .67
53	a	417	7.20	14.19	72	86	64	90	10.124	.536	9.000	− .137	−2.426	− 31,406	+ 6.99
	b			6.27			63	89	9.683	.512	10,000	− .161	−2.867	− 36,417	− .93
	c			5.16			63	90	9.792	.518	8,000	− .155	−2.758	− 35,650	− 2.04
54	a	380	7.48	7.93	53½	88	65	90	10.463	.554	4,000	+ .194	+3.522	+ 47,618	+ .45
	a			17.72			65½	92	10.875	.577	3,000	+ .217	+3.934	+ 51,577	+10.24
	c			11.18			61	94	9.568	.504	3,900	+ .144	+2.627	+ 33,846	+ 3.70
55	a	180	6.08	19.04	32	100	64½	87	9.950	.527	6,600	+ .347	+6.332	+114,067	+12.96
	b			13.13			64	92	10.349	.547	9,500	+ .367	+6.731	+116,404	+ 7.05
	c			12.32			65	90	10.463	.554	3,960	+ .374	+6.845	+119,055	+ 6.24
56	a	580	3.14	12.10	75	41	66	90	10.814	.574	1,600	+ .219	+4.231	+ 12,905	+ 8.96
	b			7.16			64	85	9.561	.506	1,400	+ .151	+2.978	+ 814	+ 4.02
	c			14.19			64	90	10.124	.536	6,000	+ .181	+3.541	+ 4,711	+11.05
57	a	318	3.44	14.19	73	78	64	90	10.124	.536	2,000	− .096	−1.628	− 27.565	+10.75
	b			9.76			65	90	10.463	.554	2,000	− .078	−1.289	− 23.336	+ 6.32
	c						64	90	10.124	.536		− .096	−1.628	− 27.565	
69	a	290	8.10	23.48	82	72	66	90	10.814	.574	9,000	− .212	−3.656	− 52,427	+15.38
	b			27.19			67	85	10.553	.562	9,000	− .224	−3.826	− 52,216	+19.09
	c			9.24			70	88	12.044	.644	2,000	− .142	−2.335	− 36,636	+ 1.14
75	a	310	9.00	17.43	84½	70	66½	95	11.605	.617	4,000	− .210	−3.587	− 55,817	+ 8.43
	b			8.11			69	90	11.923	.636	4,000	− .191	−3.269	− 48,879	− .89
	c			12.31			69½	88	11.851	.633	1,200	− .194	−3.341	− 48,382	+ 3.31
78	a	140	7.83	21.91	80	76	66	100	12.015	.638	5,000	− .139	−2.255	− 42,230	+14.08
	b			18.53			66	100	12.015	.638	5,000	− .139	−2.255	− 42,230	+10.70
	c			9.24			71	95	13.425	.719		− .058	− .845	− 21,536	+ 1.41

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[a] The place of sampling is denoted by letters as follows: a—a dust taken inside the last crosscut of an entry or room near the "working face;" b—a dust taken along the subsidiary haulage roads; c—a dust collected on the main haulage roads.

[b] Road-dust samples.

TABLE 11.—*Humidity changes between the intake and underground stations*—Continued.

No. of mine.	Place of sampling.[1]	Depth of shaft.	Air-drying loss of face sample.	Air-drying loss of rib-dust sample.	Intake.		Mine station.				Changes between intake and mine station.					
					Temperature.	Relative humidity.	Temperature.	Relative humidity.	Water per 100,000 cubic feet of air (gals.).	Vapor pressure.	Distance.	In vapor pressure.	Amount of water per 100,000 cubic feet of air.	Heat given by (+) or to (−) the mine ribs per 100,000 cubic feet of air.	Air-drying loss of the rib dust as compared with that of the face sample.
		Feet.	*Per cent.*	*Per cent.*	° *F.*		° *F.*		*Gallons.*		*Feet.*	*Inches.*	*Gallons.*	*B. t. u.*	
79	a........	127	7.41	1[illegible].40	82	65	63	100	10.880	.575	6,000	− .144	−2.101	− 48,451	+ 8.99
	b........	....	....	16.60	....	....	63	100	10.880	.575	6,000	− .144	−2.101	− 48,451	+ 9.19
	c........	....	....	12.10	....	....	64	90	10.124	.536	7,000	− .183	−2.857	− 51,590	+ 4.69
80	b........	145	5.20	6.52	82	80	62½	94	10.061	.531	12,500	− .342	−5.916	− 84,563	+ 1.32
	c........	....	....	8.66	....	....	65½	87	10.284	.545	4,000	− .328	−5.693	− 67,613	+ 3.46
81	a........	200	4.80	25.49	84	66	62	94	9.894	.522	3,500	− .246	−4.118	− 67,437	+20.69
	b........	....	....	15.20	....	....	63½	97	10.626	.567	3,000	− .201	−3.386	− 59,537	+10.40
	c........	....	....	5.05	....	....	65	90	10.463	.554	10,000	− .214	−3.549	− 57,673	+ .25
82	a........	192	7.27	17.48	78	79	64	100	11.249	.595	9,000	− .161	−2.689	− 43,577	+10.21
	b........	....	....	11.63	....	....	64	95	10.686	.565	7,000	− .191	−3.252	− 47,424	+ 4.36
	c........	....	....	9.94	....	....	65	95	11.044	.585	15,000	− .171	−2.894	− 42,765	+ 2.67
83	a........	140	6.35	13.23	93	62	63	100	10.880	.575	6,000	− .384	−6.356	− 94,957	+ 6.88
	b........	....	....	9.68	....	....	67	95	11.795	.628	7,000	− .331	−5.441	− 80,318	− 3.33
	c........	....	....	12.23	....	....	67	90	11.174	.595	....	− .364	−6.062	− 84,173	+ 2.95
84	c........	320	5.85	....	93	57	69	100	13.248	.707	4,500	− .174	−2.598	− 59,027	....
88	a........	85	4.42	10.41	94	54	67	95	11.795	.628	....	− .233	−3.665	− 72,343	+ 5.97
	b........	....	....	....	....	....	63	95	10.336	.546	....	− .315	−5.124	− 88,697	....
	c........	....	....	8.01	....	....	71	90	12.719	.681	30,000	− .180	−2.741	− 57,738	+ 3.59
89	a........	85	5.18	18.26	82½	62	62	94	9.894	.522	4,600	− .166	−2.681	− 55,190	+13.06
	b........	....	....	16.70	....	....	63	100	10.880	.575	4,600	− .113	−1.695	− 46,589	+11.52
	c........	....	....	16.58	....	....	66	90	10.814	.574	9,000	− .114	−1.761	− 41,285	+11.40
90	a........	160	5.44	13.75	84	70	63	90	9.792	.518	700	− .296	−5.070	− 71,792	+ 8.31
	b........	....	....	12.03	....	....	62	94	9.894	.522	1,200	− .292	−4.968	− 73,146	+ 6.59
91	a........	217	10.77	27.83	85	100	60	89	8.761	.460	1,900	+ .257	+4.710	+ 86,314	+17.06
	d........	....	....	26.55	....	....	64½	92	10.522	.557	1,900	+ .354	+6.471	+108,500	+15.78
	c........	....	....	23.80	....	....	50½	93	6.607	.341	5,000	+ .138	+2.556	+ 51,236	+13.03
92	....	240	....	....	26½	64	56	94	7.389	.421	Return.	+ .333	+5.591	+105,697	....
							39	84	3.949	.199	500	+ .111	+2.151	+ 44,264	....
93	....	186	13.27	....	20½	85	24	87	2.209	.108	Intake.	+ .020	+ .391	+ 11,151	....
							58¼	98½	9.139	.479	Return.	+ .391	+7.321	+138,228	....
97	a........	223	11.20	23.17	42	85	61¼	92	9.525	.502	900	+ .276	+5.066	+ 76,971	+11.97
	d........	....	....	45.39	....	....	63¼	88½	9.711	.513	1,500	+ .287	+5.252	+ 81,467	+34.19
	c........	....	....	48.02	....	....	53	91	7.054	.366	1,800	+ .140	+2.595	+ 41,971	+36.82
100	b........	400	7.84	2.00	78	60	81	83	16.073	.876	2,000	+ .302	+5.487	+ 40,031	− 4.94
	c........	....	....	4.10	....	....	79	87	15.836	.860	5,400	+ .286	+5.250	+ 35,322	− 3.74

GENERALIZATIONS.

The records presented in Table 11 warrant the following generalizations:

1. The evaporation of water proceeds rapidly when conditions are favorable. In Mine No. 97, in 900 feet the amount of water evaporated per 100,000 cubic feet of air was 5.066 gallons, or 5.63 gallons per 1,000 feet of entry. In Mine No. 31, in 100 feet the amount of water evaporated was 1.693 gallons per 100,000 cubic feet of air, or 16.93 gallons per 1,000 feet of entry. In each of these mines the movement of the air was sluggish and the change in vapor pressure was very large for the difference in distance. In general, the lower the temperature of the intake air as compared with the temperature of the return air, the greater will be the quantity of water evaporated. In Mine No. 52, where the intake temperature was 36° F. below that of the return, 7.925 gallons of water was evaporated for every 100,000 cubic feet of air. In Mine No. 93, 7.321 gallons was evaporated for a temperature difference of 38° F. In Mine No. 55, for 33° F. difference in temperature, 6.845 gallons of water was evaporated.

2. The deposition of water is rapid under favorable conditions. In Mine No. 90, in 700 feet, the amount of water deposited per 100,000 cubic feet of air was 5.07 gallons, or 7.24 gallons per 1,000 feet of entry. The higher the temperature and relative humidity of the intake air as compared with the return, the larger will be the quantity of water deposited. In Mine No. 83, 6.356 gallons of water was deposited for a temperature difference of 30° F.; in Mine No. 36, 6.394 gallons in 21° F.; and in Mine No. 45, 6.466 gallons in 20° F.

3. The temperature of the intake air is quickly changed to the temperature of the mine strata. In Mine No. 97, 900 feet from the shaft, which was 223 feet deep, the temperature was raised 19½° F.; and in Mine No. 90, 700 feet from the 160-foot air shaft, the temperature was lowered 21° F. An important agent in bringing about a rapid drop in temperature is water falling in the intake shaft. At Mine No. 90, just mentioned, considerable water occurs at the shaft.

4. The dust investigations were made at irregular intervals during 1912, but the greater part of the readings (76 per cent) was taken in the summer months when the temperature of the intake air was higher than that of the return air. The influence of the relative humidity of the intake air is indicated by the fact that 42 per cent of the records show that evaporation was taking place.

5. In 92 per cent of the samples taken inside the last crosscut of an entry or room near the "working face" (location *a*) the rib dust lost more water on air drying than did the face sample of coal. In 76 per cent of the dusts taken along the subsidiary haulage roads (location *b*) more water was lost on air drying than was lost by the face sample of coal. In 78 per cent of the samples collected on the

main haulage roads (location *c*) the air-drying loss was greater than that of the face sample of coal. It is apparent that, in this series of tests, the rib dusts, especially those collected near the working faces, showed a tendency to contain more mechanically held water than did the face samples of coal. The fineness of the dusts, and the fact that most of the samples were gathered in the summer when condensation was taking place, are suggested as reasons for the high moisture content of these dusts.

6. In the majority of these tests (59 per cent) the rib dusts showed a moisture content that varied according to whether the ventilation currents had caused evaporation or deposition of moisture on entering the mine. This relation was noticed in Mines Nos. 27, 38, 41, 44, 50, 69, 78, 79, 80, 81, 82, 83, 88, 89, 90, and 100. The seeming exceptions may have been due to one or all of the causes mentioned below:

(*a*) The roadways were sprinkled shortly before some of the samples were taken. Samples Nos. 51, *b* and *c*, 52, *c*, and 56 *a*, *b*, and *c*, are examples. In the mines from which those samples were taken sprinkling is done every night—by cars at Mine No. 51 and by hose at Mines Nos. 52 and 56.

(*b*) The time of year during which the samples were collected varied. Sample No. 37, *b*, and sample No. 37, *c*, were obtained on August 10, 1912, when the temperature was temporarily below the normal for that time of year.

(*c*) The temperature of the ventilation air in some mines was lowered below the mine temperature owing to the evaporation of water, and was then reheated through the agency of the mine ribs, further evaporation being thus permitted. In Mine No. 39, at location *b*, 0.6 mile from the intake, the temperature was 64° F.; and at location *c*, 1.6 miles from the intake, the temperature was 68° F. and the dust contained 7.76 per cent less moisture than did the dust at location *b*.

(*d*) Mine water may have been present in unusually large quantities. In Mines Nos. 91 and 97 large quantities of water are found at the shafts and wherever breaks occur through to the surface. Consequently these mines even in winter are damp and the rib dusts show large amounts of mechanically held moisture.

7. The extent to which the ribs, roof, and floor of mine workings radiate and absorb heat depends on whether the temperature of the air is less or greater than that of the mine strata. In Mine No. 93 the average quantity of air delivered by the fan is 125,000 cubic feet per minute; and on February 27, 1912, when the humidity reading was made, 138,228 British thermal units was given up by the walls of the workings to each 100,000 cubic feet of the ventilation. This quantity of heat would be equivalent to that obtained by burning about 11 tons of coal per 24 hours.

The absorption of heat by the walls of the mine workings is shown to be very large in Mine No. 83, where the heat absorbed per 100,000 cubic feet of air amounted to as much as 94,957 heat units.

The main factor governing the extent of radiation or absorption of heat by the surfaces of the rooms and entries is shown in Table 11 to be the difference between the temperature of the intake air and that of the mine strata.

METHODS OF MINE HUMIDIFICATION IN ILLINOIS.

In 88 mines, carefully selected to represent the general conditions in Illinois, a study has been made of the methods adopted to humidify the coal dust, as well as of the amount, character, and occurrence of the mine water.

The quantity of water pumped from the average mine in Illinois is not excessive, the greatest quantity reported being 1,000,000 gallons per 24 hours. A few hours' pumping each day is sufficient to keep most mines free from water. In two mines visited the quantity of water was so small that no pump was installed and any excess water was loaded into tank cars and hoisted to the surface.

The character of the mine water as reported by the different mine managers was acid in 42 per cent of the mines, and alkaline in 15 per cent, and in 43 per cent was neutral, being suitable for boiler use and in many being fit for drinking.

In 55 per cent of the mines the water occurred principally at the shafts, in 20 per cent at the mine faces, and in 25 per cent at other sections, such as at points of heavy falls, breaks to the surface, and dip workings.

There was no means of artificial humidification and no attempt to control the condition of the dust in 25 per cent of the mines. In most of the mines of this group the underground workings were either naturally damp because of nearness to the surface, or the dust was so mixed with shale from roof or floor that special humidification was unnecessary. In some of the mines, however, the dust was dry, very abundant, and a source of danger.

USE OF WATER CARS.

Of the mines into which water was introduced artificially 91 per cent used water cars. The frequency of sprinkling by this means was as follows:

Frequency of sprinkling with water cars.

Interval between sprinkling.	Number of mines.	Interval between sprinkling.	Number of mines.
	Per cent.		*Per cent.*
6 months	1.7	1 week	35.0
3 months	1.7	3 days	3.3
1 month	5.0	2 days	6.7
Irregular	23.3	1 day	10.0
2 weeks	13.3		

The type of water cars varied. The usual form was that of an old mine car calked and covered and containing two or three holes through which the water flowed to the roadways. The most advanced type, found in a few mines, was that of a steel tank mounted on wheels and equipped with sprinkling apparatus, being similar to a city street sprinkler. This type contained apparatus that compressed the air to give sufficient force to the spray to wash the ribs and the roof as well as sprinkle the floor.

The capacity of the cars varied from 250 to 700 gallons. The average cost of sprinkling was $1 per 1,000 gallons. The objections to the use of water cars for sprinkling the coal dust are as follows: In winter the water supplied does not equal the amount removed by the ventilating currents; the sprinkling is performed only on haulage roads, and not near the faces where windy and blown-out shots originate; in general only a part of one surface (the floor) of the entry is moistened, whereas the other three surfaces (the two ribs and the roof) are untouched; sprinkling dry floor dust in this manner is in general ineffective, as usually the dust is not thoroughly dampened but some of it floats on the water; "loads" or "empties" often have to be switched to permit the passing of the water car; the cost is high.

The water car is valuable in many mines in supplementing some other form of humidification, such as exhaust steam, water from fixed sprays or hose, or shale dust.

USE OF WATER HOSE.

The use of water hose has been effective in eliminating the dangers from blown-out or windy shots in 6 per cent of the mines visited.

At one mine two men with hose sprinkle every night, washing the ribs, roof, and posts on the cross entries. In this way each entry is washed every week.

At another mine there is sufficient hose to reach each working face, and the ribs, roof, face, and props of each room are washed down at least once each month.

The cost of sprinkling by hose, exclusive of the piping installation, was 60 cents per 1,000 gallons; the total cost of sprinkling per ton of coal was 0.5 cent in mines where the entries were washed each week.

HUMIDIFICATION BY STEAM.

Steam was used for humidification in 16 per cent of the mines. The common practice was to admit the exhaust from the fan engine into the intake shaft.

In one mine (No. 29) steam jets placed at various points in the air shaft effectively preheated the intake air during cold weather.

In three mines (Nos. 25, 59, and 100) the downcast air passed over radiators, one radiator containing 695 feet of 1-inch pipe, another 1,000 feet of 1½-inch pipe, and the third 3,000 feet of 1-inch pipe. In the two radiators first mentioned the steam was carried at about 80 pounds pressure and was exhausted outside; the radiator last mentioned was connected with the exhaust of the fan engine and was guaranteed to preheat 50,000 cubic feet of air per minute from 0° to 35°.

In many mines advantage was taken of the heat radiated from steam pipes by conducting the pipes through the downcast shafts, thus causing some preheating of the intake air.

The cost of using exhaust steam was practically nothing, and the cost of live steam (with coal at the mine valued at $1 per ton) was 44 cents per 1,000 gallons of water evaporated.

The use of steam could be made more efficient if the problem at each mine were studied in the light of the information given on pages 52 to 61 of this report.

ADMIXTURE OF FOREIGN MATERIAL WITH ROAD DUST.

The admixture of shale, ashes, or other foreign material with the road dust is a method of dust control that proved effective in a number of mines. In the Longwall field the shale falls naturally and in sufficient quantities to check the propagation of a dust explosion.

USE OF ASHES AS ROAD BALLAST.

In one room-and-pillar mine (No. 62) ashes obtained from the boiler plant on the surface were applied to the haulage roads in the following manner: When an entry had been driven far enough to warrant laying of the permanent road, the ballasting was done entirely with ashes, the roadway having been cleaned of all dust and dirt. Then, as coal dust became mixed with the ashes, additional ashes were applied, 76 cubic feet of ashes covering 40 linear feet of track. In this manner 150 cubic feet of ashes was used every day. This method was supplemented by sprinkling the roads every night by water cars, about 2,000 gallons of water being applied every 24 hours. At this mine the rib dust seemed damp. Two years ago a bad gas explosion started and killed eight men; but the force died down because of the dampness of the dust and the admixture of ashes. This method of diluting the coal dust is effective.

USE OF STANDING BODIES OF WATER.

Standing bodies of water on the floor of intake air courses supply a certain amount of moisture to the air. In one mine (No. 56) a permanent body of water, with an average depth of 18 inches, covered

150 feet of the air course. This form of humidification is valuable merely as an auxiliary to some more approved method and then only provided it does not obstruct the airway.

USE OF CALCIUM CHLORIDE.

The use of calcium chloride was noted at one mine in Saline County. One and one-fourth pounds per square yard of floor, with the coal dust 1 inch thick, was reported to have kept the dust moist for six fall and winter months. The cost of the calcium chloride was $13 per ton f. o. b. the mine. The results were claimed to have been satisfactory.

At mines where the roads were cleaned frequently it was noted that less sprinkling was required.

EXHAUST FANS COMPARED WITH PRESSURE FANS.

Most of the mines of Illinois do not generate much methane. In the comparatively few mines where this explosive gas is found, the amount generated in the ordinary mining operations is small. There is not a "safety-lamp" mine in the State. For convenience therefore most of the mines are ventilated by force fans.

Where a pressure fan is in use and the hoisting shaft is the upcast the following conditions prevail:

EFFECT OF PRESSURE FAN ON BOTTOM OF MAIN SHAFT.

1. At the bottom of the main shaft there is throughout the year an approximately uniform temperature and relative humidity. This uniformity is appreciated by the cagers because they are not subjected to the heat of summer or the cold of winter—effects of temperature that are intensified by the high velocity of the air current. The roof also is not weakened by either the seasonal or the diurnal changes in temperature and humidity, and there is therefore less likelihood of falls at places where a number of men are continuously at work, all men departing from and entering the hoisting cage together, usually at least once each shift.

EFFECT OF PRESSURE FAN ON MAIN HAULAGE ROADS.

2. The main haulage roads are on the return of the air. The temperature and relative humidity are much more uniform than in the intake airways; falls are therefore less likely to occur along the haulage roads, accidents are not so frequent, and delays in handling the coal are reduced. Furthermore the coal dust that leaks through or is spilled from the cars and ground by the wheels on the road and by the tramping of men and mules is not robbed so completely of moisture during the winter as would be the case if these

roads were on the intake of the air. Also the air that leaks through brattices and stoppings passes from the air courses into the haulage roads, and if moisture is deposited it is on the return side of the stoppings, whether the temperature of the leaking air is higher or lower than that of the return, owing to the reduction in temperature of the air of higher temperature by that of the lower temperature.

USE OF PRESSURE FAN MAKES POSSIBLE EFFICIENT HUMIDIFICATION.

3. Opportunities are offered for the cheap and efficient preheating and premoistening of the intake air with the aid of steam and water sprays, which will cause the temperature and relative humidity of the mine atmosphere to be uniform throughout the year.

SUMMARY AND SUGGESTIONS.

DETAILS GOVERNING CHOICE OF HUMIDIFICATION METHODS.

Before a suitable method can be adopted for humidifying the air at any mine, a thorough investigation embracing the features mentioned below should be made.

CHARACTER OF FACE SAMPLE.

A face sample should be taken and analyzed. If the sum of the ash and the moisture contents is more than 50 per cent, the possibility of an extended dust explosion is materially reduced but not obviated.

The explosibility indicates the relative danger of a given coal dust and gives an index of the need of changing the character of the mine air. The relative explosibility of 100 Illinois coals is being studied by the representatives of the bureau engaged in the cooperative work at Urbana, and a report is being prepared. A brief statement of the method of this investigation is given in the preliminary report.[a]

CHARACTER OF RIB AND ROAD DUSTS.

The rib dust is the fine dust that lodges lightly on the roof and ribs and is easily thrown into suspension by runaway trips, blown-out shots, and explosions. Road dust is the dust that accumulates along roadways and is caused by leaky cars, weak roofs, and soft bottoms. A proximate analysis and an explosibility test should be made for these dusts as outlined for the face sample.

Another important consideration is the quantity and fineness of these dusts. Under the cooperative mining investigation three samples of rib dust and two samples of road dust were collected from each of 100 mines in Illinois; and the investigation has shown that in every case the quantity of rib dust finer than 20-mesh is sufficiently large to propagate an explosion.

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[a] Preliminary report on organization and methods: Illinois Coal-Mining Investigations, Cooperative Agreement, 1913, pp. 43-45, 67-71.

The Danville field is an example of a district where the mines contain large quantities of fine rib dust and yet where there has never been an explosion. The weak shale roof spalls off, falls on the roadways, and mixes with the coal dust; the comparatively shallow mines contain a large amount of water; and the result is that, although the face sample would indicate a coal with an inflammable dust, there have been no explosions because the aggregate quantity of the moisture and ash in the rib and road dusts is very high.

PRESENCE AND SITUATION OF MINE WATER.

The presence and situation of the mine water and its effect in increasing the humidity of the air should be carefully ascertained by means of psychrometric readings. The places where the water enters the mine and the quantity per 24 hours are important considerations. If the water occurs at the face and is loaded out with the coal, it will assist in sprinkling the roadways as the cars go to the shaft; it will also prevent a large amount of dust from being thrown into suspension during the loading of the coal, and will thus lessen the amount of float dust and of rib and road dusts.

If the water runs in ditches down slope haulageways, it will assist in preventing the formation of dry dust along these roads. If the water occurs along the intake air courses, it will assist in supplying the moisture needed by the intake air during the greater part of the year and will lessen the amount of water extracted from the dust at the working faces. If the water occurs in the shafts, it will be of benefit in the downcast air shaft only.

CLIMATIC CONDITIONS.

The temperature and absolute humidity of the intake air vary with the climatic conditions. In Illinois the climatic conditions, although comparatively uniform throughout the State, differ greatly from those in the dry climate of Colorado or Oklahoma. In general, the less the average amount of moisture in the outside air and the colder the average temperature, the greater will be the need for humidification.

SEA-LEVEL ELEVATION OF THE MINE WORKINGS.

In any particular coal field, the deeper the mine the higher will be the temperature of the return air; but in Illinois the deepest mine at present is only 1,004 feet, and the deepest coals in the south central part of the Illinois basin are not over 1,500 feet. At these depths there is no serious difficulty in adequately humidifying coal mines.

QUANTITY OF AIR IN CIRCULATION.

As the quantity of water in a unit volume of return air at any mine is nearly constant during the entire year, the amount of moisture extracted from the dust will vary directly with the quantity of air in circulation. Therefore the quantities of heat and moisture required for humidifying the intake air depend on the volume of the ventilating current.

In many mines, 60 to 80 per cent of the air delivered by the fan is lost through leaky stoppings and doors; but this air absorbs heat and moisture before leaving the mine. If, therefore, by the construction of tight masonry or concrete stoppings along main haulageways and by the substitution of overcasts for doors it is possible to reduce the amount of air required to be delivered by the fan and at the same time to maintain sufficient air at the working faces, less water will be extracted from the mine by the return air current and less heat and moisture will be required to humidify the intake air.

UPCAST OR DOWNCAST AT HOISTING SHAFT.

If a force fan is used, it is usually possible and convenient to apply any efficient method of humidification, such as the use of exhaust steam or water sprays directly to the intake air without interfering with the haulage, travel of men, etc. Furthermore, the humidification of the air may be started in the air shaft as soon as the air leaves the fan.

With an exhaust fan the problem becomes more difficult. Unless the top of the hoisting shaft is inclosed and the air preheated before entering the shaft, humidification must be delayed until the air has begun to travel through the mine. Sprays must then be placed on main haulage and traveling ways and will cause inconvenience. But gaseous mines require that the haulage be on the intake roads, and in those mines the fan must exhaust.

When a mine is opened in a district where the coal dust is known to be explosible, special arrangements should be made for premoistening and preheating or precooling the intake air. A separate air course would be advisable in order that the humidification of the intake air may be perfected without causing inconvenience; for example, if the intake were the traveling way for the men, exhaust steam would be objectionable because of the fog that would be caused.

PRESENCE OF METHANE OR FIRE DAMP.

Methane intensifies the dangers of coal-dust explosions. The ignition of a small body of the gas may start a dust explosion; and in the case of an overcharged shot, methane may render explosive the gases from the shot. If methane is present, special care should be taken to make the coal dust inert.

CHARACTER OF THE ROOF.

The more tender the roof, the greater the need for maintaining uniform temperature in the intake. Both the seasonal and diurnal changes in temperature cause alternate expansion and contraction of the roof, which in turn causes heavy falls.

As a rule, also, the weaker the roof, the more uniform should be the relative humidity; for alternate wetting and drying of a roof causes it to disintegrate rapidly.

TENDENCY FOR ICE TO FORM IN ESCAPE WAYS.

During the months of January and February, 1912, many mines were idle several days because of inability to prevent ice forming in the intake shafts. Reversal of the ventilation may not suffice to clear a frozen shaft of ice, because ice forms rapidly as water seeps into the shaft, whereas the ice melts slowly because only the surface of the frozen mass is exposed to the warm air. The practice of reversing the fan, therefore, may cause both shafts to become blocked with ice in severe weather. Furthermore, reversal of the air in a mine producing any fire damp is dangerous.

As a blocked escape way may cause many fatalities, it is imperative that the formation of ice in a shaft should be prevented. As a rule it is cheaper, and it is always safer, to preheat the air than to chop the ice.

METHOD OF HUMIDIFYING MINE AIR WITH STEAM.

After a series of observations, such as outlined, has shown that in a certain mining district humidification is required to make the mines safe from dust explosions, information is often desired on the use of steam for humidifying the intake air.

In humidifying by steam the exhaust from the fan engine is usually turned into the downcast shaft. This practice is inefficient because it will correct conditions at any mine for only one particular temperature of the outside air. For example, if the ventilating current comprises 50,000 cubic feet of air per minute, with a return temperature of 60° F., and if the horsepower of the fan engine is 33.4, the quantity of heat in the exhaust from the fan will correct the intake temperature only when the outside air is 43°. If that air becomes either colder or warmer, too little or too much heat will be admitted by the steam. In a mine at Gillespie, on a cold night in January, 1912, a small amount of exhaust steam that had been turned into the intake was transformed into snow, and completely blocked the foot of the air shaft. In a mine at West Frankfort, when too much steam was suddenly admitted, expansion of the roof due to the high temperature caused heavy falls of rock. In some mines the amount of steam admitted is gaged by the position of the fog line.

In order that the method of using steam may be efficient, it should be based on the quantity of steam necessary to give the heat units required to raise the temperature of the intake air to that of the return, and also on the quantity of water necessary to saturate the volume of the intake air at the higher temperature; in order that the method may be practical, the admission of the steam should be under a convenient control.

CALCULATIONS AND TABLES.

The tables and calculations following are based on the use of live steam; but exhaust steam, which carries about 85 per cent of the heat value of live steam, may be used with success, provided the conditions at any given mine are carefully considered.

FORMULAS AND THEIR DERIVATION.

Formula *a* will assist in finding the quantity of water in the form of steam necessary to raise the temperature of the intake air to that of the return air. If the temperature of the return air is 60° F. and if live steam is used, the required quantity of steam, as shown by formula *c*, is a temperature of the function of the weight of the intake air and of the difference between the intake and that of the return air.

In developing the three formulas following, values have been assigned as follows:

t'=temperature of the outside air, in degrees Fahrenheit.

t=temperature of the return air, in degrees Fahrenheit.

$T=\frac{4t+3t'}{7}$, which will approximate the average temperature at which water will evaporate in a coal mine.

W=weight in pounds of 100,000 cubic feet of air at the temperature t'.

h=specific heat of air for constant pressure, or 0.2375 British thermal units.

H=total heat above 32° F. in 1 pound of steam at 3.5 pounds pressure, or 1,150 British thermal units.

L=latent heat of evaporation, in British thermal units, per gallon of water, or $9{,}116-58.5\,(T-32)$.

P=pounds of water in the form of steam required to furnish the heat units to raise the temperature of the weight of air W from the temperature t' to the temperature t, and is equal to $\frac{\text{weight of air} \times \text{specific heat of air} \times \text{pressure difference}}{\text{latent heat in 1 pound of steam above } t \text{ degrees}}$.

G=gallons of water corresponding to P or $\frac{P}{8.35}$.

D=gallons of water in 100,000 cubic feet of return air.

D′=gallons of water in 100,000 cubic feet of outside air.

The latent heat in 1 pound of steam at t degrees is equal to H, the total heat above 32°F., minus the sensible heat between 32 degrees and t degrees.

By using the values noted above, the following formulas may be derived:

$$(a)\quad G=\frac{P}{8.35}=\frac{Wh(t-t')+L(D-D')}{8.35[H-(t-32)]},$$

which is derived as follows:

Number of B. t. u. required to heat the weight of air (W)$=Wh(t-t')$.

Number of B. t. u. required to furnish the latent heat for the evaporation of the additional water in the heated air as compared with the water in the unheated $=L(D-D')$.

Number of B. t. u. per pound of steam at the temperature $(t)=H-(t-32)$.

Therefore
$$P=\frac{Wh(t-t')+L(D-D')}{H-(t-32)}$$

or
$$G=\frac{P}{8.35}=\frac{Wh(t-t')+L(D-D')}{8.35[H-(t-32)]}.$$

If $L=0$, which will be true when live steam is used, on the assumption that the heat given up by condensation will equal that absorbed by evaporation

$$(b)\quad G=\frac{P}{8.35}=\frac{0.028{,}443{,}1W(t-t')}{1150-(t-32)}.$$

If $t=60°$ F

$$(c)\quad G=0.000{,}025{,}34W(t-t').$$

QUANTITY OF WATER AS STEAM NECESSARY TO RAISE TEMPERATURE OF OUTSIDE AIR TO 60° F.

With the aid of the above formula, Table 12 has been prepared to show the quantities of live steam necessary to raise the temperature of 100,000 cubic feet of air from various temperatures of the outside atmosphere to 60° F. From the figures for the boiler horsepower and the tonnage of coal required to furnish the necessary steam the cost data can be prepared for comparison with other methods of humidification.

TABLE 12.—*Quantities of live steam required to raise a volume of 100,000 cubic feet of air from various outside temperatures to 60° F.*

Temperature of the intake air.	Weight of 100,000 cubic feet of air.	Heat required to raise 100,000 cubic feet of intake air to 60° F.	Water required in the form of steam at 3.5-pound pressure to raise temperature of intake air.	Boiler horsepower.	Coal required per 24 hours.
°F.	*Pounds.*	*B. t. u.*	*Gallons.*		*Tons.*
0	8,635.4	123,054	13.13	219	7.8
10	8,459.4	100,455	10.72	179	6.4
20	8,275.5	78,617	8.39	140	5.0
30	8,106.3	57,757	6.16	103	3.7
40	7,943.9	37,734	4.02	67	2.4
50	7,787.9	18,496	1.97	33	1.2
60	7,637.9	....	....	....	....

Figure 4 shows the quantity of steam, in gallons per minute, required to raise any given volume of ventilating current from different temperatures of the outside atmosphere to 60° F. The boiler horsepower and coal required to furnish the steam would be as follows:

Boiler horsepower and coal required to raise temperature of intake air.

Quantity of water, as steam, necessary to raise temperature of intake air to 60° F.	Boiler horsepower required to evaporate water.	Coal burned per 24 hours.
Gallons per minute.	*Continuous horsepower.*	*Tons.*
1	16.7	0.60
2	33.4	1.19
3	50.1	1.79
4	66.8	2.39
5	83.5	2.98
6	100.2	3.58
7	116.9	4.17
8	133.6	4.77
9	150.3	5.37
10	167.0	5.96
11	183.7	6.56
12	200.4	7.15
13	217.1	7.75
14	233.8	8.35
15	250.5	8.94
16	267.2	9.54
17	283.9	10.13
18	300.6	10.73
19	317.3	11.33

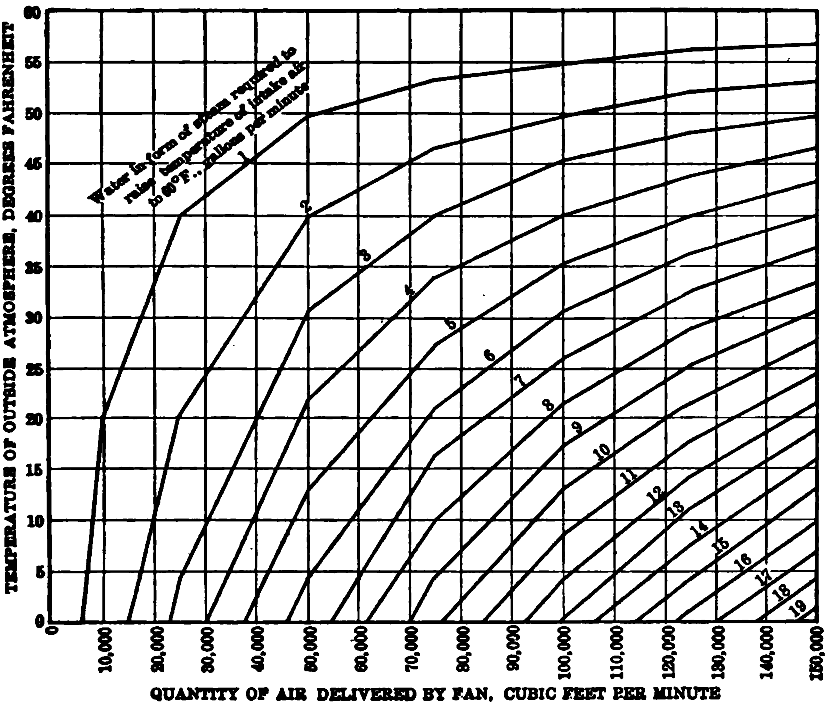

FIGURE 4.—Curves showing quantity of live steam necessary to raise temperature of 100,000 cubic feet of intake air from various temperatures to a temperature of 60° F.

QUANTITY OF WATER REQUIRED TO SATURATE IN RAISING TEMPERATURE.

The quantity of moisture in any volume of the outside atmosphere depends on the temperature and relative humidity of the air. A volume of 100,000 cubic feet at a temperature of 60° F. requires for saturation 9.843 gallons, or approximately 10 gallons, of water.

Figure 5 shows, for different temperatures and relative humidities of the outside air, the quantity of water that must be added per 100,000 cubic feet for saturation if the temperature of the air is raised to 60° F.

QUANTITY OF MOISTURE IN OUTSIDE AIR ACCORDING TO TEMPERATURE AND RELATIVE HUMIDITY.

A volume of intake air, when heated by steam to the temperature of the return, will contain a quantity of moisture equal to that in a similar volume of the outside atmosphere, plus the water vapor, entering the air from the steam. For certain temperatures and relative humidities of the outside air, this quantity of moisture will give 100 per cent relative humidity at the temperature of the return; often additional water must be supplied. This additional water may conveniently be added in the form of steam, if steam is used for preheating the air, or it may be introduced into the shaft by sprays. In many mines this additional moisture may be obtained naturally from the water falling down the shaft.

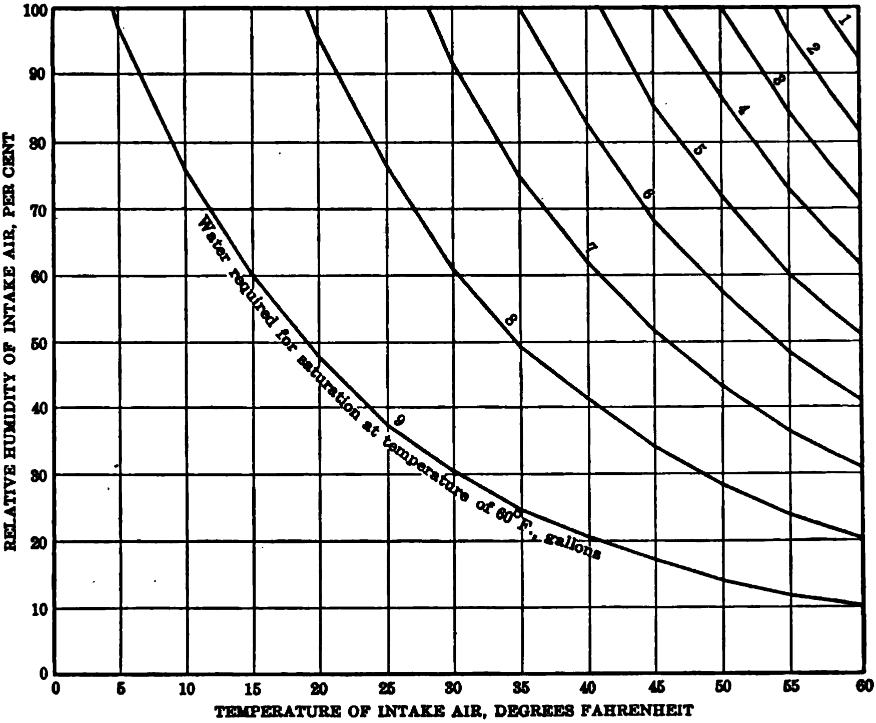

FIGURE 5.—Quantity of water necessary for different temperatures and relative humidities to cause saturation of the intake air if the temperature is raised to 60° F.

Table 13, which follows, shows different temperatures and relative humidities, the amount of water in the outside air, the quantity of steam required to raise the temperature of the intake to 60° F., and the water that must be supplied to produce 100 per cent relative humidity at 60° F.

TABLE 13.—*Data on premoistening 100,000 cubic feet of intake air.*

Temperature of outside air.	Water for steam necessary to raise temperature to 60° F.	Additional water required for saturation.	Relative humidity outside air.							
			30 per cent.		40 per cent.		50 per cent.		60 per cent.	
			Water in outside air.	Additional water required for saturation.	Water in outside air.	Additional water required for saturation.	Water in outside air.	Additional water required for saturation.	Water in outside air.	Additional water required for saturation.
° F.	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*
0	13.13	....	0.247	....	0.329	....	0.412	....	0.494	....
10	10.72	....	.399	....	.532	....	.665	....	.797	....
20	8.39	1.453	.634	0.819	.846	0.607	1.057	0.396	1.268	0.185
30	6.16	3.683	.994	2.687	1.325	2.358	1.656	2.027	1.987	1.696
40	4.02	5.823	1.463	4.360	1.951	3.872	2.439	3.384	2.927	2.896
50	1.97	7.873	2.094	5.779	2.792	5.081	3.490	4.383	4.187	3.686
60	....	9.843	2.953	6.890	3.937	5.906	4.922	4.921	5.906	3.937
70	....	9.843	4.106	5.737	5.474	4.369	6.843	3.000	8.212	1.631
80	....	9.843	5.633	4.210	7.510	2.333	9.388	.455	11.266	....

Temperature of outside air.	Water for steam necessary to raise temperature to 60° F.	Additional water required for saturation.	Relative humidity outside air.							
			70 per cent.		80 per cent.		90 per cent.		100 per cent.	
			Water in outside air.	Additional water required for saturation.	Water in outside air.	Additional water required for saturation.	Water in outside air.	Additional water required for saturation.	Water in outside air.	Additional water required for saturation.
° F.	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*	*Galls.*
0	13.13	....	0.576	....	0.658	....	0.741	....	0.823	....
10	10.72	....	.930	....	1.063	....	1.196	....	1.329	....
20	8.39	1.453	1.480	....	1.691	....	1.903	....	2.114	....
30	6.16	3.683	2.318	1.365	2.650	1.033	2.981	.722	3.312	0.371
40	4.02	5.823	3.451	2.408	3.902	1.921	4.390	1.433	4.878	.945
50	1.97	7.873	4.885	2.988	5.583	2.290	6.281	1.592	6.979	.894
60	....	9.843	6.890	2.953	7.874	1.969	8.859	.984	9.843	....
70	....	9.843	9.580	.263	10.949	....	12.317	....	13.686	....
80	....	9.843	13.143	....	15.021	....	16.989	....	18.776	....

Figure 6, based on the calculations presented in Table 13, shows for different temperatures and relative humidities of 100,000 cubic feet of outside air, both the sum of the water in the outside air plus that in the steam required to heat the air to 60° F., and the additional water necessary for saturation at 60° F.

EXAMPLES SHOWING USE OF TABLES.

The following examples will illustrate the use of Tables 12 and 13:

1. Assume the temperature of the return air to be 60°; the temperature of the intake air to be 20°; the relative humidity of the intake air to be 40 per cent; and the quantity of air per minute to be 100,000 cubic feet. The tables show that the water in the steam required to raise the temperature will be 8.39 gallons per minute; that the horsepower necessary to furnish the steam will be 140 horsepower; that the coal necessary to furnish the steam will be 5.0 tons per 24 hours; and that the additional water necessary for saturation will be 0.607 gallon.

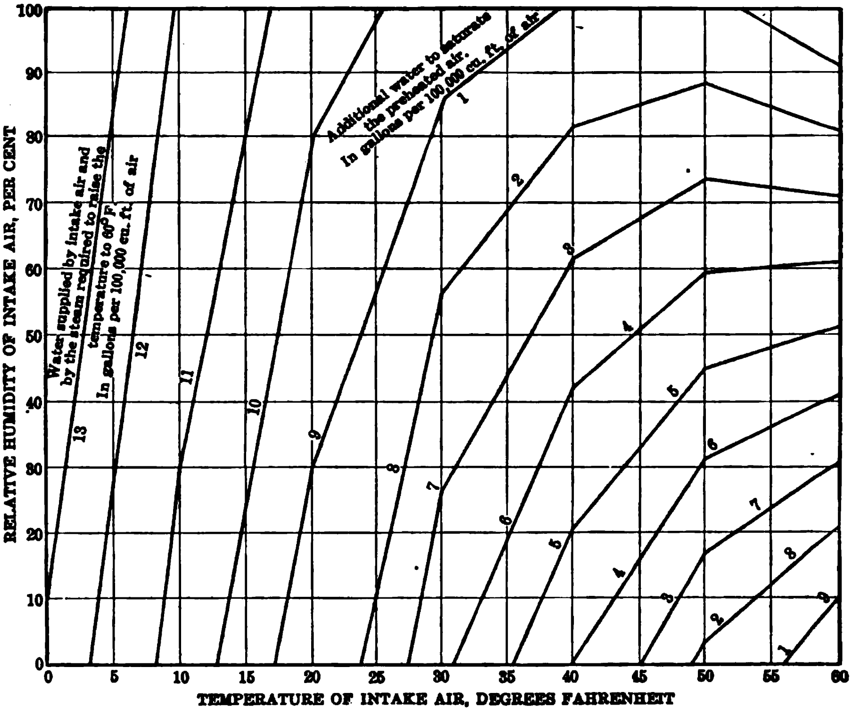

FIGURE 6.—Curves showing variation in quantity of water needed to saturate different 100,000 cubic-foot volumes of air of different relative humidities and temperatures.

2. Assume the temperature of the return air to be 60°; the temperature of the intake air to be 10°; the relative humidity of the intake air to be 85 per cent; and the quantity of air per minute to be 50,000 cubic feet. The tables show that the water in the steam required to raise the temperature will be 5.36 gallons [½(10.72)]; that the horsepower necessary to furnish the steam will be 89.5 horsepower [½(179)]; that the coal necessary to furnish the steam will be

3.2 tons per 24 hours [½(6.4)]; and that the additional water necessary for saturation will be 0.

3. Assume the temperature of the return air to be 60°; the temperature of the intake air to be 60°; the relative humidity of the intake air to be 70 per cent; and the quantity of air per minute to be 75,000 cubic feet. The tables show that the water in the steam required to raise the temperature will be 0; that the horsepower necessary to furnish the steam will be 0; that the coal necessary to furnish the steam will be 0; and that the additional water required for saturation will be 2.215 gallons per minute [¾(2.953)].

METHOD OF INTRODUCING STEAM FOR HUMIDIFICATION.

The time may come when thermostats will be used for controlling the introduction of steam into mines to raise the temperature of the intake air. At present no automatic mechanism has been applied for this purpose.

A convenient arrangement may be provided at any mine by installing at the intake a series of steam pipes, each controlled by a valve and each of a diameter calculated to permit the outflow of steam in quantities per minute sufficient to raise a given number of degrees the temperature of the intake air. The successful use of this device involves a knowledge of the volume of the ventilating current in cubic feet per minute, of the initial gage pressure of the steam in pounds per square inch, and of the temperature interval at which the device will act.

EXAMPLE AND CONVENIENT TABLES.

The use of this method of control may be illustrated by the following example: Assume that the volume of the ventilating current delivered by the fan is 100,000 cubic feet per minute, that the initial gage pressure of the steam is 100.3 pounds per square inch, and that it is desired to control the temperature of the intake air within a range of 6° F. It is then necessary to arrange a series of pipes that will control the outflow of steam required to heat the intake air to 60° F. from any temperature of the outside air. At the gage pressure of 100.3 pounds per square inch, the discharge per square inch of orifice, according to Table 14 following, will be 11.83 gallons per minute. The internal sectional areas of different sizes of standard pipes are given in Table 15 following. From these tables it is possible to arrange the following schedule for the problem under consideration.

Typical mine schedule for control of steam introduction.

Temperature of the outside air.	Required temperature raise.	Pipes required to be opened.		Water per minute in steam discharge.
		Number.	Size.	
° F.	*° F.*		*Inches.*	*Gallons.*
54	6	1	1/4	1.23
48	12	1	3/8	2.26
42	18	1 1	1/4 3/8	3.49
36	24	2	3/8	4.52
29	31	1	3/4	6.31
23	37	1 1	1/4 3/4	7.54
19	41	1 1	3/8 3/4	8.57
14	46	1 1 1	1/4 3/8 3/4	9.80
10	50	2 1	3/8 3/4	10.83
5	55	1 2 1	1/4 3/8 3/4	12.06
0	60	3 1	3/8 3/4	13.09
−4	64	1 3 1	1/4 3/8 3/4	14.32

The above schedule indicates that five pipes will be required for the discharge of the steam, as follows: One ¼-inch pipe, three ⅜-inch pipes, and one ¾-inch pipe.

TABLE 14.—*Outflow of steam into atmosphere.*[a]

Gage pressure per square inch.	Discharge per square inch of orifice per minute.	Gage pressure per square inch.	Discharge per square inch of orifice per minute.
Pounds.	*Gallons.*	*Pounds.*	*Gallons.*
35.3	5.28	100.3	11.83
45.3	6.29	120.3	13.85
55.3	7.31	140.3	15.83
60.3	7.82	150.3	16.82
75.3	9.33	200.3	21.75
85.3	10.34		

[a] From Kent's mechanical engineers' handbook.

TABLE 15.—*Dimensions of standard pipe.*[a]

Size.	Internal sectional area.	Size.	Internal sectional area.
Inches.	*Sq. inches.*	*Inches.*	*Sq. Inches.*
⅛	0.057	1¼	1.496
¼	.104	1½	2.036
⅜	.191	2	3.356
½	.304	2½	4.788
¾	.533	3	7.393
1	.864		

[a] From Briggs's standard.

In the practical use of the method of steam introduction outlined above a "top man" should be detailed to make daily two or three thermometer readings of the temperature of the outside air and to open the valves of the steam pipes designated on the particular schedule in effect at the mine.

The above outline of the use of steam humidification is based on the fact that in many mines there is sufficient moisture to keep the dust damp, provided the ventilating currents are prevented from removing the moisture naturally present. Therefore the suggestion has been made that the temperature of the intake air be raised to that of the return. If a condition of extreme dryness makes it necessary to have an active sprinkling take place in the mine, the intake air can be heated slightly above the temperature of the return; and for every degree that 100,000 cubic feet per minute of intake air at 100 per cent relative humidity is heated higher than the return approximately 500 gallons of water will be deposited in 24 hours throughout the mine on ribs, roof, and floor. This deposition of moisture is probably made throughout the mine because of the gradual drop in the fan pressure on the air, the reduction in pressure causing a lowering of temperature and a consequent deposition of moisture.

COST OF USING STEAM.

The cost of using live steam would be about 44 cents per 1,000 gallons of water evaporated under the conditions observed, as compared with 60 cents for the methods calling for the use of hose, and $1 for the water-car method. Exhaust steam from the fan and other engines would be used at most mines to furnish the greater part of the heat and moisture required properly to humidify the intake air, and live steam would be installed and used to bring about the desired control and adjustment. The combined use of exhaust and live steam under proper control would present the most economical, efficient, and practical method for mine humidification under the conditions that prevail in Illinois.

In mines where the volume of the ventilating current is very large and the quantity of steam required to humidify the intake air would be excessive, advantage may be taken of the heat radiated from the surfaces of the entries by conducting the steam in pipes a distance of a thousand or more feet into the mine before allowing it to exhaust into the intake air. Reference has already been made to the large quantity of heat that the ribs will yield to the air currents, and the possibility of practical utilization of this heat is apparent.

METHODS OF HUMIDIFICATION IN HOT WEATHER.

The above discussion relates to humidifying mine air during the colder months of the year. There are, however, some mines in which the heat of the intake air during the summer months causes heavy falls, due to the expansion of friable roofs, and in which it would be desirable to lower the temperature of the intake air when it is over 10° higher than that of the return air.

In some mines, during the hot weather, cakes of ice are strewn along the intake air courses to lower the temperature of the intake air. This method is expensive and not as efficient as the one described below.

Table 9 indicates that the average temperature of the return air of a room-and-pillar mine is 63.85° F. Table 7 shows that the average temperature of the outside air during the night is at no time over 10° F. higher than that of the return air, but that at 2 o'clock in the afternoon the average temperature of the outside air exceeds by more than 10° F. the mine temperature during June, July, August, and September. During these four months the relative humidity of the outside air (Table 8) at 4 a. m. is 86 per cent, and at 2 p. m. is 52 per cent. These figures make it evident that if water be sprayed into the intake shaft during summer the latent heat of evaporation will reduce the daytime temperature of the intake air.

Figure 7 shows the drop in the temperature of different volumes of air caused by the evaporation of 1 gallon of water.

The practical method of applying evaporation is to arrange a series of spray nozzles of known capacity at the top of the intake shaft. The smaller the particles of water delivered by the nozzles the more rapid will be the evaporation of the water. During the daytime of clear days in hot weather a sufficient number of these nozzles should be operating to give the desired drop in temperature to the known volume of the ventilating current. Care should be exercised not to reduce the intake temperature below that of the return, in order that the air currents may not rob moisture from the mine workings.

GENERAL CONCLUSIONS.

The fact that dry bituminous coal dust will explode under certain conditions has been proved both by the experience of the past and by laboratory tests and is now generally admitted by all coal men.

That coal dust may be rendered inert by the proper application of moisture has been shown both by laboratory tests and by the absence of explosions at mines in which moisture is present in the proper proportion to the quantity of dust produced.

The result of the investigation in Illinois of the humidity of mine air leads to the belief that steam may be applied to the intake air in such a way as to offer the most economical and efficient method of dampening coal dust.

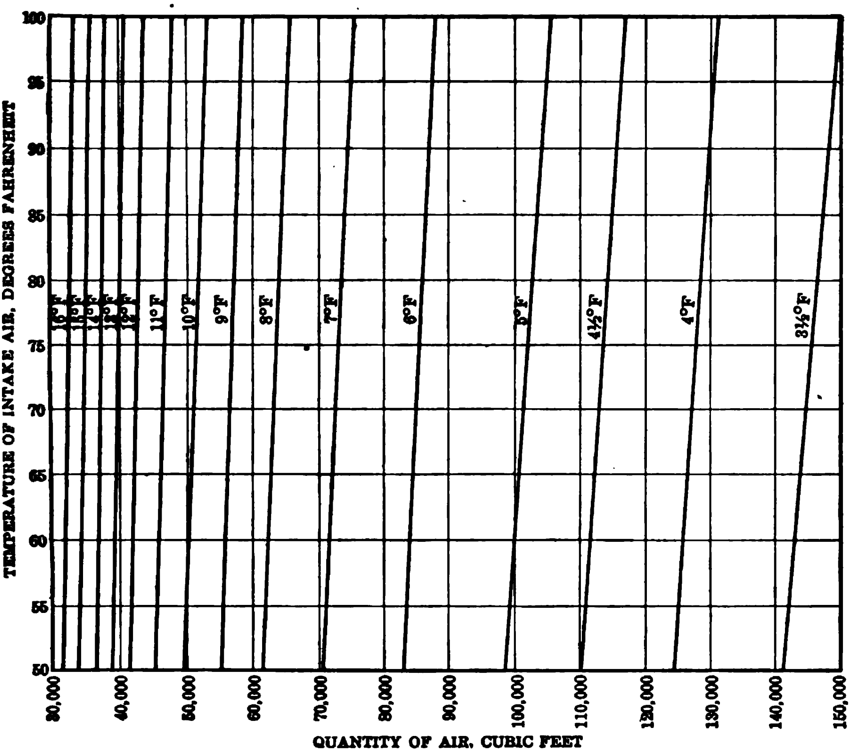

FIGURE 7.—Curves showing influence on temperature of different volumes of air caused by evaporation of 1 gallon of water.

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Bulletin 84

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, Director

METALLURGICAL SMOKE

BY

CHARLES H. FULTON

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METALLURGICAL SMOKE.

By CHARLES H. FULTON.

INTRODUCTION.

One of the first matters to receive attention from the Bureau of Mines when it was authorized to undertake investigations relating to wastes in the mineral industries was the smoke problem at smelting and ore-roasting plants. This bulletin is published by the bureau primarily for persons who seek information on the subject of metallurgical smoke but are unacquainted with the technical language of metallurgy. It is, of course, not possible to free a technical subject entirely from technicalities, but, so far as possible, statements are made in plain and simple English.

Metallurgical smoke causes considerable friction between the metallurgical industry and agriculture in certain parts of the country. Owners of smelting plants are making every effort to devise ways and means to do away with possible damage and annoyance from smoke and are meeting with success. The problem is peculiarly difficult in this country because of the large tonnages of material that must be handled. The solution of the problem is not yet at hand and much work still remains to be done. As the mineral industry is one of the great basic industries of the country and of necessity is entitled to full consideration, it should be accorded freedom to work out the smoke problem to the benefit of all concerned. The effort is made in this paper to present the problem of metallurgical smoke as it actually exists, without bias of any kind.

The writer takes this opportunity to express his indebtedness to the sources of information on which he drew, many of which are enumerated in the selected bibliography at the end of this paper. In connection with the author's use of the present tense in describing plants and methods, the reader should note that none of the references in that bibliography is of later date than 1913. Much information was obtained by personal visits and by correspondence, and thanks are hereby tendered to all persons who kindly furnished data. The writer desires to thank particularly F. G. Cottrell, chief chemist of the Bureau of Mines, for criticism and comment.

GENERAL ASPECTS OF THE SMOKE PROBLEM.

DISTILLATION AND SUBLIMATION.

Distillation is a metallurgical process whereby a metal is converted into its vapor and then "condensed" if it collects in the liquid state, or "sublimed" if it passes directly from the vapor into the solid state without going through the intervening liquid condition. In the general sense "distillation" embodies both the vaporization from the liquid state and subsequent condensation of the metal again to liquid, whereas "sublimation" embodies the vaporization from the solid state and the subsequent sublimation to the solid state again, the liquid state not being passed through. Examples of distillation in metallurgy are the recovery of mercury from the ore, usually cinnabar (HgS), by heating in retorts or furnaces with free access of air, distilling the mercury as vapor, and cooling the vapor so that the metal is caught as liquid mercury in suitable receivers. Zinc is also obtained by distillation.

In some smelting operations one of the products is "fume." Thus, in lead smelting in the blast furnace, a part of the lead, 3 to 10 per cent, is volatilized in the furnace in the form of lead oxide and lead sulphide, with compounds of arsenic and antimony. These vaporized compounds of the metals pass with the furnace gases through the flues and dust chambers and thence out through the stack. When the temperature is low enough they sublime in the flues and chambers into finely divided solid material called "fume." This fume, however, is in many cases so exceedingly fine that it does not completely settle in the flues and chambers, but unless removed by filtering or in some other way it is carried out of the stack with the gases.

For these reasons the following usage has been adopted in this bulletin:

"Distillation" involves (1) the vaporization of the substance, and (2) the "condensation" of this vapor into a liquid. "Sublimation" embodies (1) the vaporization of a substance, and (2) the "subliming" of the vapor into fine solid particles without the intervening liquid stage being passed through. The word "condense" when used in this connection is therefore associated with the liquid state, whereas the word "sublime" is associated with the solid state. Thus when it is stated that the fume is condensed and sublimed it means that there are produced from the vapor state both minute liquid particles and minute solid particles.

COMPOSITION OF SMOKE.

In this bulletin the term "metallurgical smoke" is applied to the gases and vapors, and the fine dust entrained by them, that issue from the throat of furnaces, such as blast furnaces, reverberatory smelting furnaces, or roasting furnaces. At some plants the amount of metallurgical smoke is very great, exceeding in weight that of the solid material charged into the furnaces. Thus, in pyritic smelting, 1 ton of furnace charge requires approximately 1.2 tons of air to smelt it and gives rise to 1.4 tons of metallurgical smoke or furnace gases. Also, the volume of this smoke in the highly heated condition (400° C.) in which it is discharged from the furnace is very large. In the instance cited it is about 71,620 cubic feet per ton of charge. As an example of the tremendous amount of metallurgical smoke produced by a large smelting plant, data regarding the Washoe smelter at Anaconda, Mont., are given. That plant, which produces about 300 tons of copper a day, has a stack 30 feet in internal diameter at the top, which discharges 1,341,000,000 cubic feet of smoke per 24 hours, as measured at standard pressure and temperature. The smoke contains about 1.734 per cent by volume of sulphur-dioxide gas, or 23,252,940 cubic feet per 24 hours, which is equivalent to 2,092.7 tons, and, if converted into sulphuric acid, would yield 3,427 tons of acid, with a specific gravity of 66° B., or about one-third of the total production in the United States.

The volume of smoke discharged from the top of a stack is, of course, decidedly larger than that discharged at the throat of the furnaces of the plant, for in its passage through the dust chambers and flue system the smoke mixes with air that filters in accidentially or is purposely admitted.

GASES IN SMELTER SMOKE.

Metallurgical smoke consists of three distinct substances—the gases, the flue dust, and the fume. Of these three constituents the gases make up the great bulk. They come from the air used in the furnace to burn the fuel and also from chemical reactions between the oxygen of the air and certain constituents of the charge. The most abundant gas is nitrogen, derived from atmospheric nitrogen, which passes through the furnaces unaltered. It constitutes usually from 60 to 87 per cent of the bulk of metallurgical smoke. The gas ranking next in importance is carbon dioxide, which is produced by the combustion of carbon. The proportion of this gas ranges usually from about 3 to 5 per cent in pyritic smelting to about 15 to 20 per cent in lead blast-furnace smelting, being between these figures for other types of blast-furnace and reverberatory-furnace smelting. Gases from furnaces for roasting sulphide copper ores usually contain only insignificant amounts of carbon dioxide, but those from furnaces roasting lead ores and gold and silver ores in which carbonaceous fuel is burnt contain appreciable amounts of it. Carbon monoxide, a very poisonous gas, is not present to any degree in metallurgical smoke arising from the treatment of lead, copper, gold, and silver ores, the only exception being the gases from lead blast-furnace smelting, in which, on account of the considerable amount of fuel used, 5 to 10 per cent of carbon monoxide is found. Otherwise, the content of carbon monoxide rarely exceeds 1 per cent and sometimes is much less, none at all being present in gases from pyritic smelting and from certain roasting furnaces.

SULPHUR DIOXIDE.

The gases so far discussed are the constituents of smelter smoke that came no complaint and are the same as those that issue from the ordinary boiler stack. Sulphur dioxide is the constituent that largely gives smelter smoke its annoying properties, and requires special attention in order to be rendered innocuous.

In both roasting and smelting furnaces the sulphur dioxide is from the combustion of the sulphur in the sulphide minerals of the ore charge. The proportion of the gas in metallurgical smoke varies widely. The maximum amount of sulphur dioxide is found in the gases from pyritic smelting, where it constitutes 4½ to 9 per cent by volume of the smoke issuing from the furnace throat. In partial pyritic copper smelting the gases contain 2 to 3½ per cent of sulphur dioxide. The gases from the roasting of sulphide copper ores contain about 2 to 2.75 per cent, and may contain more. Gases from reverberatory smelting furnaces contain usually less than 0.5 per cent of sulphur dioxide, except in the early stages of smelting, during charging. Gases from lead-smelting furnaces and from the roasting of ores, not mentioned above, contain smaller proportions, usually less than 1½ per cent. These figures refer to the gases as discharged from the furnaces. Before the gases pass off into the atmosphere from the stack, they are usually considerably diluted. Formerly metallurgical smoke containing 1.5 per cent and more of sulphur dioxide was frequently discharged from stacks at some plants, but at present, in consequence of a number of court decisions, this proportion is reduced by dilution before the smoke reaches the stack. Some recent court decisions have fixed the amount of sulphur dioxide that may be discharged from stacks in certain districts at not more than 0.75 per cent.

SULPHUR TRIOXIDE.

Another gas found in metallurgical smoke, as it issues at high temperature from the furnaces, is sulphur trioxide or sulphuric anhydride. This substance, when combined with water, molecule for molecule, forms common sulphuric acid. It is formed in furnace operations by the oxidation of sulphide minerals, although in comparatively small amounts under most conditions, and if the temperature be high enough, the gas is found in the discharged gases, uncombined with water or with metallic bases, as sulphur trioxide. As the gases cool it combines with water, present in the metallurgical smoke as steam, to form sulphuric acid which, when the temperature becomes low enough, finally condenses. In this way sulphur trioxide in smelter smoke becomes an exceedingly harmful substance, one that may do a great deal of damage. Fortunately, however, it is present only in small proportions and, unlike the true gases, can be, and in most instances is, removed from the smoke before it is discharged into the atmosphere. Usually gases from blast-furnace smelting of copper and lead ores contain insignificant amounts of sulphur trioxide, but gases from roasting furnaces may contain appreciable proportions. Thus, for instance, gases from the roasting of sulphide copper ores may contain 0.25 to 0.75 per cent of sulphur trioxide. Aside from the gases mentioned, metallurgical smoke contains considerable water in the form of steam and occasionally small quantities of hydrocarbons and other gases.

FLUE DUST IN SMELTER SMOKE.

The flue dust that is carried by the gases as they issue from the furnaces consists of small particles of the different ores, fluxes, or fuel. The gases, on account of their high velocity, entrain these fine solid particles, and when the velocity is reduced in the dust chambers the particles settle. The amount of flue dust made in different smelting operations varies greatly, depending on such factors as design of furnace, fineness or coarseness of the ore charge, blast pressure or draft employed, and working condition of the furnace. Estimated on the basis of the tonnage of charge that is smelted, it may range from a fraction of 1 per cent up to 10 per cent of the weight of the furnace charge. Thus, in copper blast-furnace smelting at the Copper Queen mine in Arizona, 1 ton of flue dust was produced for every 17 tons of charge, or 5.55 per cent. In some instances, it is convenient to express the amount of flue dust in metallurgical smoke in the manner that iron metallurgists state the content of dust in blast furnace gas, that is, in grains (weight) per cubic foot of gas. This manner of expressing the dust content of the smoke stream is particularly applicable after the settling of the coarser particles.

Generally speaking, the composition of the flue dust is similar to that of the ore charge of the furnace. However, it may be modified by some constituents of the charge being finer than others and thus producing more dust. Flue dust is usually considered a troublesome product, for its treatment is rather difficult and expensive, although a number of successful methods have been devised. Flue dust varies in size from 0.25-inch to 0.125-inch particles of the lighter material down to an extremely fine dust resembling impalpable powder. No matter what its size, however, true flue dust is to be considered as composed of original particles of the ore charge, either unaltered or chemically changed. Its precise composition is discussed in some detail on pages 27 and 28.

FUME IN SMELTER SMOKE.

Related to the flue dust and in part recovered with it is the fume. Sometimes no distinction is made between fume and flue dust, for in copper smelting the amount of true fume is relatively small although certain plants treating arsenical ores make considerable fume. In lead smelting, however, considerable amounts of fume are produced. The fume consists of the volatile constituents of the ore charge which pass off in the form of vapor with the gases from the furnaces and then sublime into minute solid particles in the cooler part of the flues. In examining the dust chambers and the flue system of a smelting plant, it will be noted that a fine gray-white impalpable powder is deposited on the walls, roof, and floor of the flues near the stack end of the flue system, the amount being greatest near the stack and diminishing as the furnace is approached. In certain sections of the flues and chambers this fume is mixed with the fine flue dust.

On chemical examination, the fume is found to consist largely of lead compounds—such as the sulphate, sulphide, and oxide—arsenious oxide, antimony oxide, zinc oxide and zinc sulphate, elemental sulphur, and small amounts of the oxides of tellurium and selenium. In general, all the volatile constituents of the ore charge are represented. The fume frequently contains appreciable amounts of silver, which is markedly volatile under certain conditions. The particles of fume are very fine and are under the stress of certain physical forces, so they do not settle easily, as most of the flue dust does, but in large proportion pass through the stack with the gases and spread over the surrounding country, unless special preventive methods are applied, as is now generally done.

When zinc minerals are present in the furnace charge, sulphur trioxide is rendered harmless in so far as the formation of sulphuric acid is concerned, as sulphur trioxide unites with zinc oxide to form sulphate of zinc, which is deposited with the fume. Lead minerals act similarly to zinc minerals, when the action in the furnace is oxidizing. The most harmful constituents of true fume, if allowed to escape from the stack, are sulphuric acid, lead compounds, and arsenious oxide. These substances settle on the surrounding country and under certain atmospheric conditions of moisture and wind may sufficiently concentrate to damage vegetation and injure animal life.

Most of the more modern metallurgical plants make ample provision for the recovery of fume from furnace smoke. Stacks from metallurgical works sometimes emit white or grayish-white smoke of varying degrees of density, from smoke that is barely visible to very dense smoke. In general one may state that it is the fume that renders metallurgical smoke visible, except when visibility is due to the condensation of water vapor on cold days. The true gases in metallurgical smoke are invisible.

From the foregoing statement it is evident that metallurgical smoke contains three distinct types of constituents—gases, flue dust, and fume. Of these, all the true gases escape from the stack, and all but the very finest flue dust may be settled out by ample and properly designed dust chambers. The fume is only partly recovered with the flue dust even by the use of elaborate dust chambers and flues; special apparatus must be erected to recover it.

HARMFUL CONSTITUENTS OF METALLURGICAL SMOKE.

It is apparent that the harmful constituents of smelter smoke are: (1) Sulphur dioxide gas, (2) sulphuric acid formed from sulphur trioxide generated in the furnaces, (3) fume or fine solid particles containing lead and arsenic compounds and acid sulphates.

SULPHUR DIOXIDE.

The most important of these at the present time is sulphur dioxide, for no universally applicable method of eliminating this gas from smelter smoke is as yet in commercial use. Sulphur trioxide and sulphuric acid in their behavior may be classed with true fume, and methods can be used that will completely eliminate these substances from the smoke before it is discharged into the atmosphere. Sulphur dioxide is therefore considered first.

SULPHUR DIOXIDE IN AIR.

Sulphur dioxide is an invisible gas with a highly characteristic penetrating and pungent odor familiar to almost everyone. It is stated that air containing 0.007 per cent by volume of this gas is unbreathable, the gas not having a toxic effect but causing a spasmodic contraction of the air cells of the lungs. Only very small proportions of the gas are ordinarily present in the atmosphere, but measurable quantities are invariably found in the air of cities, especially manufacturing centers where much coal is burned, being formed by the combustion of the sulphide minerals in coal, all coal containing some sulphur. As an example of the quantity of sulphur dioxide in the air of cities, the following examples are given: Berlin, 0.000035 per cent by volume; London, 0.00039 per cent; Glasgow, 0.00042 per cent. Air in the vicinity of Salt Lake City, Utah, contained from 0.001 per cent to none.

The gas may be detected in the air by its smell if present in the ratio of more than one part to 150,000 parts of air. Chemically, it may be readily detected in the air when the proportion present is more than one part per million. It is probable that the sulphur dioxide of the air is slowly converted into sulphuric acid by the action of the oxygen and the water vapor in the air, and is finally precipitated with the rain or snow.

EFFECT OF SULPHUR DIOXIDE ON VEGETATION.

All vegetation is not equally affected by the sulphur dioxide in the air. It is stated on good authority that the soft, resinous woods, especially spruce, are most readily injured, one part of sulphur dioxide in 500,000 parts of air doing damage if permanently present. One part in 500,000 is equivalent to 0.0002 per cent by volume or to 0.0071 grams of sulphur dioxide per cubic meter of air. The trees most resistant to the action of sulphur dioxide are the hardwoods, such as oak and maple. There are comparatively few reliable data as to the precise effect of sulphur dioxide and the amount that is necessary to damage different varieties of plants.[a] In recent litigation in this country on smelter smoke, much expert testimony was offered by both sides. That this evidence would be contradictory was to be expected with the present systems of taking expert testimony, and, for evident reasons, much of the evidence had no practical or scientific value.

The prevention of damage to vegetation by sulphur dioxide in the atmosphere needs the close study of disinterested scientific observers. It is probable that the question will be prominent for some time, as the elimination of sulphur dioxide from smelter smoke seems to be in the far future, and the amelioration of conditions depends at present on diluting the smoke until the sulphur dioxide content is so small as to be harmless. This in itself is a great engineering problem for large plants. The point to which dilution must be carried no doubt depends upon the nature of the vegetation in the vicinity of the plant, the climate, whether wet or dry, and the season of the year. It has been found in Europe and in Japan that some crops are much more subject to injury at certain stages of their growth than at others, and proposals have been made to put seasonal limitations on the tonnage of sulphide ore smelted. An agreement of this kind is in effect in southeastern Tennessee and also in Japan. During the active crop-growing season the amount of high sulphur-bearing ore smelted is reduced, and is again increased during the fall and winter months.

That comparatively small amounts of sulphur dioxide and sulphuric acid (it must be borne in mind that sulphur dioxide is probably slowly converted into sulphuric acid in the atmosphere) do harm to vegetation is attested to by an examination of the trees in large industrial cities. These, although in many instances seemingly healthy, suffer from sulphate poisoning and lose their leaves much earlier in the fall than other trees in the vicinity not subject to the contaminating atmosphere.

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[a] For a full discussion, see Report of the Selby Smelter Commission, 1915.

Sometimes, in discussions of the smelter-smoke question by interested parties, gross misstatements of fact are made. Thus, the agriculturist sees bad effects from smelter smoke which other people can not find, and the metallurgist sometimes goes so far as to claim that smelter smoke is beneficial to vegetation, and even to assume that the damaging constituent of smelter smoke is sulphur trioxide only, giving rise to sulphuric acid, and to ignore entirely the sulphur dioxide content. It is evident that nothing is to be gained by a policy of this kind, and the solution of the problem lies in its study by disinterested commissions, such as the one recently appointed by an agreement between the Department of Justice and the attorneys of the Anaconda company to study the smoke question in Montana in its relation to the forest reserves.

COMPARISON OF SMELTER-SMOKE PROBLEM IN EUROPE AND THE UNITED STATES.

The problem of damage by smelter smoke has been rather closely studied for many years in European countries, particularly in Germany, and it might seem as if profit could be derived from this study and experience. Although this is true to some extent, a closer study of the question shows that conditions in Europe and in this country are widely different. At European plants the amount of sulphide ores smelted, and hence the volume of fume discharged from furnaces there, is small as compared with those at plants in this country. Moreover, methods for handling smoke that are satisfactory for small operations are inadequate and unsatisfactory for large-scale work. For example, relatively small amounts of smoke may be put through a washing process and most of the deleterious constituents removed, but such washing is impracticable for very large volumes of gas. Even in European countries where it would seem that the question might have been settled long ago, contention between metallurgical and agricultural interests is not infrequent. Evidently the problem is inherently difficult to solve in a satisfactory manner.

SMELTER-SMOKE PROBLEM IN THE WESTERN STATES.

It is interesting to review the general situation in this country. Much of the trouble is in the Western States, and chiefly in Montana, Utah, and California. In those States very large tonnages of sulphide ores are smelted with the production of great volumes of smelter smoke, comparatively rich in sulphur dioxide and other injurious constituents. A number of the smelters have been in operation for many years and have experienced difficulties only comparatively recently.

It is well known that mining and metallurgy are often the advance agents of civilization and the pioneers in establishing industrial centers. In many localities the mines and the smelters supplied by them were essentially the only locators within great stretches of territory. Around them grew communities and cities dependent upon them for a livelihood. Agriculture was of relatively small importance. But conditions changed with the increasing population of the country and the closer settlement of the West and, coincident with this increase, an agricultural industry has grown which, in the vicinity of the smelting plants, naturally suffers in some degree from smoke. Still, the area damaged is comparatively small and even this damage can be much lessened by the adoption of proper means. Since some plants have installed devices to remove sulphuric acid from the smelter smoke and have the smoke diluted with air, conditions have been much ameliorated in certain districts.

EFFECT OF CLIMATIC CONDITIONS.

The damage that smoke can do depends largely on climatic conditions, especially the relative humidity of the atmosphere. The damage is much greater in regions having a moist than in those having a dry climate. In this respect such States as Montana, Utah, and Arizona are favorably located and smelters in those States can probably discharge gases richer in sulphur dioxide without doing damage than those in regions having a moister climate. For the same reason, the damage done at different seasons of the year varies. The moist atmosphere and the rains of spring and early summer conduce to increase damage to vegetation, particularly as at that time of the year the early crops and young vegetation are susceptible to serious injury.

It has been customary to discharge the smelter smoke by means of very tall chimneys, on the assumption that if the noxious gases are discharged at considerable height they will have opportunity to diffuse more thoroughly and thus become so diluted as to be comparatively harmless, but the efficiency of this method is now being questioned. There is reason to believe that the use of high stacks increases the area subject to damage, whereas low stacks may intensify the damage but concentrate it within a smaller area. Probably high chimneys do not serve their purpose as well as was anticipated, and perhaps at present the better method may be to dilute the smelter smoke and discharge it from a number of stacks.

SMELTER SMOKE IN LARGE CITIES.

In reference to smoke and damage suits by agricultural interests in western States, the statement is sometimes made that it is not apparent why these plants should be complained of, when within such cities as Omaha, Chicago, and New York there are large smelters and metallurgical works that discharge smelter smoke. The answer to this statement is to be found in the fact that none of the metallurgical plants near these cities treats any considerable tonnage of high-sulphur ores. Most of the smelters so situated are refining plants, which may smelt some sulphide ores, but not in sufficient amount to cause trouble from sulphur dioxide. It frequently happens, too, that there is serious complaint in cities concerning the smoke nuisance and damage from metallurgical plants. For instance, Staten Island, N. Y., sometimes suffers inconvenience, if not damage, from certain large metallurgical plants in New Jersey. This inconvenience, however, is not caused by sulphur dioxide, so far as the writer knows, but rather by fume and, in certain cases, fine flue dust. Such conditions as these are comparatively easy to remedy.

THE THREE MAIN CONSTITUENTS OF METALLURGICAL SMOKE.

Metallurgical smoke is here discussed in detail in relation to the three main constituents of smelter smoke—gas, flue dust, and fume. In practice it is difficult to sharply differentiate flue dust and fume. Smelter smoke issuing from the throats of furnaces contains all three constituents. The flue dust, consisting of the solid particles entrained by the gases, settles as the velocity of the gas stream diminishes, all except the very fine particles, which may not settle even in the lowest velocity that the gas stream attains in its passage through the flues and dust chambers. Certain metallic vapors that form in the furnaces, as for instance, arsenious oxide, lead sulphide, and zinc oxide,[a] are carried as true vapors in the smoke at high temperature, and then, as the smoke passes to the cooler parts of the flue system, they sublime as very small particles of fume. As an illustration the following example is given. If solid arsenious oxide be placed in a vessel that is open to the air at normal pressure (760 mm. of mercury), and then heated, it sends off into the space above it arsenious oxide vapor, which in part replaces the overlying air. Before the arsenious oxide is heated the pressure overlying it (760 mm.) is due solely to the air. After the heating however, although the total pressure is still 760 mm., part of the pressure is due to the vapor of arsenious oxide, which has replaced a certain amount of air. The pressure making a part of the total pressure overlying the solid is termed a "partial pressure," and is due to the vapor tension of the solid at the existing temperature. The vapor tension varies with the temperature. Vapor tension is small at a low temperature, and increases as the temperature rises.

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[a] Zinc oxide is not markedly volatile except at high temperature. Under certain furnace conditions, however, zinc and zinc compounds vaporize, and these vapors are converted into oxide fume when they come in contact with atmospheric oxygen.

PRECIPITATION OF SMOKE PARTICLES.

EFFECT OF VAPOR TENSION.

A substance boils when its vapor tension becomes equal to the pressure above it. If a volume of gas that has been in contact with solid arsenious oxide at a certain high temperature, and hence is saturated with arsenious vapor, be removed and passed with other gases to cooler parts of a flue system, arsenious oxide will sublime or precipitate in very fine solid particles, because, with a falling temperature, the vapor tension of the arsenious oxide diminishes and part of the oxide sublimes into the solid state.

An analogous case is that of water vapor. If water overlain by air be heated, water vaporizes as steam, which will mix with the air above it in the same manner as the arsenious oxide mentioned above. If this vapor-laden air be removed from above the water and passed to regions where the temperature is lower, water will precipitate or condense from the air in the shape of fog or rain. Air ordinarily contains water vapor, the amount being dependent on the temperature. If, for a given temperature, the air holds all of the water vapor possible, it is said to be saturated. When saturated air cools water is precipitated. For every given temperature the air has a definite saturation point as regards water vapor. If the amount of water in a given volume of air is not sufficient to saturate it for the temperature existing, and it is then cooled, water will not precipitate from the air until a temperature is reached that represents the saturation point for the amount of water vapor present.

These principles govern the condensation and sublimation of liquid and solid particles respectively from their vapors carried in gas streams. In metallurgical smoke, they are particularly applicable to water vapor, sulphur trioxide, sulphuric acid, arsenious oxide, zinc oxide, lead sulphide, etc. At the high temperature in metallurgical furnaces, these substances are vaporized and pass into the smoke stream, but usually the smoke stream does not become saturated with any of them, as they are present in quantities much below the saturation point for the existing temperature. Then, as the smoke stream passes into the flues and cools, the saturation point for the existing temperature of some of the substances is reached, and they begin to precipitate either as fine solid particles of fume, such as arsenious oxide and lead oxide, or as fine particles of liquid, as, for instance, sulphuric acid.

NEED OF COOLING THE GASES.

It is evident from the foregoing statement that in order to separate from the smoke stream the substances that make the true fume, it is necessary not only to decrease the velocity of the gases, but also to cool the gases to such a point that practically none of the substances can continue to exist in the vapor state. To remove both true flue dust and fume it is necessary to cool the smoke stream to a comparatively low temperature and to reduce the velocity of the stream until it is low. On considering the ordinary flue system of a smelting plant, beginning with the connecting flues from the furnaces and then considering in order the enlarged dust chamber, the flues to the chimney, and the chimney itself, it will be found that the temperature gradually diminishes from the furnaces to the chimney and that the velocity of the smoke stream is higher near the furnaces, where the gas volume is greatly increased by the high temperature, and is lowest in the dust chamber where the cross-sectional area is such as to produce a minimum linear velocity. In the flue system near the furnaces, the temperature is too high for any fume to be precipitated. In the dust chamber, where a lower temperature prevails, the flue dust, consisting of certain small particles of the ore charge and some fume, is deposited. Toward the chimney the size of the flue-dust particles decreases and true fume commences to appear, the proportion increasing as the chimney is approached. Therefore, in certain sections of the flue system, the solids that settle from the smoke stream are a mixture of true flue dust and fume. and the ratio of fume to flue dust increases toward the chimney.

In taking samples of settled solids from different parts of the flue system, it is not possible to state that one material is flue dust and another material is fume, for in nearly all samples the flue dust and the fume are mixed. A ready distinction between the flue dust and fume can, however, be made by means of chemical analysis. The flue dust, as already stated, consists of solid particles of the furnace charge which have been carried over into the flue system by the velocity of the gases. An invariable constituent of the furnace charge is silica, so that flue dust always has a high content of silica or insoluble matter. Fume, on the other hand, consisting as it does of condensed metallic vapors, is essentially free from silica, so that practically a silica limit might be established to differentiate between flue dust and fume. Material above 5 per cent silica could be termed "flue dust" and material below that "fume." This distinction is further discussed elsewhere (pp. 27 to 36).

The gases present in the smoke, chiefly nitrogen, oxygen, carbon dioxide, and sulphur dioxide, pass from the stack to the atmosphere. The difference in state between these gases and the metallic vapors existing at the high temperature near the furnace is one of degree only. The vapors can be condensed or sublimed at relatively high temperatures, but the gases can be condensed only at much lower temperatures. To condense nitrogen, oxygen, and carbon dioxide to the liquid state is, of course, out of the question, and entirely unnecessary to consider in this instance, but the condensation of sulphur-dioxide gas to a liquid merits attention, and is considered in subsequent pages.

GASES IN METALLURGICAL SMOKE.

The common gases in metallurgical smoke are nitrogen, oxygen, sulphur dioxide, carbon dioxide, sulphur trioxide, and carbon monoxide; occasionally hydrocarbon gases are present in small amounts. At high temperatures, water in the form of vapor is usually present. As the gases issue from the throats of blast, reverberatory, and roasting furnaces, they always become somewhat diluted by what may be termed "false air," and the amount of such dilution increases from the furnace throat to the stack, being dependent on the tightness of dust chambers and flues and on the intentional admission of false air to cool the gas stream or to lower the percentage of some of the deleterious constituents, such as sulphur dioxide. This dilution, of course, changes the volume composition of the gases, although it does not change the total weights of certain constituents, such as sulphur dioxide, that are emitted from the flue and stack system.

The composition of smelter gases therefore varies somewhat with the point of sampling in the flue system. The accompanying table gives the results of analyses of typical smelter gases at a western copper smelter.

TABLE 1.—*Composition of furnace gases.*

[In percentage of volume, at 0° C. and 760 mm. pressure.]

Constituent.	Part of flue system.				
	Flue from roasting furnaces.	Flue from blast furnaces.	Flue from converters.	Flue from reverberatory furnaces.	Flue at base of stack.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Sulphur dioxide	2.545	1.274	2.845	0.423	1.164
Sulphur trioxide	.275	.086	.0505	.0044	.0395
Carbon dioxide	.1136	6.493	.2084	5.242	2.748
Water vapor	2.784	3.490	1.061	3.869	2.834
Arsenic trioxide	.0073	.0091	.00073	.0156	.00465
Oxygen	14.02	10.18	12.04	10.37	11.88
Nitrogen	81.18	78.13	83.64	79.57	80.73
Carbon monoxide	None.	None.	None.	Trace.	None.
Total	100.90	99.70	99.80	99.50	99.40

The samples corresponding to these analyses were taken from the flues and represented the furnace gases, diluted to a considerable degree with false air. It will be noted that the sulphur dioxide, as well as the other constituents of the samples, varied considerably in the gases from the different furnaces.

It is unnecessary to discuss the properties of the smelter-smoke gases—carbon monoxide, carbon dioxide, nitrogen, and oxygen—as these have no particular bearing on the problem in hand. Carbon monoxide is a highly poisonous gas and of much importance in the gases issuing from the iron-blast furnace, but it is found in small amounts only in gases from furnaces such as are under discussion, with the exception of the lead-blast furnace, the gases from which may contain as much as 10 per cent of carbon monoxide.

SULPHUR DIOXIDE.

CHARACTERISTICS.

The gases of most importance in the smoke stream are sulphur dioxide and sulphur trioxide. Sulphur dioxide is an irritant gas with the powerful odor characteristic of burning sulphur. If pure sulphur dioxide gas not admixed with other gases be cooled to —10° C. at ordinary atmospheric pressure (760 mm. of mercury, or 14.5 pounds per square inch), it condenses to a mobile liquid. If the temperature be further lowered to —76.1°. C. the liquid freezes to a white snowlike mass. At any temperature between these two, gaseous sulphur dioxide will still exist above the liquid dioxide, the amount of the gaseous dioxide being dependent on the temperature. Even below —76.1° C. some gaseous sulphur dioxide will exist, as the solid dioxide has a vapor tension, although small.

CONDITIONS GOVERNING CONDENSATION.

If smelter smoke containing 2.5 per cent of sulphur dioxide by volume be considered at ordinary atmospheric pressure, the partial pressure, or that part of the whole atmospheric pressure due to the 2.5 per cent of sulphur dioxide, will be equivalent to 19 mm. of mercury. Before any of the sulphur dioxide can be condensed to the liquid form the smelter smoke will have to be cooled to that temperature at which the vapor tension for saturation is less than 19 mm. of mercury. At —30° C. (the lowest temperature for which data are at hand) the saturation vapor tension of sulphur dioxide is still 273.6 mm. of mercury. This corresponds to 36 per cent by volume of sulphur dioxide in the smoke stream. Therefore, even if the smoke stream considered at atmospheric pressure were cooled to —30° C., it would have to contain at least 36 per cent of sulphur dioxide before any condensation of the dioxide could begin. It is hence evident that mere cooling of smelter smoke, even to very low temperatures, will be without effect on the sulphur dioxide content of smelter smoke. However, if pressure be applied to the smoke at the same time that it is cooled, condensation of the sulphur dioxide can be brought about. As already stated, for —30° C. the saturation vapor tension of sulphur dioxide is 273.6 mm. of mercury. Hence smoke containing 2.5 per cent of sulphur dioxide, which corresponds to a partial pressure of 19 mm. would have to be compressed 14.4 atmospheres ($\frac{273.6}{19}$) before the sulphur dioxide would begin to condense into liquid particles. The cooling of enormous volumes of smelter smoke to low temperatures and compression to a high degree would be so costly as to make this suggested method of eliminating sulphur dioxide from smoke entirely impracticable.

Another method proposed is that of refrigerating the gases and freezing out sulphur dioxide in connection with the water vapor. There are possibilities in this process, and it is again mentioned in a subsequent section.

EFFECT OF SULPHUR DIOXIDE ON MEN AND ON VEGETATION.

The effect of sulphur dioxide on the human system has already been briefly mentioned. Seventy parts per million make air containing it unbreathable. Some authorities hold that it is distinctly harmful, others that although its effect is highly disagreeable, and of course not beneficial, the harm done by it is not serious. It is always present in readily noticeable quantity in the air about copper-smelting and roasting furnaces; in fact, to such an extent, that the ordinary visitor to these plants finds the atmosphere practically unbearable. Furnace men, however, work in this atmosphere continuously and do not seem to suffer permanent harm from their contact with the gas. On the other hand, the effect of the gas on vegetation is seriously harmful. The proportion that does damage has been mentioned, as also the fact that different types of plants and even different plants of the same species are not affected alike.

A summary of the action of sulphur dioxide on plants may be stated as follows: The action shows itself by spots on the leaves, the whole leaf sometimes becoming seared and yellow. This drying of leaves and twigs and the falling behind in yearly growth give the general appearance of a dying plant. The severity of the symptoms varies with the proportion of sulphur dioxide in the atmosphere. The sulphur dioxide causes a derangement of the water circulation, leading to loss of water, thus causing the drying of the leaves. Practically all damage to plants by the gas is by action on the leaves and bark, but chiefly on the leaves. The gas is absorbed by the leaf surface as a whole, probably acting on the chlorophyll in the leaf, sulphates eventually being formed. There is no direct action either by sulphur dioxide or sulphur trioxide on the roots of plants, and hence no indirect action through the soil. It is generally conceded that the soil is not injured by sulphur dioxide or sulphur trioxide either acting directly or dissolved in the rain. The action of sulphur dioxide on the life of the plant is intensified by light and high temperature, and by moisture in the air. Plants in damp climates or climates in which fogs occur are much more subject to damage than those in dry climates. This statement is somewhat in dispute, but from the writer's own experience, he is inclined to believe that the damage is greater in the presence of moisture than when moisture is practically absent.

Damage to vegetation can be shown by the increased sulphate content over the normal amount usually present. In this connection, however, it is necessary to take into consideration the sulphate content of the soil, as the percentage of sulphate in the leaves depends upon the nature of the soil. In general, conifers, or plants whose leaves are resinous are most easily injured. Certain species of pine, such as the white pine, may be injured by as little as one part of sulphur dioxide per million parts of air if this amount be constantly present. This tree may be severely injured or even killed when other trees show no injury whatever. The red fir is also easily susceptible to damage. Post oak, white oak, and the red maple are comparatively resistant to injury by sulphur dioxide. Junipers, although belonging to the conifer family, resist injury well. Black and Spanish oak, yellow pine, and lodge-pole pine are rather easily injured.

PROPORTION OF SULPHUR DIOXIDE IN AIR NEAR SMELTERS.

The proportion of sulphur dioxide gas in the atmosphere in the vicinity of smelters naturally depends upon the ore capacity of the plants and the amount of sulphur in the ores. A correct measure is the volume of smoke discharged by the stack and the sulphur dioxide content of this smoke. Data for one large plant have already been given. As damage to vegetation is done when the sulphur dioxide reaches a certain concentration in the atmosphere, it would appear that it is necessary under present conditions, with no practical process available for the removal of this gas from the smoke, to have this so diluted when it issues from the stack that the concentration necessary for damage is not attained. If the smoke as it issues from the top of the stack would rapidly and uniformly diffuse itself throughout the atmosphere, large volumes of smoke containing 1.5 to 2 per cent of sulphur dioxide might possibly be discharged with little damage to vegetation. Unfortunately the diffusion of smoke does not take place rapidly. The smoke, as it issues from the stack, has a rapid vortex motion, which seems to hold it together to some extent in masses after it leaves the stack. These smoke masses, which are rendered visible by the fume, are of the same general composition as the original smoke stream, although considerably diluted. It is a matter of common observation that smelter smoke issuing from a high stack will trail off for miles before it finally reaches the ground.

Some of the gases in the smoke stream unquestionably are diffused much more widely than is indicated by the visible fume of the stream. On the whole, however, the contents of the smoke stream tend to hang together in masses, and when these masses strike vegetation the damage is done. That this statement is true is attested by experience gained by the use of high chimneys. It is fairly well established that when high chimneys are used for discharging the smoke into the upper regions of the atmosphere in order to attain a more complete diffusion of the gases before they reach the earth damage still occurs, although distributed over a wider area and not so pronounced. This indicates the difficulty of a thorough diffusion or mixing of the gases with air.

Efforts have been made to obtain definite figures as to the sulphur dioxide content of the atmosphere in the vicinity of smelters. Data of this sort must always be considered with caution, for the results obtained depend largely on the location of the place where the sample is taken in reference to the stack and to the prevailing wind at the time. For, as the diffusion is so irregular, the place sampled might indicate at one time only traces of sulphur dioxide, whereas a few hours later the dioxide might be present in damaging proportions.

SULPHUR TRIOXIDE.

CHARACTERISTICS AND ORIGIN.

Sulphur trioxide is an oxide of sulphur of a higher degree of oxidation than sulphur dioxide and is the basis of the common oil of vitriol or sulphuric acid of commerce. It is formed to some extent in roasting furnaces and to a much lesser extent in other furnaces. The proportion in the smoke stream is never great, but even small quantities, if they be permitted to escape into the atmosphere, are very harmful. However, only with comparative infrequency does smoke discharged from smelter stacks contain more than small proportions of sulphur trioxide in such form as to do damage. Sulphur trioxide combines molecule for molecule with water to form sulphuric acid.

In order to comprehend the behavior of sulphur trioxide in the smoke, it may be followed from its origin in the furnace. If the roasting furnace be considered as the chief source of sulphur trioxide in the smoke, its origin is due to the formation of sulphates of iron, copper, lead, or zinc during the roasting process and the subsequent decomposition of these sulphates into the oxides of the metals and sulphur trioxide. When the sulphur trioxide is liberated by the decomposition of the sulphates, for which certain temperatures are necessary, it is in the form of a true gas and is carried in the smoke stream. The smoke stream also carries water vapor, the origin of which is the moisture in the ore charges fed to the furnaces. As long as the smoke stream has a temperature above 440° C., no combination of sulphur trioxide with water vapor to form sulphuric acid is possible, but as the temperature falls in the flues as the stack is approached, sulphuric acid vapor forms by the combination of water vapor and sulphur trioxide vapor, until at about 350° C. one-half of the sulphur trioxide present in the smoke stream is in the form of sulphuric acid vapor. At ordinary atmospheric pressure (760 mm.) sulphuric acid, or rather a mixture of 98.54 per cent of sulphuric acid and 1.46 per cent of water, has a boiling point of 338° C. From this it follows that above 338° C. all sulphuric acid present must be in a vapor form or dissociated, but that below this temperature it may be present in the smoke stream in the liquid form as a fine mist or small liquid particles. As sulphuric acid, like most other liquids, has a vapor tension below its boiling point, a certain proportion of it may still exist in the vapor form below its boiling point, the exact percentage being dependent on its vapor tension for any corresponding temperature. The vapor tension of sulphuric acid is generally considered to be practically zero at about 200° C.[a] The acid that condenses at this temperature has a gravity of about 1.75, corresponding to 62° B. Therefore, the conclusion can be made that with an excess of water vapor in the smoke, or more than is necessary to form sulphuric acid with the sulphur trioxide present, all of the sulphur trioxide in the smoke will be in the form of a mist of sulphuric acid at about 200° C.

FORM OF OCCURRENCE OF SULPHUR TRIOXIDE IN SMELTER SMOKE.

The influence of temperature on the conditions of the sulphur trioxide is indicated below:

Influence of temperature on the condition of sulphur trioxide in smelter smoke.

Temperature of smoke stream.	Condition of sulphur trioxide.
°C.	
440+	Free state, uncombined with water vapor.
350-440	About one-half combined with water as sulphuric acid.
338±	Combined with water as sulphuric acid, but in vapor state.
200-338	Sulphuric acid present partly as a condensed mist and partly as vapor, the relative proportions being dependent on the total proportion of sulphuric acid in the smoke stream.
200—	Sulphuric acid all in condensed form or as a fine mist or fume.

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[a] Very little definite information has been available as to the vapor pressure of sulphuric acid, and the discussion following is subject to correction when pertinent data become available.

As already stated the relations indicated in the table apply only when there is an excess of water vapor over and above that required to form sulphuric acid with all of the sulphur trioxide present. This condition invariably occurs in ordinary smelter smoke as may be seen from Table 1. Under certain conditions, however, the amount of water vapor present may be insufficient to form sulphuric acid with all of the sulphur trioxide. Somewhat different conditions may then exist, and free sulphur trioxide in the form of a fine mist or fume may be present in the smelter smoke. Sulphur trioxide vapor condenses into a liquid at 46° C., and freezes into a solid at 14.8° C. The liquid trioxide has, of course, a considerable vapor tension, so that most of the free trioxide in smelter smoke, owing to the small amount, will be in the form of vapor until the solidification temperature of 14.8° C. has been reached.

If the sulphur trioxide is present finally in the form of sulphuric acid, the latter will be deposited in part as a mist or fume in the flues and in part will be carried out the stack with the gases as a fume, provided there is nothing present in the smoke stream to act as a base to sulphuric acid and combine with it. In most instances, however, the smoke stream, at the time the sulphuric acid forms, will still carry some fine flue dust in which there is iron oxide, zinc oxide, copper oxide, lead oxide, etc. The fine dust, in intimate contact with the fine particles of sulphuric acid, will react and cause the formation of sulphates. The sulphates deposit with the flue dust.

Many smelter gases do not carry sufficient amounts of these reacting bases in the form of fine dust or fume at the time that the sulphuric acid is ready to combine with them, and hence, under such conditions, sulphuric acid may be discharged from the stack. When the ore charge smelted or roasted contains a considerable percentage of volatile metals such as lead, zinc, and cadmium, these partly pass to the fume and, under oxidizing conditions, form oxides. The fine fume particles combine readily with the sulphuric acid formed, giving rise to sulphates and the consequent neutralization of the acid. The neutralization or the formation of sulphates probably does not readily take place until temperatures below 200° C. are reached, meaning practically that before sulphates are formed the sulphuric acid must be below its condensation point. The neutralization is probably effected by small particles of fume becoming centers of condensation for the acid. In most lead blast-furnace smelting the smelter smoke is filtered through a large number of long woolen or cotton bags, in order to recover from it all of the fine fume or solid particles and discharge only gases into the atmosphere. The smelter smoke when it reaches the bag house must fulfill two conditions: first, it must have a temperature low enough so as not to scorch the bags and injure their fabric, and, second, it must be essentially free from sulphuric acid, which rapidly corrodes the bags.

Both of these objects are accomplished by cooling the gases to about 100° F. for at that temperature no scorching can take place and the sulphuric acid will have condensed out. Neutralization in smoke from the lead blast furnace is imperfect, as the volatile metallic compounds are not oxides because the smelting is done under reducing conditions.

The gases from roasting furnaces, as already mentioned, contain more sulphur troxide than other furnace gases, and until recently it was considered impracticable to pass them through the bag house, for, as they did not contain sufficient metallic fume to neutralize the free sulphuric acid formed, their action on the bags was so destructive as to make filtering impracticable. However, by introducing into the smoke stream a comparatively large proportion of zinc oxide fume or finely powdered lime, the sulphuric acid can practically all be neutralized, if the temperature of the gases be low enough. The process is known as the "Sprague process" and is discussed elsewhere.

In order to compare the foregoing data with actual conditions, the following flue temperatures taken from practice are cited. At the copper smelter in Great Falls, Mont., the temperature at a certain point in the blast-furnace flue was 199° C. (393° F.); in the roasting flue it was 215° C. (419° F.); at the base of the stack it was 155° C. (311° F.); at the copper smelter at Anaconda, Mont., the temperature of the gases at the base of the stack in winter was 159° C. (318.2 F.). It would therefore seem that, under conditions such that a copper smelter discharges smoke, free sulphuric acid may be present, as the temperature of discharge is somewhat too high for complete neutralization to take place, even if enough metallic oxide fume be present.

FLUE DUST IN METALLURGICAL SMOKE.

COMPOSITION OF FLUE DUST.

Flue dust consists of the fine particles of furnace charge, either in their original condition or chemically changed, that have been carried into the dust chambers and flue system by the action of the furnace gases. True flue dust does not include fume. This consists of the metallic compounds that have been vaporized by the heat of the furnace and then sublimed or condensed by being subjected to cooling in the flues. From a practical standpoint, however, it is impossible to completely differentiate flue dust from fume for reasons already given. Tables 2 and 3 following give the results of incomplete analyses of flue dusts from Great Falls, Mont., and from plants near Salt Lake City, Utah.

TABLE 2.—*Results of incomplete analyses of flue dusts from smelter at Great Falls, Mont.*

Source of dust sample.	Composition.											Remarks.
	Copper content stated as metallic copper.	Silver.	Gold.	Insoluble constituents.	Silica.	Iron content stated as ferrous oxide.	Alumina.	Lime.	Total sulphur present as sulphide and sulphate.	Sulphur trioxide free and combined.	Arsenic trioxide.	
	P. ct.	*Oz.*	*Oz.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*		
Experimental flue	6.20	3.4	0.025	38.2	26.0	7.32	13.3	0.80	7.10	1.410	Some	
Blast-furnace flue	8.08	2.7	.019	34.7	26.8	33.00	7.10	1.70	16.10	......	do	
Roasting-furnace flue	10.22	3.6	.023	37.6	29.0	27.10	7.50	0.30	21.30	......	do	
Main dust chamber	8.61	3.3	.020	33.3	23.6	14.5	8.00	.70	11.80	......	do	
Base of chimney	2.21	......	......	8.4	5.4	3.8	2.8	.10	10.70	......	Considerable.	This may be considered as fume.

TABLE 3.—*Results of incomplete analyses of flue dusts from plants near Salt Lake City, Utah.*

Source of dust sample.	Moisture or water below 100° C.	Amount of sulphur trioxide present in water-soluble sulphates.	Total sulphur trioxide.	Amount of iron present in such form as to be soluble in water.	Total iron content stated as metallic iron.	Amount of copper present in such form as to be soluble in water.	Total copper content stated as metallic copper.	Insoluble SiO_2.	Lead.	Arsenic.	Zinc.	Remarks.
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
Roasting-furnace flue.	0.01	6.7	20.6	0.20	8.10	Tr.	2.20	31.4	16.3	1.31	4.40	Sample taken from lead-smelter flue near stack.
Blast-furnace flue.	.70	2.3	31.8	None.	33.0	None.	3.70	27.4	2.4	.47	.95	Sample taken from copper smelter.
Dust concretion at top of a stack.	4.10	17.7	33.9	3.6	16.7	1.6	4.20	28.5	.8	.83	.15	Do.
End of flue near stack.	2.00	15.7	26.8	3.3	15.2	1.6	5.10	21.6	6.0	9.75	.65	Sample taken from copper smelter; collected by condensation on glass tube.
Do	.70	14.2	27.3	3.6	18.3	.5	3.50	22.9	10.7	6.14	.95	Do.
Do	.30	6.1	21.0	.8	10.3	None.	1.00	18.0	19.7	13.52	4.25	Sample taken from lead-smelter roaster flue and from blast-furnace flue near stack.
Do	.90	6.9	24.4	.7	9.4	None.	.30	16.6	24.1	10.26	1.25	Sample taken from lead-smelter blast-furnace flue.
Do	3.30	17.8	29.0	4.20	9.6	.80	1.50	15.8	15.5	6.00	1.90	Sample taken from lead smelter blast furnace.

FORMATION OF SULPHATES.

The particles of ore charge that are thrown over by the action of the furnace gases, especially sulphide-ore particles, usually suffer chemical change. The particles, carried in a gas stream which sooner or later becomes somewhat oxidizing, are changed in part to sulphates, and, as may be seen from the results of analyses, the sulphuric acid content of most of the flue dusts is rather high. The sulphates found in flue dust are: Normal ferrous sulphate, $FeSO_4$; basic ferric sulphate, $Fe_2O_3.2SO_3$; normal copper sulphate, $CuSO_4$; basic copper sulphate, $2CuO.SO_3$; normal zinc sulphate, $ZnSO_4$; basic zinc sulphate, $3ZnO.2SO_3$; normal lead sulphate, $PbSO_4$; and basic lead sulphate, $6PbO.5SO_3$. It has already been stated that the sulphur trioxide content of the furnace gases is largely due to the decomposition of sulphates.

Particles of sulphide minerals, such as pyrite, pyrrhotite, chalcopyrite, and chalcocite, when subjected to high temperature in the presence of oxygen under certain conditions, particularly such as prevail in roasting furnaces, are changed in part to sulphates. It is probable that the sulphates first formed are the normal sulphates, and that with an increased temperature these are in part decomposed, with the formation of basic sulphates and sulphur trioxide. A still further increase in temperature causes the decomposition of the basic sulphates, with the formation of oxides of the metals and a further liberation of sulphur trioxide. This action is particularly in evidence in roasting furnaces. Particles of charge thus changed are carried over as flue dust.

A formation of sulphates also takes place in the flues, where, at high temperature, sulphide particles in the flue dust are subjected to the oxidizing action of the smoke stream. An examination of Table 3 shows the high sulphur trioxide content of all the flue dusts. This sulphur trioxide is divided into that which is soluble in water and that which is not soluble in water. The normal sulphates of copper, iron, and zinc, and free sulphuric acid and free sulphur trioxide are readily soluble in water, whereas the basic sulphates of these metals and the normal and basic sulphates of lead are practically insoluble in water. If a sample of flue dust be treated with water and the resultant water analyzed for sulphur trioxide, the result will be a measure of the amount of soluble sulphates, sulphuric acid, and sulphur trioxide present, and the total sulphur trioxide, less the soluble sulphur trioxide, will give the proportion that is combined as basic sulphates and lead sulphate. Most of the sulphates are hygroscopic; that is, they will absorb water and become hydrated. Some of the normal sulphates have an acid reaction, and flue dust high in sulphates has a corrosive action on vegetation.

EFFECT OF VELOCITY OF SMOKE STREAM ON COMPOSITION OF FLUE DUST.

In some of the older plants of this country the velocity of the smoke stream in the flues was frequently more than 30 feet per second, with the result that the stacks emitted a considerable quantity of true flue dust which was scattered broadcast in the vicinity of the plant, doing damage. The more modern plants do not, however, have such high velocities in their flues and little true dust is discharged from their stacks. The chemical composition of flue dust is, of course, dependent upon the nature of the ore charge smelted; that from a copper smelter will differ from that of a lead smelter. The general nature of flue dust, particularly in reference to sulphate content, is very similar at all plants. Formerly, with the high velocities of gases in dust chambers and flues, the losses from dust were considerable. At the present time the large plants make a serious effort to reduce these dust losses. Examination of Table 2 will show that the copper content of the flue-dust samples is considerable.

LOSSES FROM FLUE DUST.

Even with only small quantities of dust in the gases, as they are discharged from the stack, the losses are serious. Thus, at Great Falls, Mont., the stack discharges daily 931,204,320 cubic feet of gas (standard conditions), which carries 50,690 pounds of dust and fume. The Anaconda, Mont., plant discharges daily 1,341,000,000 cubic feet of gas (standard conditions), which carries 153,554 pounds of dust and fume. Even with only 1.5 to 2.5 per cent of copper in this dust, the loss for the year will bulk large.

IMPORTANCE OF PROPER DESIGN OF DUST CHAMBER.

The importance of so designing the dust chamber and flue system as to minimize this dust loss is therefore evident. Recently experimentation and practical experience have proven that it is possible, by adopting proper means, to cause practically all of the true flue dust to settle from the smoke stream. If the smoke stream be conducted through dust chambers of correct cross section and length, the dust will settle satisfactorily. Although formerly it was the practice to maintain a velocity of gas in dust chambers and flues as high as 30 to 40 feet per second, the speed now commonly accepted as proper for the smoke stream in dust chambers is 2½ feet per second in plain chambers and 5 feet per second in chambers hung with wire baffles (Roesing system). The length of dust chambers must be sufficient to give the dust an opportunity to settle. With a speed of 2½ feet per second all but the very finest dust settles in 50 seconds in plain chambers. Hence, if the cross sections of the dust chamber be proportioned to the volume of the gases, so as to give a speed of 2½ feet per second, the length of the dust chamber should be made

125 to 150 feet. If wire baffles are used the speed may be doubled and hence the cross section diminished by one-half of the above. These figures represent practice at two great copper plants. The speed of the gas stream in flues may be greater than this. At the Great Falls plant, mentioned above, the velocities were as follows: Blast-furnace flue, 17.3 feet per second; roasting-furnace flue, 17.5 feet per second; main dust chamber, 5 feet per second; connecting flue to chimney, 21 feet per second; chimney, 10 feet per second. At the Anaconda plant the velocity in the main flue, at a point 30 feet below the stack, was 21.43 feet per second. In general, the practice is to build flues of such cross section as not to exceed a speed of about 20 feet per second and at some convenient place to interpose in the flue a dust chamber of such cross section as to diminish the velocity of gases to about 5 feet per second, and to so design the entrance or exit of the flues into or from the dust chamber that the gas stream is forced to expand and uniformly reduce its velocity in its passage through the chamber.

LOSSES FROM FINE DUST.

Only a part of the true fume can be recovered by a reduction of the velocity of the gas stream, even with proper cooling so as to precipitate the fume. The reluctance of fume particles to settle like true dust particles is discussed more in detail elsewhere. Losses in fine dust and fume through the stack are usually expressed in terms of grains per cubic foot of smoke discharged. One pound avoirdupois contains 7,000 grains. The fine dust and fume discharged with the smoke stream is generally referred to as the solids in the smoke stream. The total quantity of solids discharged by the gas stream at Great Falls and Anaconda has already been mentioned. Expressed in grains the figures are as follows: Anaconda, 0.798 grain per cubic foot; Great Falls, 0.378 grain per cubic foot. It is necessary to state that the methods used at Anaconda and Great Falls to determine this data are radically different and that perhaps the figures are not strictly comparable. At the time the Balaklala plant at Coram, Cal., was in operation the smoke stream discharged from the stack contained 0.638 grain per cubic foot. During trials on a large scale of the electrical precipitation process, this was reduced to 0.149 grain per cubic foot.

ARSENIOUS OXIDE IN FLUE DUST.

The fine flue dust and fume that settles in the flues near the stack frequently contains a considerable proportion of arsenious oxide fume. The oxide is especially likely to occur at smelters that treat ores containing appreciable arsenic, such as the Butte (Mont.) ores. The flue dust is the source of most of the arsenious oxide or white arsenic produced in the country. The copper-smelting plants at Anaconda and at Great Falls, Mont., each produce some arsenious oxide. Arsenious oxide is also produced from fume at the smelter at Midvale, Utah, and at the Globe plant at Denver, Colo. The arsenic-bearing flue dust in copper smelters and the fume from the bag houses of lead smelters are heated in specially constructed reverberatory furnaces to volatilize the arsenious oxide, which then again sublimes in the flues. This crude arsenic product is gathered and again subjected to heating in a refining furnace, and the volatilized arsenious oxide once more sublimes in flues, yielding the refined arsenious oxide, or white arsenic, of commerce. Only part of the arsenic in the ores smelted is recovered from the flues of copper smelters. In lead smelting practically all of the arsenic is caught with the fume in the bag houses. The present demand for arsenious oxide is not great enough to warrant copper-smelting plants to seek a full recovery of the arsenic in their ores.

HANDLING OF FLUE DUST.

The flue dust that is recovered from the dust chambers and flue system of a plant must of necessity be again treated. The economic treatment of this fine material, which represents usually 2 to 10 per cent of the total material smelted, is often a serious problem. It is to be noted that this material has already passed through a number of costly operations and stands charged with costs for labor and power. The problem of a successful and economic handling of fine flue dust and other fine material is general in metallurgy, to be found in the metallurgy of iron as well as in that of copper, lead, gold, and silver. In large copper-smelting plants, reverberatory smelting furnaces afford a means of successfully resmelting flue dust in conjunction with other fine material, but in lead and iron smelting, it is necessary to charge the dust back into the blast furnace, where, of course, if it be returned in its original condition, it will again be largely blown into the flues. Within the past few years, sintering machines have been successfully applied to fine material and flue dust. These machines operate so as to sinter or incipiently fuse the small particles of the flue dust and fines into an agglomerated porous mass, which can be broken readily into pieces suitable for blast-furnace smelting. Also, flue dust may be briquetted for smelting.

METALLURGICAL FUME.

COMPOSITION AND CLASSIFICATION OF FUME.

Fume may be defined as a mixture of very fine particles of elements and metallic and nonmetallic compounds either sublimed or condensed from the vapor state. In practice, it has mixed with it comparatively small proportions of fine flue dust. Moreover, fume may consist of very small solid particles and of very small liquid particles, the latter like a mist or fog. The composition of metallurgical fume varies within wide limits, dependent upon the method of smelting employed. Broadly speaking, it can be divided into two classes—first, oxidized fume, which contains the metallic elements in the form of oxides or highly oxidized compounds, and, second, nonoxidized or raw fume, which contains the metallic elements in the form of compounds not associated with oxygen. The first class is typically represented by fume from copper blast furnaces and converters, and the second class by lead blast-furnace fume.

ANALYSES OF TYPICAL FUME SAMPLES.

In Table 4, following, are presented the results of analyses of fume from various kinds of plants in different parts of the United States.

TABLE 4.—*Results of analyses of fume from various plants.*

Constituent.	Anaconda, Mont., solids discharged from stack.	Western copper smelter, solids discharged from stack.	Mammoth, Cal., blast-furnace bag-house material.	Murray, Utah, plant.		Omaha, Nebr., plant.		Fume from a flue receiving gases from mechanical roasters and sintering machines in a lead smelting plant.[a]
				Blast-furnace bag-house material, raw.	Blast-furnace bag-house material, sintered.	Fume from converter bag-house.	Raw fume from blast-furnace bag-house.	
	1	2	3	4	5	6	7	8
	Oz. p. ton.	*Oz. p. ton.*	*Oz. p. ton.*	*Oz. p. ton.*	*Oz. p. ton.*	*Oz. p. ton.*	*Oz. p. ton.*	*Oz. p. ton.*
Silver	5.07	....	6.70	2.62	2.40	2.80	10.00	....
Gold	....	....	.01	.0149	.014	....	....	....
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Silica	4.19	2.30	1.48	2.09	2.50	....	....	....
Alumina	3.14	[b]7.08	1.68	....	....	....	....	....
Ferric oxide	2.78	[c]7.08	4.22	....	....	....	....	....
Zinc oxide	11.84	3.31	40.57	....	....	....	....	....
Antimonious oxide	1.17	1.47	.54	....	....	....	....	....
Lead oxide	11.63	.49	9.33	....	....	....	....	....
Bismuth oxide	1.15	.81	.26	....	....	....	....	....
Cupric oxide	1.05	....	.90	....	....	....	....	....
Arsenious oxide	34.34	38.10	10.02	....	....	....	....	....
Tellurous oxide	.35	....	....	....	....	....	....	....
Sulphur	[d]1.88	....	[d].68	[e]5.20	[e]4.20	....	[e]5.40	[e]25.50
Sulphur trioxide	[f]19.65 [g]3.65	[h]16.67	[e]23.84	....	....	[f]21.63	....	....
Lime	.50	.18	.91	....	....	....	....	....
Magnesia	.69	....	.10	....	....	....	....	....
Soda	.49	....	.88	....	....	....	....	....
Potassa	.42	....	.38	....	....	....	....	....
Carbon	.80	....	....	....	....	....	Considerable.	10.03
Copper	....	.70	....	.058	.05	....	Trace.	.30
Sulphuric acid	....	22.23	....	....	....	....	....	....
Water	....	[i]4.45	[j]3.80	....	....	....	....	9.70
Phosphorus pentoxide	....	....	.15	....	....	....	....	....
Selenium	....	....	.05	....	....	.24	....	....
Oxygen	....	....	.34	....	....	....	....	16.4
Iron	....	....	....	5.10	5.20	....	....	5.03
Zinc	....	....	....	3.20	4.20	1.90	3.00	1.50
Arsenic	....	....	....	36.60	33.50	2.00	14.20	2.67
Lead	....	....	....	28.42	27.30	66.00	52.50	22.50
Antimony	....	....	....	....	....	.20	....	.87
Tellurium	....	....	....	....	....	.20	Trace.	....
Insoluble constituents	....	....	....	....	....	....	....	5.50

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[a] From other data as regards the manner in which sulphur was present the probable actual composition of this fume was as follows: Insoluble constituents, 5.5 per cent; zinc sulphide, 0.68 per cent; iron sulphide, 3.93 per cent; elemental sulphur, 13.82 per cent; copper sulphate, 0.75 per cent; ferrous sulphate, 6.90 per cent; zinc sulphate, 2.60 per cent; lead sulphate, 31.24 per cent; lead sulphide, 1.34 per cent; sulphuric acid, 8.98 per cent; arsenic sulphide (As_2S_3), 3.81 per cent; antimony sulphide (Sb_2S_3), 1.22 per cent; water, 9.70 per cent; remainder, carbon and some oxygen in basic sulphates. The first three constituents were probably present as fine "flue dust." Silver, 5.40 ounces per ton; gold, 0.015 ounce per ton.

[b] Includes ferric oxide.

[c] Includes alumina.

[d] Free and as sulphide.

[e] Total.

[f] Combined.

[g] Free.

[h] As sulphate.

[i] At 100° C.

[j] At 250° C.

In the table, analyses 1, 2, 3, and 6 represent the first class and 4 and 7 the second class. In the copper blast furnace and converter the atmosphere is usually oxidizing and the oxide compounds of the volatile metals are naturally produced. On the other hand, in the lead blast furnace the conditions are essentially reducing, and volatile nonoxidized compounds are produced. The fume from lead blast furnaces may readily be ignited and will smolder and burn, the unoxidized compounds becoming converted into oxidized compounds. In Table 4, analysis 4 represents raw or unburned fume and analysis 5 the same fume burned and sintered. The constituents and figures given in Table 4 do not necessarily represent the true composition of the fume. Thus arsenic is given in the table both in the form of elemental arsenic and of arsenious oxide. Sulphur is given as free sulphur, as combined in the form of sulphide, as combined in the form of sulphuric acid, as free sulphur trioxide, and as sulphur trioxide combined with a base. The manner in which the substances are stated depends largely on the chemist that makes the analysis. No attempt has been made in the table (except in analysis 1 of fume from Anaconda) to combine the constituents into the compounds such as they actually exist in the fume.

SUBSTANCES FOUND IN FUME.

The following substances may be found in the type of fume under discussion: Metallic gold and metallic silver; quartz and silicates of potassa and soda; silicates of alumina, lime, and ferrous oxide, probably carried into the fume as fine-blown slag and ore particles; zinc oxide; normal zinc sulphate; basic zinc sulphate; cadmium oxide; antimonious oxide; antimonious sulphide; lead oxide; lead sulphide; lead sulphate; cupric oxide; cuprous oxide; elemental arsenic; arsenious sulphide; arsenious oxide; elemental tellurium; tellurous oxide; elemental sulphur; sulphur trioxide; free sulphuric acid; elemental selenium; selenium oxide; thallium oxide; soda and potassa.

Which of the above compounds occur in fume depends upon the condition under which the fume is produced. Thus, in copper smelting, any lead in the ore is probably vaporized as lead oxide and is found in the fume as such or in the form of lead sulphate if considerable sulphuric acid be present in the smoke stream. Zinc under similar conditions is in part vaporized as zinc oxide from the combustion of zinc sulphide mineral and is found as such in the fume or as zinc sulphate. Usually, the bases, zinc and lead oxide, neutralize most of the sulphuric acid which is formed in the fume in the manner previously described. The oxides of zinc and lead are carried as extremely minute solid particles in the smoke stream after a certain degree of cooling and probably act as centers of condensation for the sulphuric acid vapor, thus causing the formation of the sulphates of zinc and lead.

FACTORS DETERMINING STATE OF OCCURRENCE OF METALLIC PARTICLES IN FUME.

Whether a metallic compound exists in the smoke stream as a vapor or is sublimed or condensed depends, of course, upon the temperature and the saturation vapor tension of the compound for that temperature, as is more definitely explained for the compound arsenious oxide, one of the most important constituents of fume. Figure 1 gives the vapor-tension curve of this compound, plotted on the basis of amount of arsenious oxide per cubic foot of smoke stream, and the temperature. It will be noted that at a temperature of 833° F. a cubic foot of the smoke stream can hold 0.325 pound of arsenious oxide in the vapor condition. At 700° F. a cubic foot can hold only 0.05 pound. Therefore, if a cubic foot of smoke saturated with oxide be cooled from 833° to 700° F. 0.275 pound of arsenious oxide will be condensed as a fume. If the smoke be not saturated with arsenious oxide vapor for a given temperature, in other words, if less arsenious oxide than the smoke can possibly hold be present, the smoke will have to be cooled to a temperature such that the proportion of arsenious oxide present represents saturation, before any sublimation of fume takes place. From the curve it may be determined that when the smoke has been cooled to 250° F. a trace only of the arsenious oxide is in the vapor form, practically all of the oxide has sublimed and is in the form of very fine solid particles.[a]

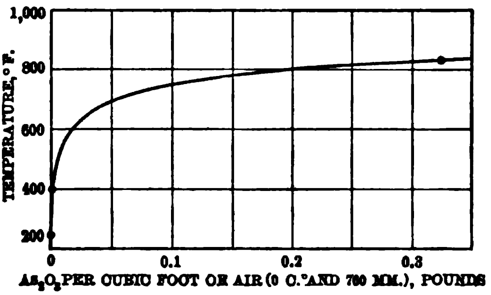

FIGURE 1.—Arsenic trioxide saturation curve.

What has been stated for arsenious oxides applies in a general way to all of the compounds in the fume, except that the saturation pressures and temperatures vary for each and every compound. The temperatures at which sublimation or condensation is practically complete for the compounds named as common in fume are higher than that for arsenious oxide. Hence no metallic compounds as vapor pass up the stack with the smoke stream, but such as are lost are in the form of sublimed or condensed particles. Evidently the cooling of the smoke stream, in order to permit sublimation and condensation, is very important. The cooling is accomplished in some plants by the admission of cold air to the smoke stream, and in others, as, for instance, in certain lead-smelting plants, by passing the gases through a long flue system, part of which consists of sheet-iron or steel flues, the cooling effect of which on account of their high heat conductivity is pronounced.

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[a] For a discussion of the vapor pressure of arsenic trioxide, see Welch, H. V., and Duschak, L. H., The vapor pressure of arsenic trioxide: Tech. Paper 81, Bureau of Mines.

As already stated, fume particles are in an extremely minute state of division and do not settle from the gas stream by gravity as ordinary flue-dust particles do. Even though the gas stream be cooled so that complete sublimation or condensation can take place and the velocity of the stream be reduced to a low rate, little settling of fume occurs. In fact, surfaces somewhat heated, particularly if they be moist, seem to have a decidedly repellant action on the fume particles. This peculiar action of fine particles near heated surfaces is well known. It is probably due to their bombardment by gas molecules repelled from the heated surfaces. A similar force operates the well-known device known as Crooke's radiometer. The behavior of minute particles in a fluid, gas, or liquid has been investigated by Stokes. With a given viscosity of the fluid, particles of a certain minuteness do not settle by gravity, the frictional forces being sufficient to keep the particles in suspension. In the settlement of very fine colloid particles (slimes) in liquids, similar conditions prevail and settlement can not be effected. It would seem that the laws governing these very fine solid particles in fluids are still little known and afford an interesting field for research.

AN UNSATISFACTORY METHOD OF REMOVING FUME.

One of the methods that has been advocated for many years to remove fume from smelter smoke is the spraying of the gas stream with water, accomplished by forcing the gas stream to rise against water sprays in towers or by adding to the gas stream, while still hot, large quantities of steam, which later condenses, or by passing the smoke stream through centrifugal gas washers, such as the Theisen washer, with a view to thus removing the fume. The method removes flue dust with fair success, but is not satisfactory for fume, so that the statement may be safely made that the application of water to the smoke stream in various ways for the removal of fume holds out little promise. Fume can be readily removed by filtering the smoke stream through fabrics or by electrically precipitating the particles. These methods for the removal of fume from the smoke stream are discussed in following pages.

DAMAGE CAUSED BY FUME.

The escape of fume from the stacks causes not only a loss of valuable material under certain conditions, but is also occasionally the cause of considerable damage. Damage by smelter smoke, as already outlined, may be caused by two constituents of the smoke, first, the sulphur dioxide, and, second, the fume and flue dust. The damage by the solids is done when they are of a poisonous nature, as for instance, arsenious oxide and lead compounds, or when they consist of acid sulphates or particles to which adhere sulphur trioxide and sulphuric acid. When the smoke stream contains considerable arsenious oxide, vegetation in the vicinity of the smelters may become contaminated with this poison, under certain climatic conditions, particularly when little rain falls, so that the oxide concentrates on plants and leads to the poisoning of stock. High stacks, which tend to scatter this poisonous fume over large areas and thus prevent any concentration of the arsenious oxide in sufficient quantity to be really harmful, usually solve the difficulty experienced from this source.

A number of plants recover considerable arsenious oxide as a by-product, and could recover a still larger percentage of that in the ore if the market warranted such recovery. However, the demand for this substance is somewhat limited, so that some of it is permitted to escape. The damage to country surrounding smelters from the solid constituents of the smelter smoke is now not as great as formerly, since the installation of adequate flue-dust settling facilities and high stacks. Formerly, with low stacks and inadequate settling facilities and high velocities of smoke stream in flues, the damage due to solids was probably considerable. The corrosive effect on vegetation of flue-dust particles and fume charged with sulphuric acid, metallic sulphates, and sulphur trioxide is very pronounced, and when much of this material is discharged from stacks, as will happen in connection with poorly designed flue systems, the damage to vegetation may be serious. It is to be noted here that damage by sulphur dioxide gas is not included in this discussion and is something entirely apart.

It is an interesting fact that when the discharge from smelter stacks is made invisible, or nearly so, by the removal of fume, either by filtering the discharge through bags or by electrical precipitation, little complaint is heard from the surrounding country, even when the fume itself is practically harmless, and when real damage may still be done by the sulphur dioxide gas, which, of course, is invisible. The writer knows of one plant that discharges a nonvisible smoke stream containing considerable sulphur dioxide which occasionally in damp weather causes minor damage. This damage is not particularly noticeable, but at times when a bag house is not in working order for a brief period and the smoke becomes visible, owing to unfiltered fume, complaints arise about damage done, although the occurrence of any real damage is questionable. From a psychological standpoint it seems to be desirable to make the smoke stream invisible by the complete removal of fume.

RECOVERY OF FLUE DUST AND FUME AND METHODS PROPOSED FOR RENDERING SMOKE HARMLESS.

RECOVERY OF FLUE DUST.

Within the past five or six years the realization has come to smelter managers that in many plants a considerable loss of valuable material may take place in the fine flue dust discharged with the smoke stream. This question has already been discussed in some detail in the preceding pages, and the intention here is to outline the actual methods adopted to overcome the loss. Officials of two large copper-smelting plants in this country have investigated this question thoroughly and have acted on their investigations by constructing suitable devices to overcome losses. One is the Anaconda company's smelter at Great Falls, Mont., and the other is the smelting plant of the Copper Queen mine, in Arizona.

SAVING AT ONE PLANT USING IMPROVED FLUE SYSTEM.

It was demonstrated at the Great Falls plant that with the old flue system in use there was a daily loss in the dust of 3,775 pounds of copper, 106 ounces of silver, and 0.71 ounce of gold. By using the new flue system, which makes adequate provision for dust recovery, the losses have been reduced to 347 pounds of copper, 28.5 ounces of silver, and 0.2 ounce of gold. The copper loss amounts to 0.19 per cent of the copper charged into the furnaces. The saving thus effected is $372.18 a day, or $130,263 a year. The cost of the new flue system was $1,100,000, so that the saving made much more than pays the interest charges on the investment.

LOSSES FROM FLUE DUST CONSIDERABLE.

At the plant of the Copper Queen mine it was discovered that there was considerable loss through dust passing out the stack. The daily copper loss due to this cause was estimated to be about 6,000 to 18,000 pounds, dependent on the number of furnaces in operation. Reference to Table 2, giving flue-dust composition at some Utah plants, will show that certain of these dusts were taken at the base of the chimney and that their silica content is high. This would indicate a large percentage of true flue dust in the smoke stream, and, of course, a considerable discharge of dust into the atmosphere.

EFFECT OF VELOCITY OF SMOKE STREAM ON DUST DISCHARGE.

There is no question but that some plants in this country were formerly, and are even at the present time, suffering considerable loss from fine dust being discharged with the smoke stream. The work at the Great Falls and the Copper Queen plants has shown conclusively that dust losses may be largely avoided by a reduction in the velocity of the gas stream to about 5 feet per second. The reduction is accomplished by passing the smoke stream into large dust chambers hung with a myriad of wires in such a way that there is no direct channel through the dust chamber, there being at least 10 obstructions to any passage. The use of wire baffles for the purpose of settling out dust from smoke is known as the Roesing system.

Formerly it was accepted as a fact that the passing of the smoke stream through a long system of flues of such cross section that the velocity did not exceed 20 to 30 feet per second would cause the dust to settle. It was considered that length was essential. In the more modern construction it is found that if dust is to be settled the velocity must be reduced to not more than 2½ feet per second in clear chambers or to 5 feet per second in chambers hung with wires.

DESCRIPTION OF PROPERLY CONSTRUCTED FLUE SYSTEM.

A modern flue system, therefore, may be considered to consist of (1) the individual flues leading from the different furnaces and of such cross section that the gas velocity shall not exceed 20 to 25 feet per second, the flues discharging in a proper manner into (2) a large chamber known as the dust chamber 250 to 350 feet long and of such cross section as to reduce the gas velocity to 2½ or 5 feet per second, dependent upon whether wire baffles are used, and (3) a flue connecting the dust chamber with the chimney and of such cross section that the gases have a velocity not exceeding 20 or 25 feet per second, and (4) a chimney of sufficient cross section to carry the gas stream at this velocity and of sufficient height to give ample natural draft at the furnaces.

FLUE SYSTEM AT COPPER-SMELTING PLANT AT GREAT FALLS, MONT.

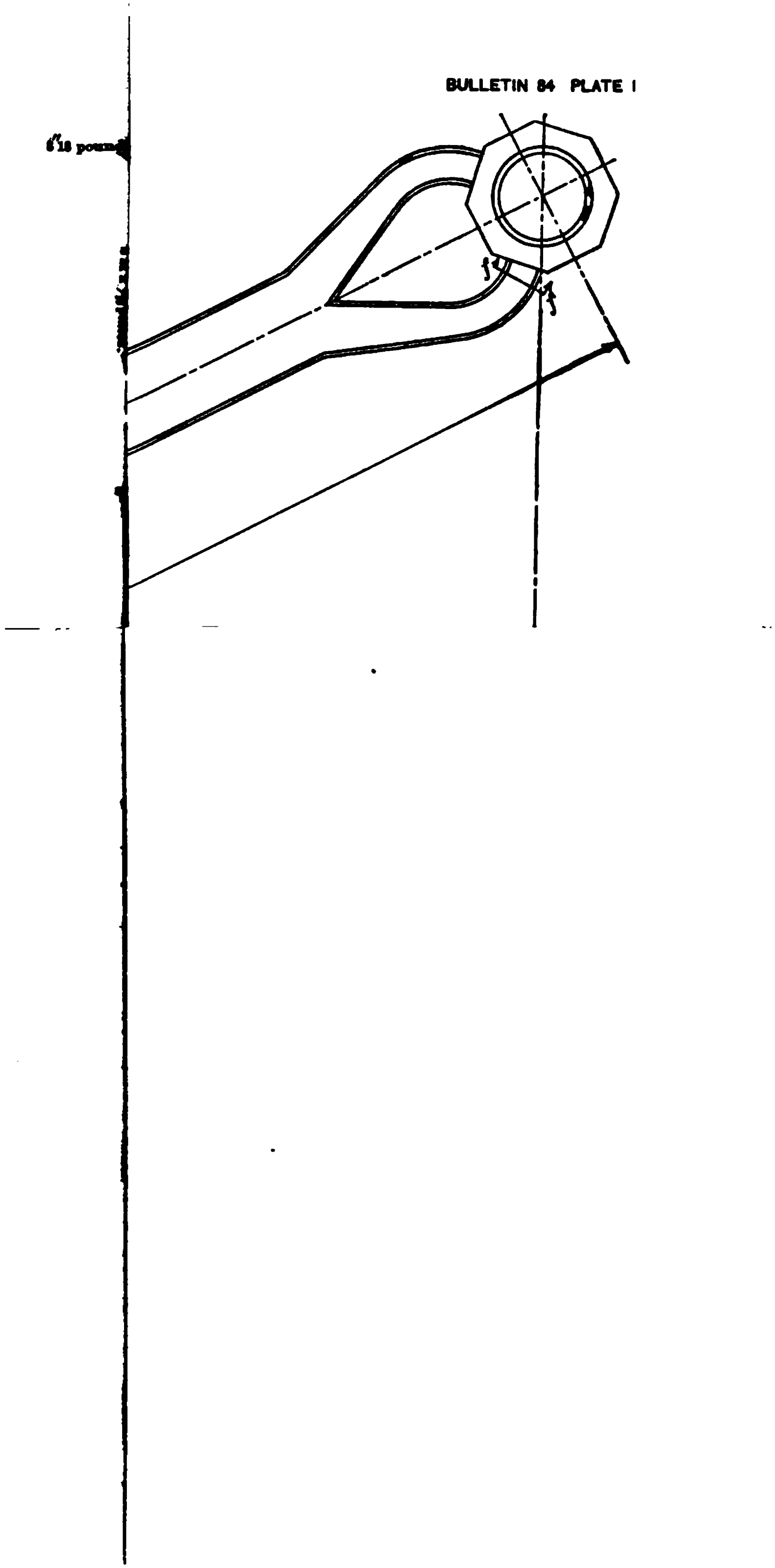
In Plate I is shown the new flue system at Great Falls. It consists of a cross-take flue that discharges into the dust chamber, which connects to the chimney by means of a long connecting flue. Into the cross take and the dust chamber discharge the individual flues from the blast furnaces, reverberatory furnaces, roasting furnaces, and converters. The arrangement of these individual flues is temporary only. They will discharge into the main system somewhat differently after the furnaces have been reconstructed. But the system, from the cross-take flue on, may be considered as a typical modern flue system.

The most interesting part of this is the dust chamber, which is a rectangular chamber 367 feet long, 177 feet wide, and 21 feet high from the top of the hoppers to the roof. It is constructed of brick and steel in the manner shown in Plate II, which depicts a transverse section and a longitudinal section. These views represent sections near the upper end of the chamber, where it joins the connecting flue, in order to show the special features of construction, such as the wire-shaking arrangements *z* and the cold-air pipes *q*. The wire-shaking arrangements provide for the shaking of the wires from the outside of the dust chamber to dislodge dust that has accumulated on the wires. The air pipes *q*, which enter the dust chamber from the top and from the bottom, admit cold air in order to decrease the temperature of the smoke stream and cause the sublimation of arsenious oxide, etc. The dust chamber is set on a large number of steel columns, and the bottom of the chamber is formed by a great number of hoppers (Pl. III, A), from which the collected flue dust may be discharged into cars. Plate IV shows the hopper plan of the bottom of the dust chamber. In all there are 1,042 hoppers. The chamber is divided longitudinally by a brick wall, shown in Plate II and as a heavy line in Plate IV.

USE OF STEEL WIRES.

The area within the dotted lines (Pl. IV) is hung with steel wires, spaced 2.3 inches, center to center, there being a total of about 1,200,000 wires, each of which weighs about 1 pound. The wires are hung in two groups, with a clear space of 47 feet between the two groups near the middle of the chamber. Plate IV indicates that a clear space is left at the inlet end, affording an unobstructed passage for the gases to distribute themselves equally over the full width of the chamber. The total average length of the chamber filled with wires is approximately 317 feet. There are two sizes of wire used in the first group. For a distance of about 51 feet from the inlet end wires 16 feet long and 0.162 inch in diameter are used, the rest are 20 feet long and 0.135 inch in diameter. The second division is hung with wires 20 feet long and 0.135 inch in diameter. At the upper end of the first division of wires there are 22 pipes for the admission of air in case it should be desirable to lower the temperature of the gases. An interior view of the dust chamber showing these air pipes, the wires, and the top of the hoppers is shown in Plate III, *B*. The wires are suspended from crimped iron screen cloth of 1.625-inch mesh stretched along the bottom of the roof purlins. Flat bars, 2½ inches by $\frac{5}{16}$ inch, bolted to the purlins hold the screen in place. The baffle wires are provided with a hook in the form of a shepherd's crook on one end and are suspended from alternate intersecting points on the screen. The wires are shaken for a period of about 30 minutes at intervals of 60 to 90 days, as may be required.

BULLETIN 84 PLATE I



USE OF BUTTERFLY DAMPERS.

At one end of the chamber a line of butterfly dampers (*x*, Pl. II) is provided, so that either side or the whole of the chamber may be closed to the passage of gases. The dampers are made of heavy cast-iron plates which are carried by a vertical shaft. The step bearing at the bottom is below the floor plate, where it is free from dust and can be oiled and inspected. The upper ends of the shafts pass out through the roof of the dust chamber, each one being equipped with a worm and hand wheel for operating the damper. While the shaking device of one side of the chamber is in operation, the dampers on this side are closed and all the gases pass to the opposite side of the longitudinal division wall.

RESULTS OBTAINABLE WITH SYSTEM.

Figures illustrating the results obtainable with the new flue system are as follows:

	Tons.
Amount of copper-bearing material treated during 41 months	1,478,026
Flue dust recovered from:	
Blast-furnace flue	87,020
Roaster-furnace flue	18,741
Uptake and cross-take flue	17,360
Main dust chamber	68,048
Connecting flue	4,000
Total	191,169
Monthly average of flue dust recovered	4,662
Proportion of flue dust recovered, per cent of the material treated	12.93

During the operation of the old flue system there was treated, in 68 months, 2,877,388 tons of copper-bearing material, from which 104,239 tons of flue dust was recovered, or 3.62 per cent. It will be noted that the percentage of flue dust recovered is much greater under the new system with its accompanying improvements than under the old, the difference practically representing what was formerly lost.

In the new system all flues, including the dust chamber, are raised above ground and have hopper discharges, so that it is an easy matter to remove flue dust from the flue system. In many of the older plants at least part of the flues are not provided with this convenience, and the flue dust accumulates in time to such an extent that it is necessary to shut down the plant and clean out the flue system. The continuous accumulation of dust in this kind of flue progressively diminishes the area and necessarily increases the gas velocity, causing increased dust losses. At Great Falls this difficulty is eliminated.

MANNER OF DEPOSITION OF FLUE DUST.

Figure 2 indicates the manner in which flue dust is deposited in the large dust chamber at Great Falls. It will be noted that a great part of the dust is deposited near the entrance of the chamber, over 50 per cent of it being thrown down in the space from the beginning of the chamber to the tenth row of hoppers. The effect of the lack of wires in the center of the chamber is plainly shown by the drop in the quantity of dust deposited.

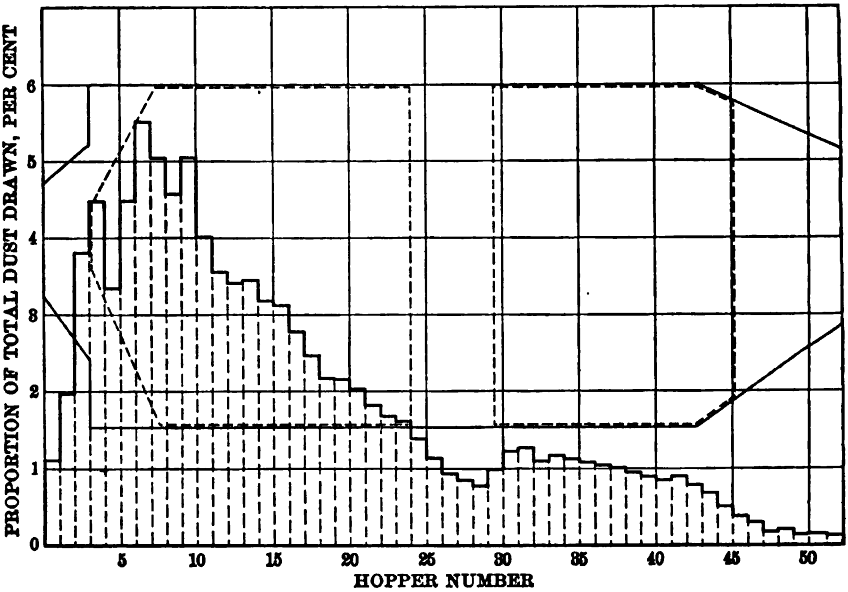
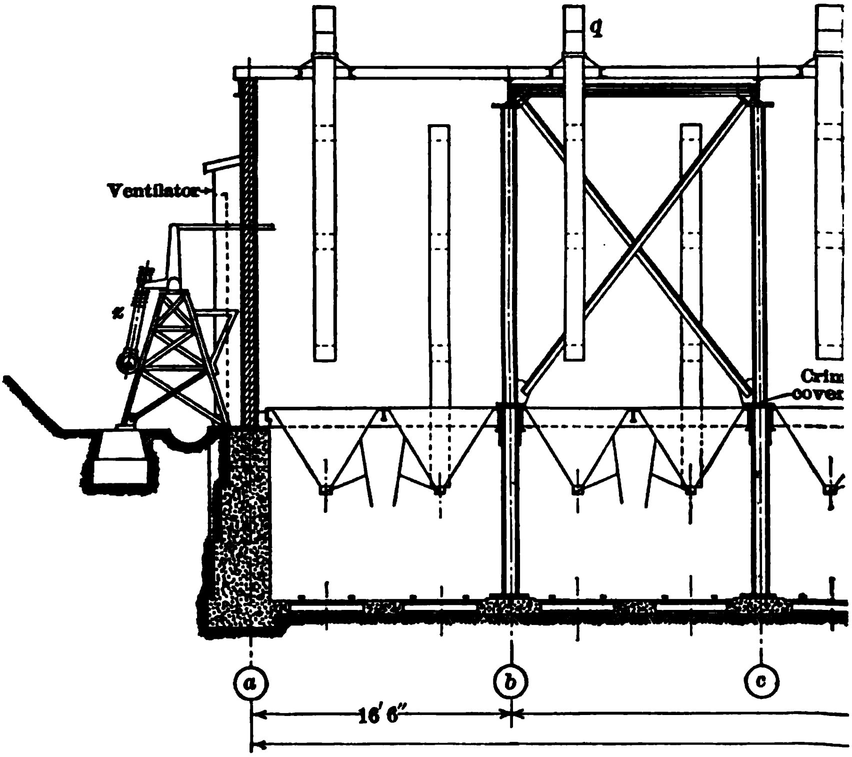


FIGURE 2.—Manner of deposition of flue dust in dust chamber (the dotted lines indicate the space filled with wires).

USE OF WIRE-HUNG DUST CHAMBERS IN OTHER SMELTING PLANTS.

At the Copper Queen smelter in Arizona a number of smaller dust chambers are in use. That connected with the reverberatory furnaces is 1,200 feet in cross section and 132 feet long and contains 30,000 wires That connected with the roasting furnaces is 1,370 feet in cross section, and 144 feet long, and contains 42,000 wires. The proportion of flue dust recovered in the chamber last mentioned is about 6 per cent of the ore roasted.

At the smelting plant at Tooele, Utah, a wire-hung dust chamber was used in connection with the roasting furnaces before an explosion in 1913, and experiments with this system are being conducted at the Tacoma plant of the same company.



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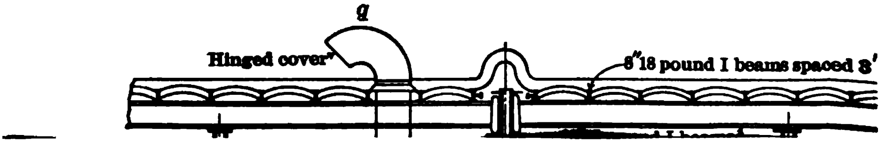


TREATMENT OF FLUE DUST.

The usual manner of treating flue dust is to charge it as it comes from the flues and dust chambers into the reverberatory smelting furnaces as part of the charge of that furnace. Plants provided with reverberatory furnaces, as most of the copper-smelting plants now are, experience little difficulty in the treatment of flue dust. But when no reverberatory furnaces are at hand, that is, when their use for certain reasons is not warranted, the treatment of the flue dust becomes a serious problem. This problem is encountered in some copper-smelting plants and in all lead-smelting plants. Recently the Dwight-Lloyd sintering machines for the agglomeration of fine sulphide-bearing materials have come into wide use, and at a number of plants they successfully treat mixtures of fine ores and flue dust and prepare them for blast-furnace smelting.

RECOVERY OF ARSENIC.

The material recovered from the flue system under the name of flue dust is not necessarily true flue dust only, but consists usually of a mixture of flue dust and fume. At Anaconda and Great Falls the flue dust that accumulates in the upper part of the flue system contains considerable arsenic. It has been mentioned that at Anaconda and Great Falls this flue dust is re-treated for the recovery of arsenious oxide. By no means can all of the arsenic in flue dust be recovered as arsenious oxide. A certain proportion of this substance is fixed in the flue dust as nonvolatile arsenates. The recovery is usually 50 to 70 per cent. It is stated that at Anaconda the total quantity of arsenious oxide per year in the ores smelted is 15,000 tons. In 1912 this plant produced about 357 tons of arsenious oxide or white arsenic. The total consumption of white arsenic in the United States in 1912 was 6,030 tons, about one-half of which was imported, and the other half is a by-product smelter production. Hence it is useless to seek the complete recovery of the arsenic oxide. Butte ores contain on an average 0.5 to 0.6 per cent of arsenic, expressed as arsenious oxide. This is eliminated in four ways, as follows: (1) In concentrating the ores; (2) as fume carried by the smoke streams discharged by the stack; (3) as arsenious oxide recovered in the arsenic plant; (4) as arsenic eliminated in the blast-furnace and reverberatory slags. The blast-furnace slag at Anaconda carries about 0.24 per cent, and the reverberatory slag about 0.30 per cent arsenic, expressed as arsenious oxide. Probably about 20 per cent of the arsenic in the ores is eliminated in the slag.

USE OF BAG HOUSES IN LEAD-SMELTING PLANTS.

Many lead-smelting plants are provided with a bag house for the recovery of fume and fine flue dust, so that the importance of the proper construction of the dust chamber for the recovery of flue dust is perhaps not so apparent, as fine dust will be readily caught in the bag house. However, a number of lead-smelting plants in this country are not provided with bag houses, and others having bag houses discharge the gases from the roasting furnaces and pots and sintering machines directly into the atmosphere. Hence nearly all lead-smelting plants will profit by installing flue systems of modern design.

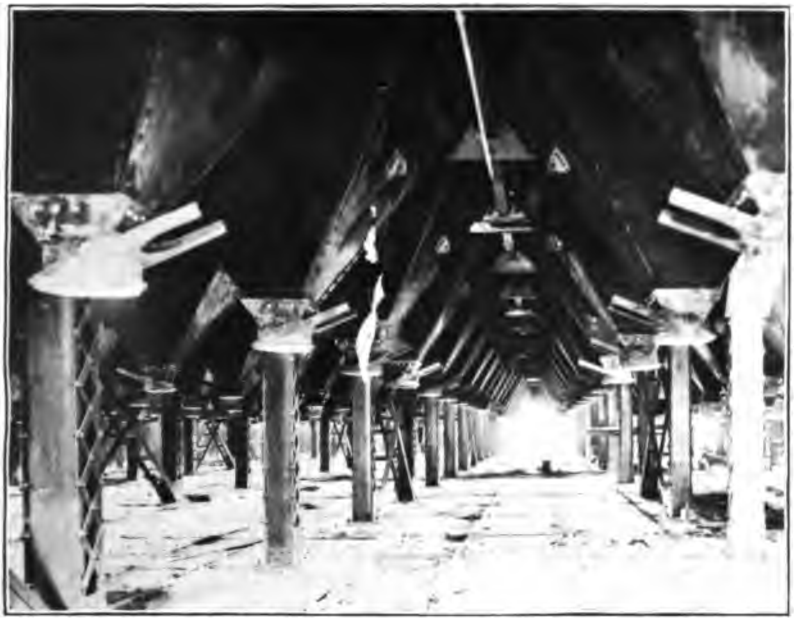
RECOVERY OF FUME.

The nature of fume has been fully described. There are at present two methods employed for the recovery of fume—filtering the smoke stream through woolen or cotton bags and electrically precipitating it by the electrical precipitation process. The first has a rather wide application; the second is just beginning to be used but gives great promise. In general it may be stated that fume will be removed from smelter smoke for two reasons: First, because the fume is valuable and its recovery is profitable, as in a number of instances in lead smelting; and, second, to remove noxious material, such as lead compounds, arsenious oxide, and sulphuric acid, from the smoke stream and to prevent its discharge into the atmosphere, where it will do damage.

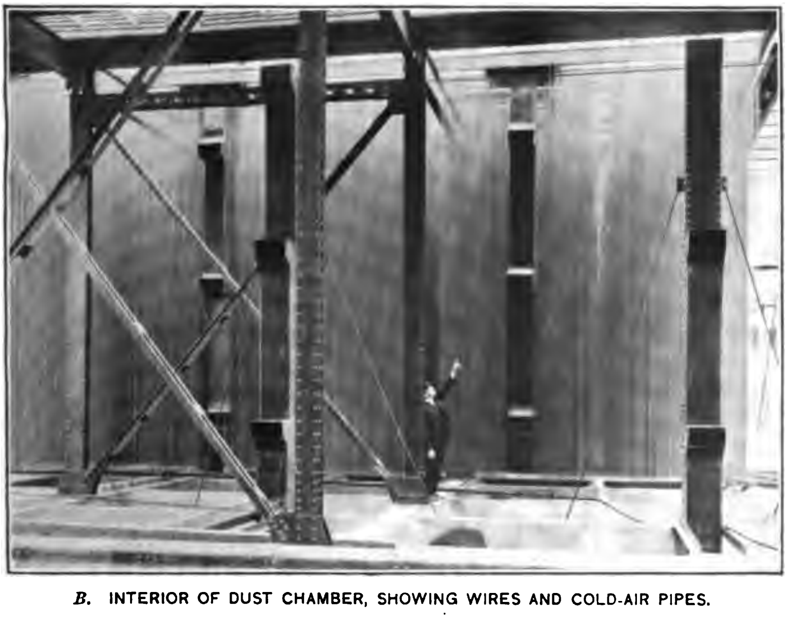
NECESSITY OF BAG HOUSES.

As previously stated, an invisible discharge from smelter stacks is desirable from many points of view. Practically all lead-smelting plants in this country, except a number in Colorado, are provided with bag houses. The bag house has also been applied at one copper-smelting plant, that at Kennett, Cal. The maintenance of a bag house is expensive, and its use is profitable only when smelting in the blast furnace with a high percentage of lead on the charge is being done. Hygienic and legal reasons, however, make its use at lead smelters practically imperative. The fume from a copper smelter is usually not as valuable as that of a lead smelter, so that a bag house for its recovery is even less likely to yield a profit. Furthermore, as regards copper smelters, the temperature of the gases is high and their volume is much greater than in lead smelters, so that the technical difficulties of bag-house operations are greatly increased. Hence the bag house is rarely considered as an adjunct to the copper smelter, except in the instance above noted, where its installation was made necessary by court decisions. The bag-house installation at this plant is described in some detail elsewhere in this report.

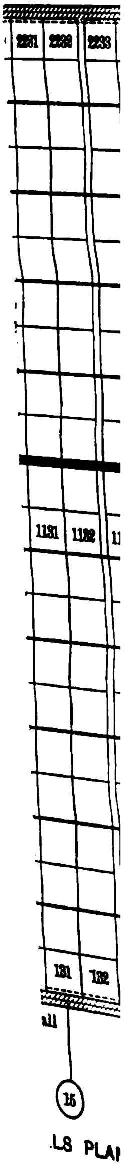
The bag house was first used in the metallurgy of zinc to filter zinc-oxide fume used in paint manufacture. It was next adapted to filtering the fume out of the gases from the lead blast furnace. The endeavor was made to filter the gases from all the furnaces of the lead smelter including roasting furnaces, but it was soon found that



A. HOPPERS UNDER DUST CHAMBER.



B. INTERIOR OF DUST CHAMBER, SHOWING WIRES AND COLD-AIR PIPES.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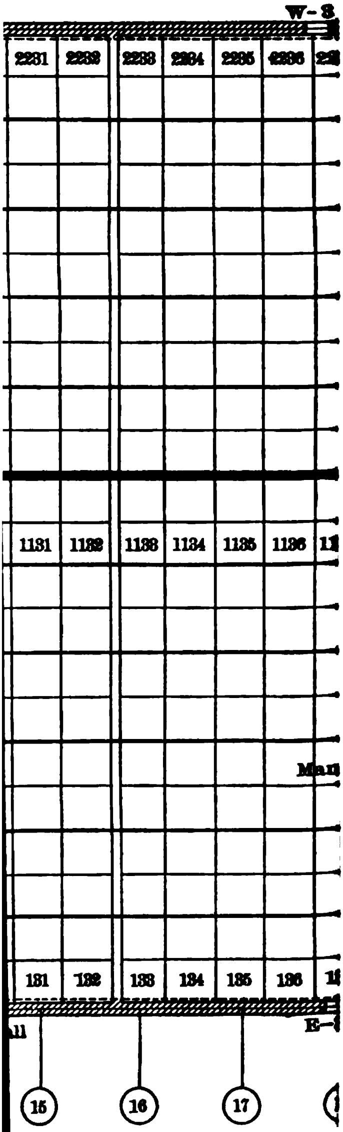
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the sulphur trioxide and sulphuric acid content of the gases from these furnaces was such as to quickly destroy the fabric of the bags employed, whether they were wool or cotton. Only within the past few years has this difficulty been largely overcome, and at present, with proper precautions, the gases from probably any furnace can be filtered and the fume recovered. In a modern lead-smelting plant there may be employed in the reduction of the ores (1) Huntington & Heberlein pot roasters, (2) Dwight-Lloyd straight-line sintering machines, (3) Wedge multiple-hearth furnaces, (4) Godfrey roasting furnaces, and (5) lead blast furnaces. At some of the plants there are still in operation a number of older style roasting furnaces, such as (6) the hand roasting reverberatory furnace, and (7) the Ropp straight-line mechanical furnace.

DIFFICULTIES WITH DIFFERENT GASES.

Gases from hand roasting furnaces caused most trouble in fume filtering, for they contained not only a relatively large percentage of sulphur trioxide, but also finely divided carbon, and hydrocarbons, products of the coal firing, which in the bag house gave much trouble from spontaneous combustion. The gases from the pot roasters do not present any great difficulty. The gases from the Dwight-Lloyd down-draft sintering machines do not contain a large percentage of sulphur trioxide, but contain considerable free sulphur, which in a number of instances recently has caused rather serious trouble in bag houses attempting to filter the smoke stream from this type of furnace. This sulphur is apt to take fire at some point in the flue and slowly burn to the bag house with disastrous results.

Before fume can be filtered from smoke by means of woolen or cotton fabric the smoke must be cooled below a certain temperature and any sulphuric acid in the smoke must be removed or neutralized. It was found in the metallurgy of zinc that zinc fume could be filtered with impunity under almost any conditions for the reason that zinc oxide would readily combine with sulphuric acid when the fume was properly cooled, as the zinc sulphate thus formed was not harmful to the bags. This principle is applied in the Sprague process, in which zinc oxide fume obtained by roasting zinc sulphide (sphalerite) concentrate on a grate furnace with crushed coal, is permitted to mingle with the smoke stream in the flues before the bag house is reached. As the process is somewhat expensive, part of the zinc may be replaced by finely powdered slaked lime, which is blown into the flues in a suitable manner.

CONSTRUCTION OF BAG HOUSE.

The bag house consists essentially of two chambers, the lower or dust chamber being 12 to 16 feet from floor to ceiling, the upper or bag compartment being 30 to 40 feet in height. The walls and partitions are usually of brick, properly buttressed or held together by tie rods. The dust chamber or smoke cellar is divided by brick partitions into as many compartments (called bays) as there are flues leading to the bag house, thus enabling a part of the bag house to be shut down for cleaning out the collected fume without interfering with the rest of the house. The ceiling of the smoke cellar is made of iron or steel plate and is provided with a number of thimbles or rings projecting into the bag compartment. From the supports near the top of the building cotton or woolen bags are hung, and the lower ends of the bags are tied to the thimbles by means of twine, wire, or hoop bands provided with strap clamps. Each compartment of the bag house has a stack to carry the filtered gases some distance into the air. Fans capable of moving 150,000 to 250,000 cubic feet of gas per minute force the smoke stream with its fume and fine flue dust through a number of large conduits into the smoke cellars and through the bags, the solids being removed so perfectly that the discharge from the stacks is invisible.

NUMBER OF BAGS PROVIDED.

The ordinary bag house is provided with 3,000 to 4,000 bags, each bag being about 18 inches in diameter and 30 to 33 feet long. It is usual to provide 300 to 500 square feet of filtering area for every ton of charge smelted in the blast furnace, or about 3.42 square feet per cubic foot of smoke stream per minute, 2.5 square feet of filtering area being considered the minimum surface per cubic foot of smoke stream per minute that is permissible, although as little as 1.14 square feet has been suggested. At the Midvale smelter in Utah, 25,000 square feet of filtering surface per ton of fume collected per day is provided.

MATERIAL USED IN BAGS.

As a filtering medium, cotton cloth having 42 to 50 threads to the linear inch and weighing 0.4 to 0.7 ounce per square foot was largely used. The objection to this material is that at a temperature of about 200° F. it becomes brittle and breaks, and if used at temperatures of about 100° F. it becomes much weakened by the acid substances in the filtered fume. Woolen material withstands the action of the higher temperature as well as the influence of the corrosive substances in the fume. It will retain its elasticity even when used at temperatures of 250° to 270° F. In modern practice, therefore, it is cheaper in the end to use the more expensive woolen bags than the cotton ones. Recent experience has shown that vegetable fibers should not be used at all; even linen thread for stitching is not satisfactory. At the Murray plant near Salt Lake City, Utah, cotton bags lasted 17 months and cost $2.15 each. These have been replaced by woolen bags which have been in use for over three years and will probably last four years; they cost $4.72 each.

In the lead smelter at Pertusola, Italy, lead fume is recovered by filtering the smoke stream through frames filled with several vertically stretched layers of asbestos thread. These frames are placed in the dust chamber through which the smoke stream is forced by fans. The collected fume is dislodged from the filter frames by releasing the tension on the thread and vibrating the frames. This method is known as the Louis B. Fiechter system. It is evident that this filtering material is proof against acid and high temperature, and may be applicable particularly for copper smelting conditions.

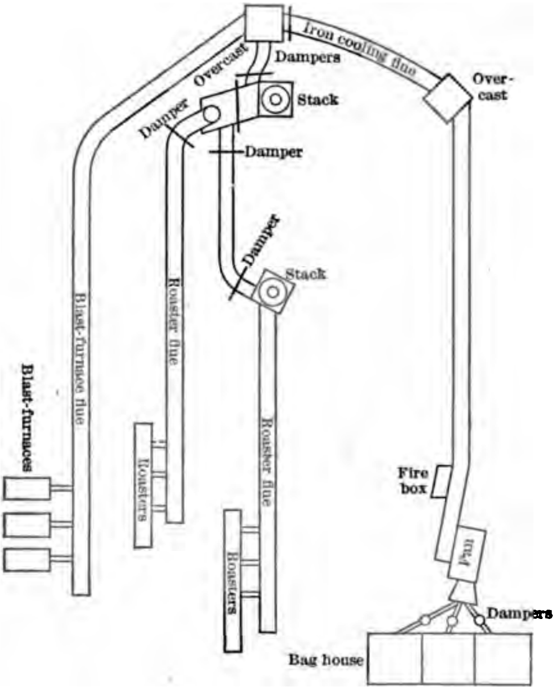


FIGURE 3.—Flue system and bag house arrangement at a western lead smelter.

Experiments in Utah indicated that woven asbestic fabrics are unsuitable to filtering, as they soon become clogged, but the frames with thread in tension are not open to this objection. With woolen bags a wide range of temperature of the smoke stream is permissible. About 150° to 175° F. is safest, but a minimum of 70° F. and a maximum of 270° F. have been used. When the temperature is low there may be trouble from the condensation of water vapor and an increased corrosion from the acid that is formed. The higher temperatures are apt to start spontaneous combustion of the fume or lead to a scorching of the bags. The best practice is to so adjust dampers at the furnaces and in the flues as to maintain a temperature of about 160° F. in the bag house. Under these conditions the material collected in the bag house will be dry and filtration can go on rapidly. At Murray, Utah, the gases entering the bag house have a temperature ranging from about 80° to 125° F. dependent on the season of the year. The placing of limits of temperature on the smoke stream to be filtered makes it necessary to have a flue system under close control. Figure 3 sketches the flue system and bag house arrangement at a western plant. To permit settling of dust and proper cooling of the smoke stream, the usual length of flue between furnaces and bag house is 1,000 to 1,800 feet.

PROPER BAG-SHAKING DEVICE ESSENTIAL.

An important item in bag-house operation is the means adopted for shaking the bags to dislodge the fume caught, so that it may fall into the smoke cellars. The most satisfactory device is the system in which the bags are suspended from short levers attached to a central shaft. The outer end of this shaft projects beyond the wall and is provided with a large lever, so that when this is moved to and fro the tops of the bags are given a similar motion and a wave motion is imparted to the bags as a whole. Bags are shaken usually once every 24 hours. Before the shaking commences, the dampers leading into the bay from the flue are closed.

METHODS OF HANDLING BAG-HOUSE PRODUCTS.

The bag-house fume collected in the smoke cellars usually has a black or dark-gray color (this refers to lead-blast furnace fume), is rather light in weight, and is difficult to handle in its original state. When the fume has accumulated to the depth of about 24 inches, the compartment or bay is closed off from the distributing flue and the cellar doors are opened, as are also the valves connecting with the return flue which leads back to the main flue by means of the fan. Then the fume is ignited by throwing hot coals all over its surface. As the fume burns some fresh fume is evolved from it and is drawn through the return flue by means of the fan and forced again into another part of the bag house. The fume on the floor of the cellar burns slowly, like punk, and gradually sinters to a product resembling soft lava, losing some arsenic and sulphur. The sintering is completed in about five or six days. The sintered fume is removed when cool by laborers using wheelbarrows and shovels. This work, however, is always dangerous and unhealthful, and in the more modern plants efforts are made to handle the fume automatically. At the Murray plant the smoke cellars are somewhat raised and the bottom terminates in a long hopper in which a screw conveyor pushes the fume and dust to the front of the bag house, where it is automatically discharged into cars. The bag house of the Mammoth company at Kennet, Cal., has 20 pyramid-shaped hoppers under the thimble floor. In these hoppers are small circular gates through which the dust is drawn into cars. The dry fume while being drawn off was so annoying to the workmen that the hoppers are now provided with nipples that project into wooden troughs filled with water so that the fume discharges into water, which is drawn into tanks, where the fume settles.

All fume is not combustible, like the fume produced under the reducing conditions obtaining in the lead blast furnace. The fume from copper blast furnaces and from converters is already thoroughly oxidized and on account of its fluffy character gives much more trouble in handling than the lead blast-furnace fume, which can be sintered as described. Lead blast-furnace fume, after sintering, is sent to the arsenic refinery, where it is heated and the arsenic volatilized as arsenious oxide and again sublimed. The residue from this treatment contains considerable lead and some silver and gold and is usually sent back to the blast furnace to be resmelted.

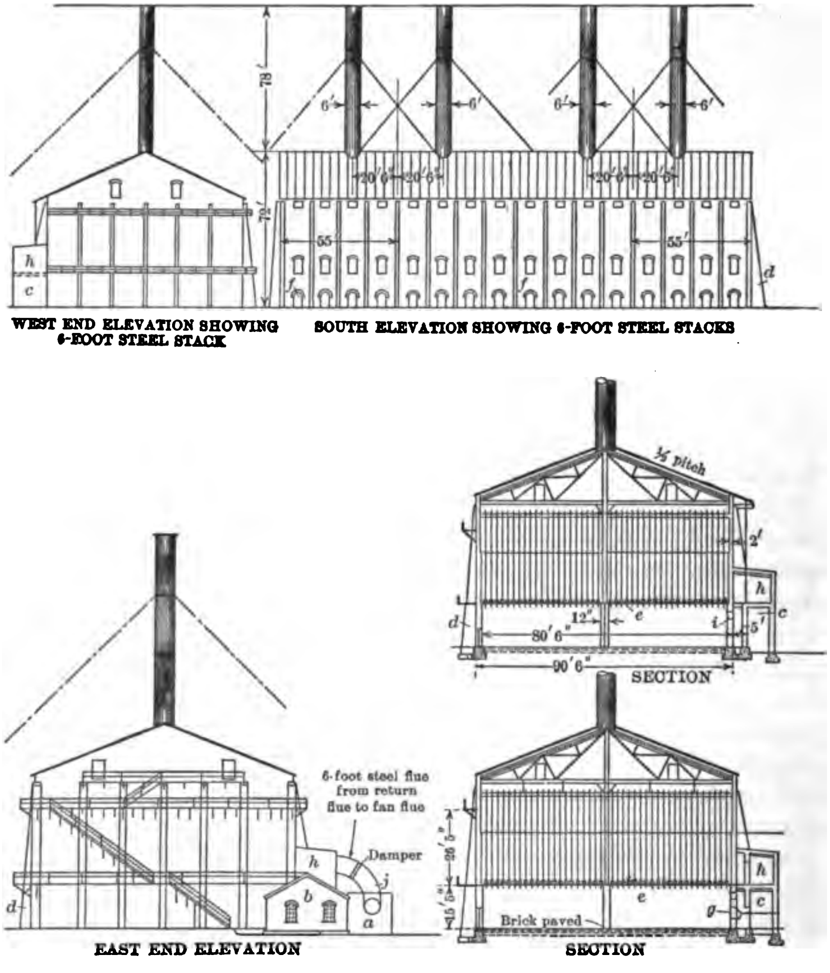
RELATION OF BAG HOUSE TO HEALTH OF WORKMEN.

The labor question in bag-house management is somewhat serious, for the work is dangerous to health. The men employed should be intelligent and have the greatest regard for thorough cleanliness. Heavy clothing, consisting of heavy underwear, socks, overalls, jumper, cap, gloves, and shoes, should be worn. Formerly, rubber coats, gloves, and hats were used, but have now been discarded. A specially designed change house, divided into three parts, should be used. When the men leave their work, they go to the third part, remove their heavy clothing, and hang it where it can be thoroughly aired. They then pass to the washroom and bathe from head to foot, using plenty of soap and hot water. They next go to the locker room where they resume their ordinary clothing. The greatest care is desirable. When eating at the works workmen should always wash their hands thoroughly and be careful not to bring food in contact with their clothing or anything that has been exposed to the fume. With these precautions, men rarely suffer injury from the handling of fume. The same precautions apply to work in the arsenic refineries. Arsenic, aside from the dangers of its ingestion, will produce chronic sores on the skin and mucous membranes. A mixture of ferric hydroxide, lanolin, and some vaseline is a good ointment for the face and other exposed parts and affords considerable protection.

BAG HOUSE AT MURRAY, UTAH.

Figure 4 shows elevations and sections of the bag house at Murray, Utah. The main flue coming from the furnaces is shown at *a*; *b* represents the fan house, containing an exhaust fan with a capacity of 225,000 cubic feet of gas per minute at a pressure of 1 inch of water. The fan is driven by a 75-horsepower motor capable of making 150 revolutions per minute. Pressure in the distributing flue *c* leading to the bag house is usually about 0.5 to 0.75 inch of water, which represents the resistance of the bags to the passage of gases. The outside dimensions of the bag house are 216.5 by 90.5 feet. The height to roof trusses is 51.5 feet.

The house is built of brick, the outside wall being 21 inches thick for a height of 16 feet, then reduced to 17 inches for 18 feet, and to 13 inches for the remaining height. Forty-eight brick buttresses *d*, projecting 4 feet at the floor line and tapering to about 1 foot at the roof-truss line, are built into the walls to stiffen them. There are 17 buttresses on each side and 7 on each end. The bag house is completely divided into four compartments, or bays, by brick partitions running from the floor line to the apex of the roof. Below


WEST END ELEVATION SHOWING 6-FOOT STEEL STACK

SOUTH ELEVATION SHOWING 6-FOOT STEEL STACKS

EAST END ELEVATION

SECTION

SECTION

FIGURE 4.—Elevations and sections of bag house of smelter at Murray, Utah.

the thimble floor *e* each compartment is divided into four chambers or smoke cellars by brick partitions. Each cellar is 87 by 12 feet in area and has a 4 by 8 foot iron door, *f*, through which the fume is removed. The distributing flue *c* is 10 by 16 feet in section, extends the entire length of the bag house, and connects with each of the 16 cellars by means of a 42-inch hinged damper, *g*, opening outward into the distributing flue *c*. On top of this distributing flue is the return flue *h*, extending the entire length of the bag house and connected with each cellar by means of a 34-inch damper, *i*. The dampers open outward into the cellars. The return flue is connected with the main flue *a*, behind the fan, by means of a 6-foot sheet-iron downtake, *j*, and when the fume in any cellar is being sintered, the valve *g* from the distributing flue is closed and the valve *i* is open, so that the path is clear through the return flue *h* and through *j* to the main flue *a*. At the same time the cellar door *f* is open.

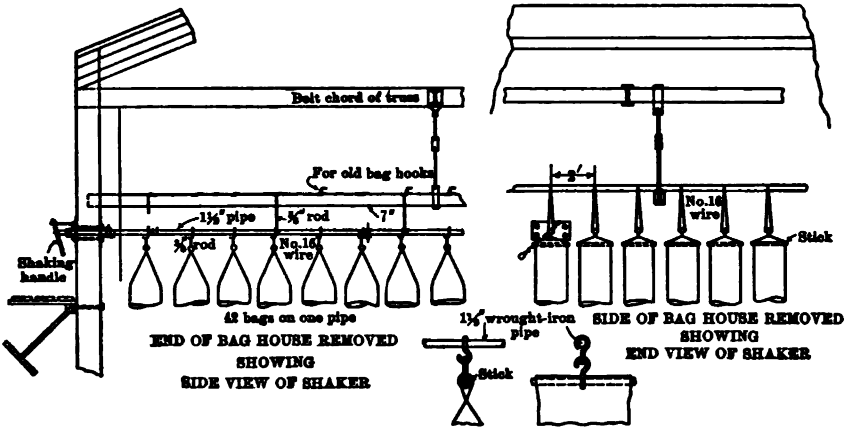


FIGURE 5.—Shaking device and two methods of hanging bags in bag house at Murray, Utah.

The thimble floor *e* is divided into four compartments, each containing 1,008 steel thimbles, 17 inches in diameter and 10 inches high. The thimbles are riveted to the steel floor. The bags, which are 30 feet long and 18 inches in diameter, and number 4,032 in the four compartments, are hung in one of the two ways shown in figure 5. This figure also illustrates the shaking device employed. At the bottom the bags are tied to the thimbles by means of wires. Figure 6 shows the fan house in some detail. The gases are drawn from the main flue *a* through the fan *x* and discharged into the distributing flue *c*. The bag house provides 570,012 feet of filtering area and filters 165,000 cubic feet of gas per minute, or at the rate of 3.45 square feet of surface per cubic foot of gas. With one compartment, or four cellars, shut down for sintering fume, there is available 2.59

square feet of filtering surface per cubic foot of gas per minute. This bag house was constructed in 1906 and cost $127,195.

BAG HOUSES AT TOOELE, UTAH.

Two bag houses were recently constructed at the International plant at Tooele, Utah. Their construction in general is similar to that of the bag house just described. One of the bag houses contains 960 woolen bags 31 feet high and 18 inches in diameter and is used to filter fume from the converters operating on lead-copper mattes. The other bag house contains 1,440 bags, some of wool and some of cotton fabric, and serves to filter the fume from the lead blast furnaces.

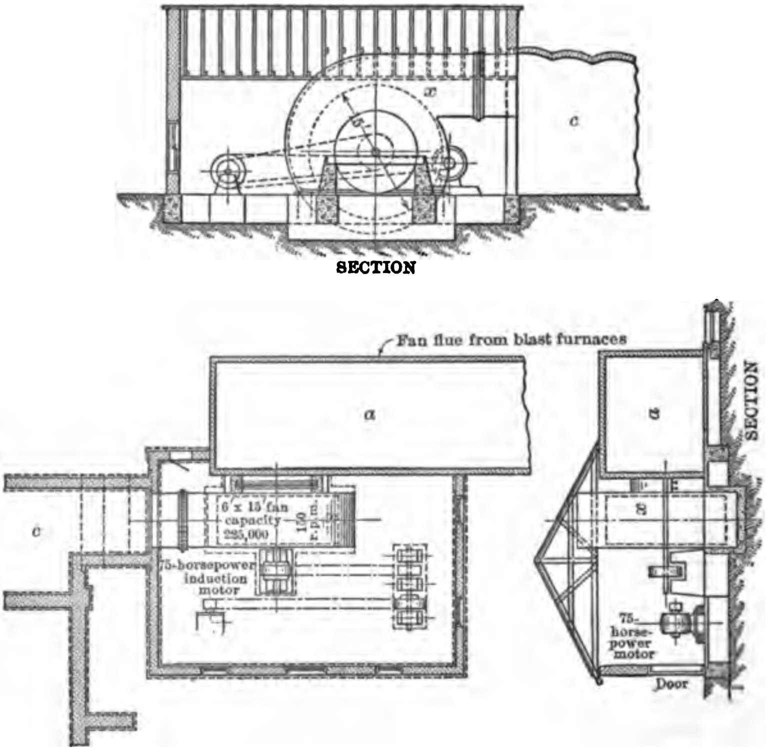

x
c
SECTION
Fan flue from blast furnaces
a
c
6' x 15' fan capacity 225,000
150 r.p.m.
75-horsepower induction motor
a
x
75-horsepower motor
Door
SECTION

Figure 6.—Details of fan house at Murray, Utah.

Figure 7 shows a cross section of one of these bag houses. The gas is discharged by a No. 20 Sirrocco fan, with a minimum capacity of 180,000 cubic feet per minute, into the brick distributing flue *a*, whence it passes through disk valves, *b*, into each of the eight smoke cellars *c*, and thence through the thimbles in a thimble floor, *d*, into the bags *e*, where the fume is filtered out, whence the clear gas escapes by means of a flue in the top of the bag house, shown in the section as a semicircle, *x*, to the chimney, 150 feet high and 15 feet in diameter. The manner of hanging the bags and an auxiliary shaking device for the bags to dislodge the fume is shown in figure 8. Usually, however, the shaking of the bags is accomplished by what is

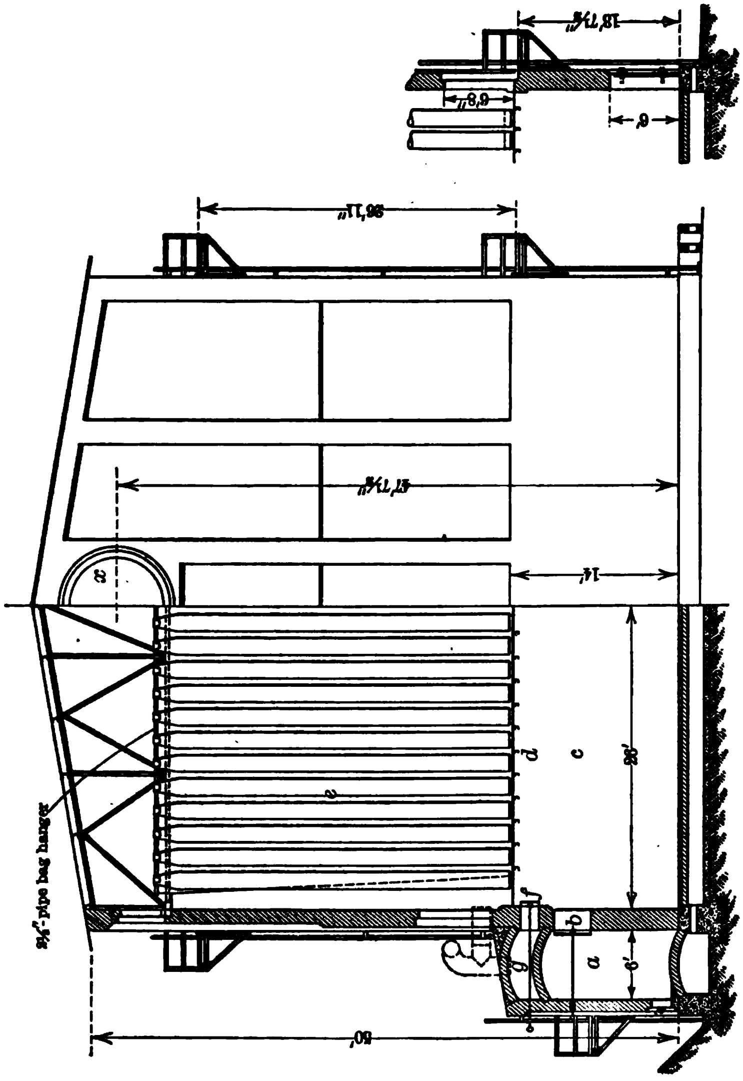


FIGURE 7.—Bag house at Tooele, Utah.

known as the reverse-current system. This comprises a small auxiliary flue, *g* (fig. 9), directly over the main flue *a* and arranged with openings into the smoke cellars. These openings are controlled by the disk valves *f*. One end of this auxiliary flue is connected with the intake or suction of a No. 4½ Sirrocco fan, the discharge from which passes back into the main flue. When the bags are to be shaken the disk valve *b* is closed and the valve *f* is open. This takes the pressure from the bags as the valve *b* shuts off the connection with the pressure discharge of the main fan and the valve *f* opens to the suction of the small fan mentioned. Hence the bags collapse and discharge their fume. By repeating the closing and opening of valves *b* and *f* a number of times, all the fume is readily dislodged. The valves are usually so operated once every eight hours.

Figure 9 shows the valve arrangement of the reverse-current bag-shaking system somewhat more in detail. At the Tooele plant the smoke from the Dwight-Lloyd sintering machines, which roast finely crushed lead sulphide ores, was formerly sent to the bag house. However, the elemental sulphur in fume from the sintering machines caused much trouble by setting fire to the bags. This practice has therefore been discontinued until a solution of the problem can be reached.

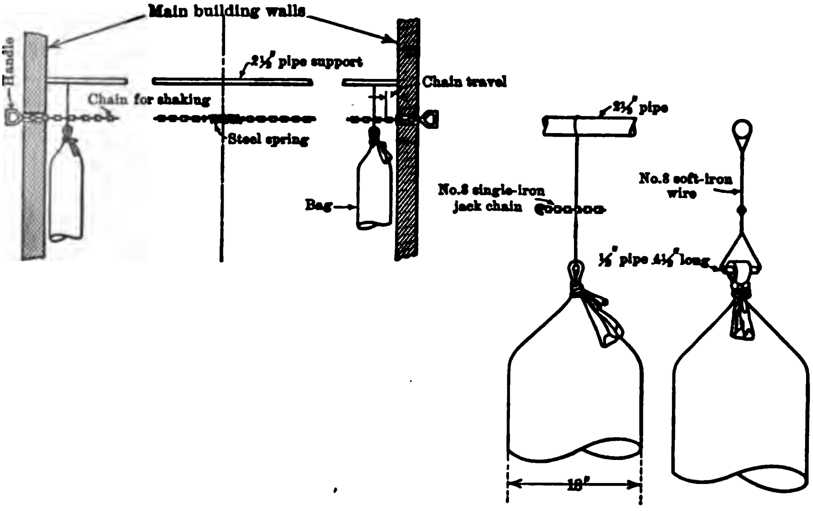


FIGURE 8.—Method of hanging bags at Tooele, Utah.

BAG-HOUSE PRACTICE AT PLANT AT MIDVALE, UTAH.

At the smelting plant at Midvale, Utah, methods were developed to filter all smelter smoke arising from a lead smelting plant, including that from blast furnaces, hand roasting reverberatories, Dwight-Lloyd sintering machines, Wedge mechanical furnaces and pot roasters. The plant employs what is known as the Sprague process for neutralizing the sulphuric acid content of the smoke.

Figure 10 shows a sketch of the smoke-filtering plant at this smelter. There are two bag houses, one (*e*) for blast-furnace smoke, and one (*f*) for the smoke from the roasting furnaces. Together they contain about 3,560 bags, each 33 feet long and 18 inches in diameter. The filtering surface is about 560,000 square feet, and 350,000 cubic feet of gas per minute is filtered. About 20 tons of fume is produced daily. The bag-house cellars terminate in hoppers as already described. The fan *c* takes the smoke from the blast-furnace flue *a* and discharges it into the distributing flue *d*, whence it passes through the cooling chambers and pipes *d′* and through the bag house *e*. At the fan *c* there is a small lime feeder, *b*, which consists of a screw conveyor feeding lime into a hopper at the discharge end of which is a push plate which pushes the finely crushed slacked lime into the fan housing, where the suction of the fan causes it to mix thoroughly with the smoke and neutralize the sulphuric acid. The fan *h* draws on the flues *j* coming from the Wedge furnaces, on the flue *k* coming from the pot roasters *l*, and on the flue *m* coming from the Dwight-Lloyd sintering machines *q*, forcing the smoke through the cooling chambers and pipes *g′* and then through the distributing flue *g* into the bag house *f*.

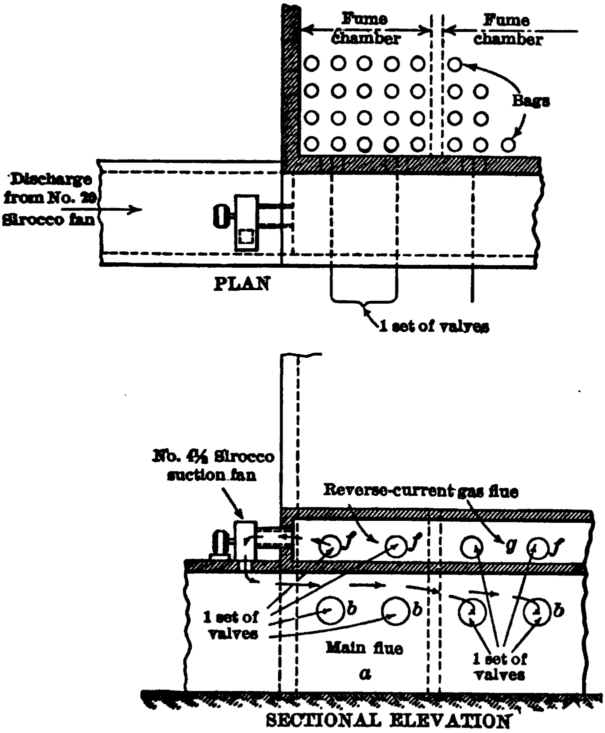


FIGURE 9.—Reverse-current shaking system used in bag house at Tooele, Utah.

At *p* are situated five small grate furnaces, on which crushed zinc sulphide ore or sphalerite concentrate mixed with coal is burnt by blowing air through the grate. The zinc oxide vapor arising from these furnaces is conducted by pipes to the flue *o* carrying the smoke from the Dwight-Lloyd sintering machines to the flue *m*. The volume of zinc vapor admitted to each flue is controlled by the plug valve *n*. At the end of the flue *j*, coming from the Wedge furnaces *r*, is a small fan, *s*, connected with a lime feeder. At this point finely slacked lime is blown into the flue. At *t* lime is blown into the flue *k* from the pot roasters and at *i* into the fan *h*. It is apparent therefore that for the roaster gases neutralization is effected by use of both zinc and lime. Thorough neutralization of these gases is essential, and it is stated that about 3,000 pounds of sulphuric acid is neutralized daily.

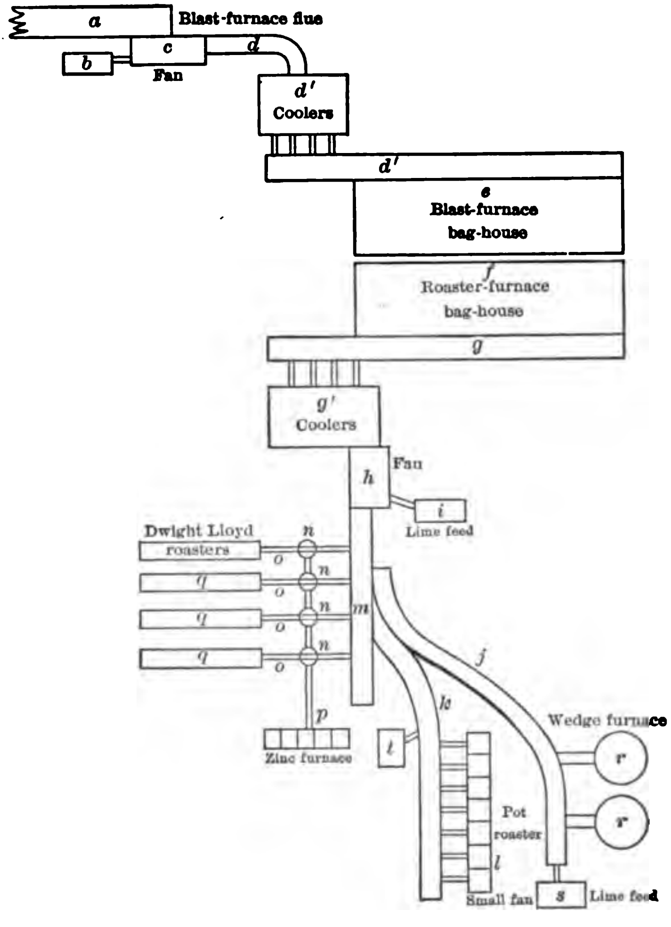


FIGURE 10.—Smoke-filtering plant at Midvale, Utah.

The quantity of zinc and lime fed is controlled by taking daily samples of the fume from the bag houses and determining their alkalinity. A certain excess of alkalinity is always carried to insure the neutralization of all acid. Although lime is fed also to the blast-furnace system, the measure is largely precautionary and not absolutely necessary. Zinc oxide is much more effective as a neutralizer than slacked lime, but on account of its expense is replaced in part by lime, which on the whole is efficient. In the absence of water, lime is not acted upon by sulphur dioxide gas to form sulphites, so that it need only be added in certain small excess over the percentage required for the neutralizing of the sulphur trioxide.

The Midvale plant is remarkable in the appearance of its stacks while in operation. There is no visible discharge from any stack of the plant, and at first sight the plant appears shut down. It is a shining example of what may be accomplished in this direction.

BAG HOUSE AT KENNET, CAL.

The plant of the Mammoth Smelting Co. at Kennet, Cal., is the only copper-smelting plant in the United States to which the bag house has been applied.

The smoke arising from copper blast furnaces and converters is filtered in a bag house 210 feet long and 63 feet wide, and with a main chamber 40 feet high, containing 2,960 woolen bags, each 34.5 feet long and 18 inches in diameter. The smoke enters the bag house at about 203° F. and the volume filtered is approximately 350,000 cubic feet per minute, which includes 70,000 to 150,000 cubic feet of cooling air drawn from the outside atmosphere, the amount depending upon the temperature. The gas comes from three blast furnaces and two converters. The furnaces smelt about 1,000 tons per day. Figure 11 presents a plan and elevation of the flue system and bag house.

ARRANGEMENT OF FLUE SYSTEM.

The gases from the furnaces and converters go through two long brick flues and dust chambers, *a*, parallel to the line of blast furnaces, into four steel pipes, *b*, each 8 feet in diameter, into a collecting chamber, *c*, 42 feet wide and 15 feet high, which contracts at one end and discharges into a steel flue, *d*, 15 by 15 feet in section. This flue leads to two 11-foot Sirrocco fans, which force the smoke each by an 8-foot flue, *e*, into the fan discharge chamber *f*, which is 20 by 15 feet in section and 160 feet long. In the main flue *d* there is vacuum of 1.7 inches of water, whereas in the fan discharge chamber *f* there is a pressure of about 3 inches. At the discharge of each fan is a device for feeding lime to the smoke stream similar to that described for the plant at Midvale. The pipes *e* are provided with dampers so as to regulate the amount of smoke handled by each fan. From the discharge chamber *f* the gases pass through a system of eight 4-foot cooling pipes, *g*, each about 200 feet long and of a U shape. Each pipe is provided with a butterfly valve. The cooling pipes discharge into the bag-house distributing chamber *h*, 13 by 14 feet in section and 210 feet long, into which a separate fan blows the necessary volume of cooling air. The distributing chamber *h* discharges through short pipes into the smoke cellars of the bag house.

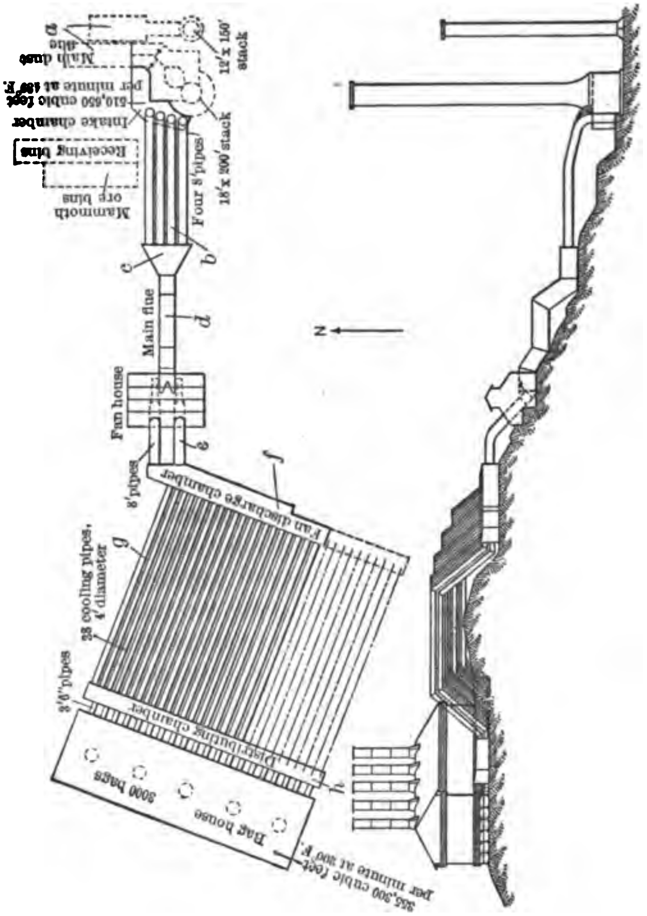


FIGURE 11.—Plan and elevation of bag house and flue system at Kennet, Cal.

NEUTRALIZATION OF SULPHUR TRIOXIDE IN FUME.

The filtering of gases from copper-blast furnaces and converters would not be possible without a neutralization of the sulphur trioxide content. The blast-furnace charge in this particular instance contains a considerable amount of zinc which under the oxidizing conditions of smelting is vaporized in part, with the formation of oxide of zinc. This most efficiently neutralizes the sulphur trioxide formed. Therefore the addition of extraneous zinc, as is done at Midvale, is not necessary. The addition of lime at the fans is said to be really not necessary, but is added as a precautionary measure. The bag house differs from others in the construction of hoppers under the cellars to facilitate the discharge of fume. The manner in which this is done has already been described on page 55. The fume produced is thoroughly oxidized and will not burn. About 16 to 18 tons is produced daily, in 1913 a large pile of it had accumulated in the yards of the company, no method having been devised for its economic treatment and the recovery of the metals contained therein. Table 3 indicates that its composition is essentially different from that of lead-furnace fume, in that the lead and arsenic contents are low and the zinc content is high. The bag house at this plant was installed to conform to court decrees that sulphuric acid should be practically absent from the smoke and that the sulphur dioxide should be reduced to less than 0.75 per cent. The bag house of course has no bearing on the sulphur-dioxide content mentioned, which is obtained by diluting the gases with air as described.

The bag house has five large, low, wooden stacks for the discharge of the filtered gases. Each of these stacks is 21 feet square. The idea of the low stacks with such large cross section is to thoroughly diffuse the gases from the bag house, and to aid in this result additional air is admitted to the bag house.

THE ELECTRICAL PRECIPITATION OF FUME.

GENERAL PRINCIPLES OF ELECTRICAL PRECIPITATION.

The precipitation of suspended matter, whether in gases or liquids, may be accelerated by electricity in the form of either direct or alternating current. Where an alternating current is applied the action consists for the most part in an agglomeration of the suspended particles into larger aggregates and a consequent more rapid settling of these aggregates under the influence of gravity. The application of the alternating current, however, is effective only when the masses of gas or liquid to be treated are fairly quiescent and a simple agglomeration of the suspended particles into larger aggregates is sufficient to effect separation by gravity. As regards large volumes of rapidly moving gases, such as occur in smelter flues, the agglomeration and settling process induced by the alternating current is too slow, even when the flues are expanded into large dust chambers. Consequently the application of the direct current, the action of which is quite different from that of the alternating current, is necessary.

USE OF DIRECT ELECTRIC CURRENT.

If a needle point connected to one side of a high-potential direct-current line be brought opposite a flat plate connected to the other side of the line the air space between becomes highly charged with electricity of the same sign as the needle point, irrespective of whether this is positive or negative. Any insulated body brought into this space instantly receives a charge of the same sign. If this body is free to move as is a floating particle, it is attracted to the plate of opposite charge and will move at a rate proportional to its charge and the potential gradient between the point and the plate. Even if there are no visible suspended particles, the gas molecules themselves undergo this same process, as is evidenced by a strong wind from the point to the plate, even in perfectly transparent gases. The familiar experiment of blowing out a candle flame by presenting it to such a charged point is another illustration of the same phenomenon. The gas is ionized[a] and some of the ions collect upon the solid particles, thereby charging them electrically. As the charged particles are within a strong electric field of constant polarity, they are driven to the collecting electrodes.

An alternating current of high potential or voltage is easily obtained, but the obtaining of a high-potential direct current is more difficult. Generators that furnish direct current at high potential are not built commercially, as they are impracticable. Therefore to obtain high-potential direct current a common custom is to take alternating current of any voltage available on a commercial line, transform it in stationary transformers to whatever voltage is desired (from 15,000 to 50,000 volts), and then obtain an intermittent direct current by means of a special rotary contact maker driven by a synchronous motor. The electrodes or terminals of the current employed are of two types, the first corresponding to the plate to which the particles are driven, and the second to the point or the discharge electrode. The plate electrode is usually made of either iron or lead, and presents no difficulty in construction.

The point electrode, however, must fulfill special conditions. In the early laboratory experiments with the process it was found that cotton-covered copper wire, by reason of the innumerable fine points presented by the threads of the cotton insulation, made a desirable discharge electrode. Under the conditions of acid vapors and high temperature this material, of course, can not be used, but a substitute was found in asbestos-covered wire, or a wire covered with mica, the fine filaments of the one and the scales of the other supplying the discharge point or edges of the excessive fineness required. The asbestos or mica was twisted with the wires or otherwise fastened to suitable metallic supports in such a way that the current had to pass only a short distance by surface leakage over them. The slight deposit of moisture or acid fume naturally settling on them serve to effect the conduction of the current.

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[a] The ionization of a gas is different from that of an ordinary liquid electrolyte. The actual gas ions under the conditions of pressure and temperature existing in the flue are probably fairly large aggregates or groups of undecomposed gas molecules clustered around electrons discharged from the corona of the negative discharge electrode. There are also probably a much smaller number of positively charged groups present, whose initial positive charge may be considered as due to taking away a negative electron from one of the neutral gas molecules.

APPLICATION IN SULPHURIC ACID PARTING PROCESS.

The first of the successful applications of the electrical precipitation process was to the waste gases and vapors from the sulphuric acid parting process for gold-silver bullion at the Selby smelter at Vallejo Junction, on San Francisco Bay, Cal. These gases, which amount to about 5,000 cubic feet per minute and contain a considerable proportion of sulphuric acid in the form of a fine mist, are conducted through a flue built of sheet lead 4 by 4 feet in cross section. In this flue (fig. 12) are hung a series of vertical lead plates, *a*, 4 inches wide, 4 feet long, and spaced 4 inches apart. Between each pair of plates is hung a lead-covered iron rod, *b*, carrying mica discharge material. These rods, which are the discharge electrodes, are supported on a grid work of bus bars, *c*, extending above the lead plates or collecting electrodes. The bus-bar grid work is connected to conductors which pass out through the sides of the flue to the insulators *d*. The collecting electrodes *a* are in electric connection with the lead sides of the flue, which, in turn, is grounded. The current is taken from the power circuit of the plant at 460 volts, 60 cycles, transformed to 17,000 volts, and is then converted into an intermittent direct current by a synchronous contact maker. The power consumption for this installation is about 2 kilowatts, which includes the driving current for the contact maker. The sulphuric acid particles collect on the lead plates and drip down into the flue, from which the accumulated acid is suitably discharged. The apparatus has been in successful use for over seven years. The gases discharged from the stack to which the lead flue leads are practically free from sulphuric acid.

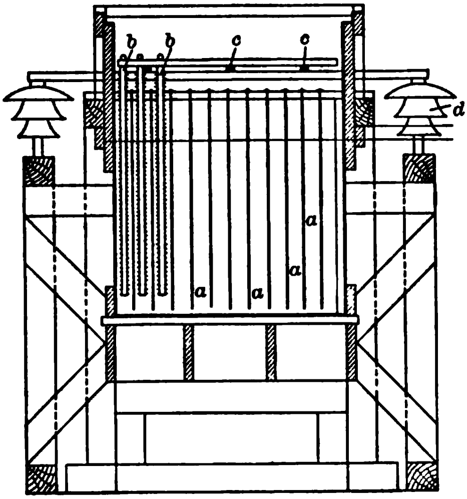

FIGURE 12.—Electric-precipitation apparatus used at Selby smelter at Vallejo Junction, Cal.

APPLICATION AT A COPPER SMELTER.

The electrical-precipitation process was applied on a large scale for the elimination of fume from smelter smoke at the plant of the Balaklala Copper Co. at Coram, Shasta County, Cal. This smelter, now (1914) shut down, daily treated about 700 to 1,000 tons of copper ore containing about 3 per cent copper and over 30 per cent sulphur. The largest part of this ore was smelted in blast furnaces, but about 10 per cent, or the fine material in the ore, was first roasted in four McDougal furnaces and then smelted in an oil-fired reverberatory furnace. There were also two converters for the conversion of matte into blister copper. All of these furnaces discharged into a common flue 18 by 20 feet in cross section. The total volume of smoke was 250,000 to 500,000 cubic feet per minute, giving a linear velocity in the flue of 10 to 20 feet per second. The nature of the installation is shown in figures 13 and 14. The gases from the furnaces in flue *a* (fig. 13) were sent through nine precipitating units, *b*, from which they passed to the outlet flue, *c*, then through the two fans *d* and out through the stack *e*. The gases entered the precipitating units at 100° to 150° C. Ordinarily, the fans were not in operation, but when the furnace charges were especially high in sulphur, it was necessary to use the fans in order to draw in diluting air and mix it with the smoke stream, so that the discharge from the stack would contain less than 0.75 per cent sulphur dioxide to conform to a court decree.

Figure 14 represents a cross section of one of the precipitating units. The collecting electrodes, each 6 inches wide and 10 feet high, made of No. 10 sheet iron, are shown at *a*. The discharge electrodes, consisting of two iron-wire strands, between which was twisted the discharge material for which both asbestos and mica preparations were used at this plant, are shown at *b*. The collecting electrodes *a* were carried by bars, *c*, which were connected directly to the frame of the chambers themselves and thus grounded. The discharge electrodes *b* were spanned by springs *d* between a system of bus bars, *e*, which were carried on externally placed insulators *f*. To the auxiliary chamber *g* surrounding the insulators a small regulated amount of air was admitted to prevent conductive dust or fume from settling on the insulators and thus causing a connection with the sides of the chamber, to which the collecting electrodes were connected. The cam and shaker rod *h*, which extended across the middle of the unit, was designed for the purpose of shaking the collecting electrodes *a* to dislodge the accumulated fume and cause it to drop into the hopper *i*. In actual operation it was found easier to remove the cover *j* after the current had been cut from the unit and shake the electrodes by hand every six or eight hours. The accumulated fume in the hopper *i* was removed by the means of a screw conveyor. In each precipitating unit there were 24 rows of 24 electrodes of each type.

The current used was taken from a three-phase power circuit at 2,300 volts, 60 cycles, which, after having been transformed by stationary transformers to 30,000 volts, was converted into an intermittent direct current by 10 contact makers, driven by synchronous motors, one contact maker for each unit, with an extra one for use in case of accident. The amount of power consumed was about 120 kilowatts. Tests of the apparatus showed that the process removed

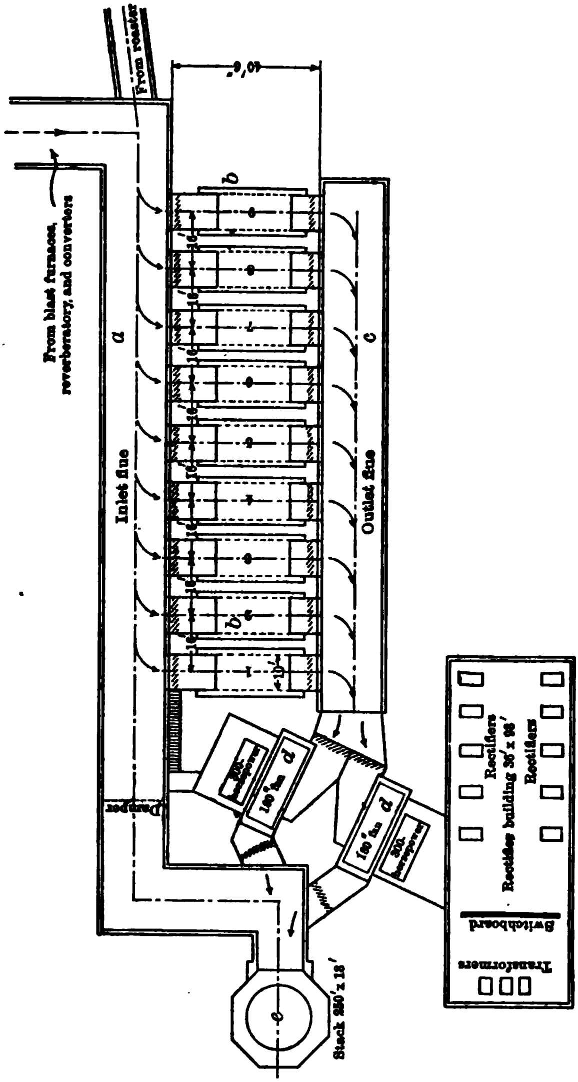

FIGURE 13.—General arrangement of electrical-precipitation plant at Coram, Cal.

nearly 90 per cent of the total solids in the smoke stream. The amount of fume recovered daily was 6 to 8 tons.

One of the most serious difficulties encountered in this installation was the maintenance of the conductivity of the fine fibers of the discharge electrodes. Under ordinary conditions, smelter smoke will contain enough sulphur trioxide or sulphuric acid, which condenses on the fibers, to make them conductive, but in this instance the furnace charge contained a considerable quantity of zinc which formed zinc sulphate. The lack of sulphuric acid greatly reduces the conductivity of the fibers. This difficulty, however, has been overcome in recent work by the use of higher potentials and better designed electrodes by means of which the necessity for the fine discharge fibers, such as the asbestos and mica represented, has been obviated.

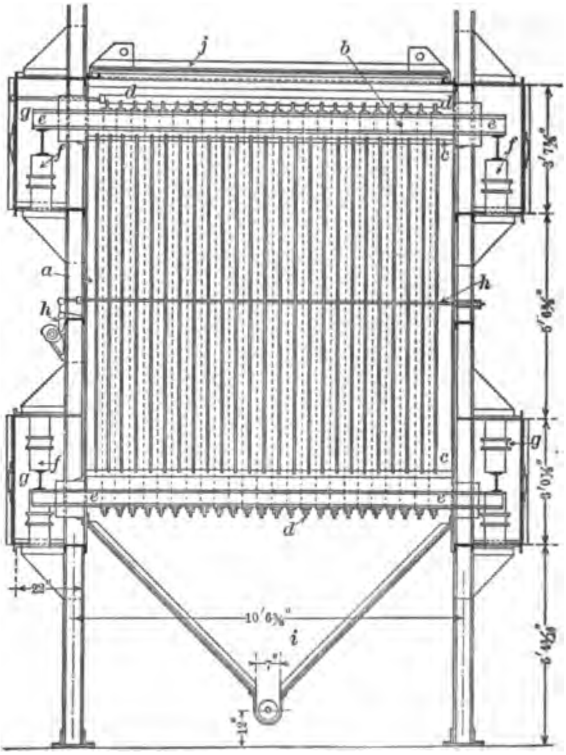

FIGURE 14.—Cross section of precipitation unit in electrical-precipitation plant at Coram, Cal.

The Coram installation is not now in operation, as the smelter was closed in September, 1911, for failure to conform to a court decree. The process, however, may be considered to have been successfully applied and there is little doubt but that with additional experimentation such changes could have been devised as would have made the process entirely successful.

RECENT APPLICATION IN COPPER REFINING.

A recent application of the electrical-precipitation process is an installation at Perth Amboy, N. J., to the gases from furnaces smelting the high-grade residues or slimes from electrolytic copper refining. The material is high in gold and silver contents and contains volatile constituents, such as tellurium, selenium, and lead, and the fume condensed from the gases contains appreciable quantities of precious metal.

The process was also applied experimentally to the smoke from the copper converters at Garfield, Utah. The matte converted at this plant contains somewhat more lead than is usual for copper matte. The lead is volatilized during the conversion and passes to the smoke. The process was applied to this material successfully and in somewhat different form than already described. The so-called "tubular treaters" were used. These consist of a number of pipes, each 10 feet long and 5 inches in diameter, forming a collecting electrode, down the center of which passes the discharge electrode in the form either of single wires or of a metal rod, with four or more radial wings of sheet metal to act as the discharge edges. The pipes may be of sheet lead, vitrified tile, or wood, dependent upon conditions. The smoke enters the pipes at the bottom, the fume being precipitated on the walls of the pipes, eventually dropping into a collecting hopper; the gases escape. In Plate V, *A* and *B*, is shown such a tubular treater with the current off and with it on. Voltages up to 50,000 were used. A permanent installation[a] for the treatment of all the converter smoke is now in operation.

METHODS FOR REMOVAL OF SULPHUR DIOXIDE FROM SMELTER SMOKE.

The effect of sulphur dioxide has been fully discussed in the preceding pages. Obviously, if its concentration in smelter smoke discharged from chimneys reaches a certain figure, the value of which at the present time has not been determined, the natural diffusive forces do not act quickly enough to dilute the smoke stream to such an extent as to prevent all damage. The question as to whether sulphur dioxide or the solids and sulphuric acid discharged as fume are the chief cause of damage has been widely discussed. There is, however, no question but that both have caused damage. Lead-smelting plants discharge gases that on the whole are comparatively low in sulphur dioxide, and it is probable that damage formerly caused by the smoke stream from such plants was due in large part to the solids. In copper smelting the sulphur content of the furnace charges is much higher, and it is probable that such damage as has been done was largely due to sulphur dioxide. Therefore much effort has been directed toward finding methods for the removal or destruction of this substance, but, to the present time, with little promise of success. The amount of sulphur dioxide discharged daily by all the smelting plants of the country is enormous, and the utilization of all of it in some form, such as sulphuric acid or sulphur, is at present not feasible, as the amount of these substances produced would many times supply the possible market demand.

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[a] See Howard, W. H., Electrical fume precipitation at Garfield: Trans. Am. Inst. Min. Eng., Bull. 92, 1914, p. 2029.

UTILIZATION OF SULPHUR DIOXIDE AS BY-PRODUCT DESIRABLE.

The first thought, of course, that comes to the engineer, is to transform this now useless substance into some valuable by-product, such as the sulphuric acid or sulphur mentioned above. The solution of the problem along these lines is so influenced by local conditions that no general statements can safely be made. In the South, there is a considerable demand for cheap sulphuric acid in connection with the treatment of phosphate rock in the manufacture of fertilizer, and also for the purpose of converting the ammonia by-product of coking ovens into ammonium sulphate. This demand made it possible, in a large measure, to solve the smoke problem in the vicinity of Ducktown and Copperhill, Tenn., where two smelting plants manufacture sulphuric acid from the sulphur dioxide in smelter smoke. The smelters of the West are not favorably situated in this respect, at least at present, as the market for sulphuric acid is comparatively small. In Utah, however, there are large phosphate deposits which no doubt will be utilized in time, so that a great demand for cheap acid may develop. As already stated, the problem is always local. At Anaconda the question of treating the tailings or refuse product from the concentration mills for the recovery of their copper content is now being seriously considered. The process that will be applied is one of roasting the tailings and leaching the roasted product with sulphuric acid. This sulphuric acid will probably be manufactured locally from the gases of the McDougal furnaces that roast the concentrate of the mills. Although all of the sulphur dioxide produced at Anaconda can not be utilized, the removal of a part of it from the smoke stream will have a generally desirable effect.[a]

The Hall and the Thiogen processes, now prominent in the amelioration of the smoke difficulty, are designed to remove sulphur from the smoke stream in the form of elemental sulphur. The Hall process eliminates the sulphur in the ores direct in the form of elemental

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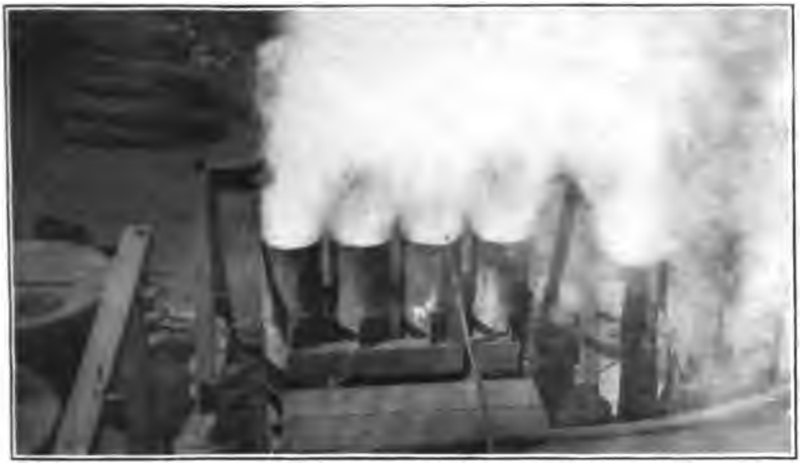
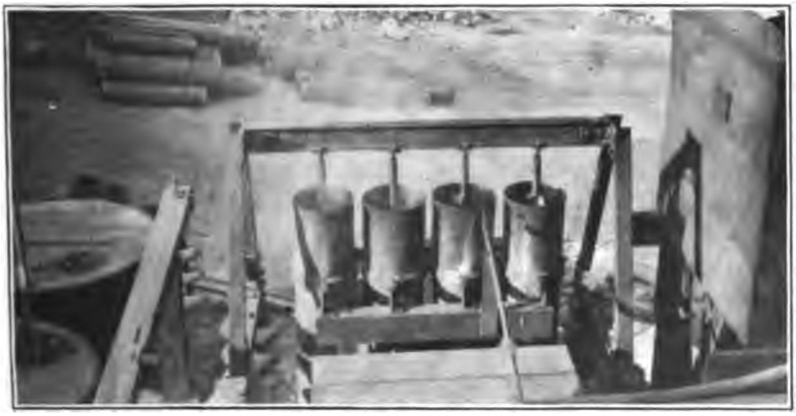
[a] Incidentally another desirable feature is that the spent liquors containing iron sulphate and other salts from the leaching process can be dumped back over the tailings and thus cement the tailings into a coherent mass and prevent their blowing away as fine dust clouds, which are at present a considerable nuisance in certain parts of the surrounding valley.



A. ELECTRICAL TUBULAR PRECIPITATOR, CURRENT OFF. SMOKE IS LADEN WITH FUME.



B. ELECTRICAL TUBULAR PRECIPITATOR, CURRENT ON. SMOKE IS INVISIBLE.

C. SMELTER AND ACID PLANT AT COPPERHILL, TENN.

sulphur and prevents the formation of sulphur dioxide; the Thiogen process destroys the sulphur dioxide in the smoke stream, with the production of elemental sulphur. The leading idea here also is the winning of a valuable by-product, which will yield a profit or at least meet the expense of the applied process. Elemental sulphur at present has a ready market, but it is easy to show that the possible production of sulphur by these means will glut the market. The problem is again a local one.

If the sulphur dioxide in smelter smoke can not be converted into valuable by-products, the problem becomes one of finding a cheap means to destroy the sulphur dioxide and convert it into a harmless substance. From this standpoint, although for obvious reasons the sulphur dioxide may not be converted into sulphuric acid, it might still be converted into elemental sulphur, which could be stored in isolated places. At various times many suggestions have been made for the destruction or removal of sulphur dioxide, such as its removal by solution in water or its absorption by limewater, with the formation of lime sulphite, all of which, except under certain local conditions when they may become applicable, in the end give rise to just as great a nuisance as does the original substance. At present the best-known method for minimizing the nuisance of sulphur dioxide in smelter smoke is to dilute the smoke and diffuse the dioxide to such an extent that it will be practically harmless.

MANUFACTURE OF SULPHURIC ACID FROM SMELTER SMOKE.

In this country two neighboring plants manufacture sulphuric acid from smelter smoke. One, that of the Tennessee Copper Co., is at Copperhill, Tenn., and the other, that of the Ducktown company, is at Isabella, Tenn. Both plants practice pyritic smelting, the ores consisting largely of pyrrhotite, cupriferous pyrite, and chalcopyrite, with smaller amounts of other minerals. The sulphur content of the furnace charge is high, about 21 per cent. The copper in the ores is about 2 per cent. If sulphuric acid is to be manufactured from sulphur dioxide in smelter smoke, certain conditions must necessarily be fulfilled. The chief one is that the gas have a high enough sulphur dioxide content—not less than about 4.5 to 5 per cent. The proportion of carbon dioxide gas should not be over 4 per cent.

Methods have been proposed to concentrate sulphur dioxide by passing the smoke stream upward through towers filled with coke or other material, over which a stream of cold water is constantly trickling. The water discharged from the towers will contain considerable sulphur dioxide, and if it be heated this gas will be liberated. Whether the process would afford sufficient concentration under existing conditions is questionable, and the added step might inordinately increase the expense. It is safe to state that sulphuric acid can be manufactured commercially only in those plants in which the gas contains the required amount of sulphur dioxide without concentration. This requirement eliminates many smelters and limits others to gas from certain furnaces only, such as the roasting furnaces, which treat charges high in sulphur.

DESCRIPTION OF COPPERHILL AND ISABELLA PLANTS.

At Copperhill, the gases from the four blast furnaces smelting the crude or green ore are sent to the acid plant. The gases from the converters and from the blast furnaces that resmelt the matte from the green ore furnaces into a more concentrated product are sent directly to the stack, as they do not contain sufficient sulphur dioxide.

Figure 15 shows the various steps of the acid process. The path of the gas and acid is indicated by light lines with arrows. The furnace gas has a composition approximately as follows: Carbon dioxide, 3.5 per cent; sulphur dioxide, 5.3 per cent; oxygen, 8 to 10 per cent; remainder, nitrogen. This gas passes from the comparatively tight furnace tops of the four furnaces by means of individual downtakes into the acid-plant dust chamber, from which it passes by a cross flue into a large distributing flue or secondary dust chamber, from which in turn it passes by two cross flues into the two niter houses, where the nitrous oxides are added.

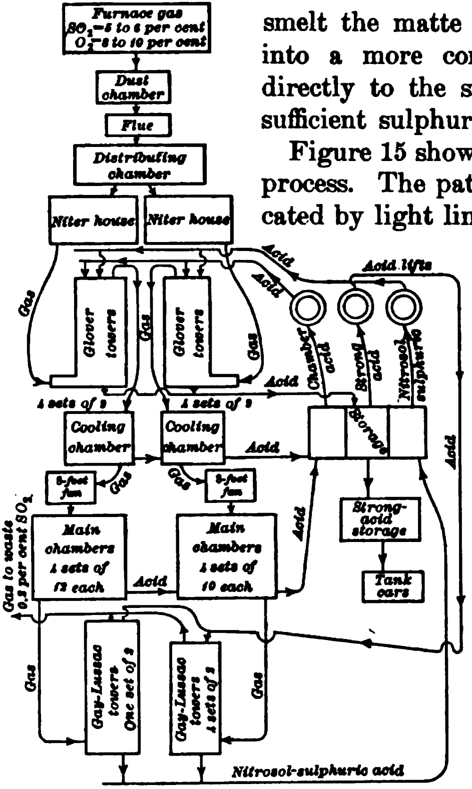
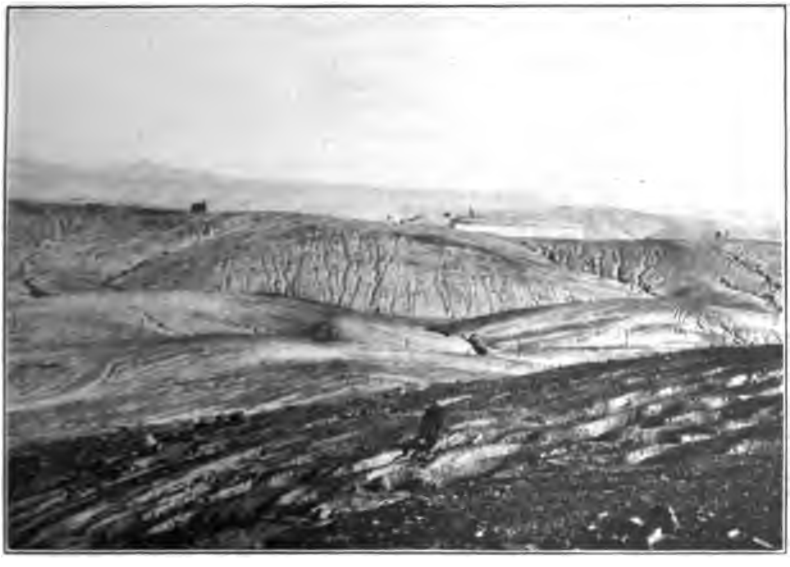


FIGURE 15.—Various steps of process of making sulphuric acid from smoke of smelter at Copperhill, Tenn.

Plate V, *C*, shows a general view of the smelting and acid plants. Although acid is manufactured from the gases, the main stack in the rear of the picture is seen to be discharging fume. This is due to the fact that only part of the gas was going to the acid plant, as already stated. The stack in the foreground is the boiler stack.

Plate VI, *A*, shows a view of the Isabella acid plant and smelter, which treats all of its smoke for acid. Plate VI, *B*, is a view near the Isabella plant, showing damage due to the heap roasting formerly employed.

A. SMELTER AND ACID PLANT AT ISABELLA, TENN.



B. VIEW NEAR DUCKTOWN, TENN., SHOWING DAMAGE DONE BY HEAP ROASTING.

PRODUCTION OF ACID AT COPPERHILL.

In 1911 the plant at Copperhill, Tenn., smelted 383,248 tons of sulphide ore. The gas from the furnaces was made into acid. The ore contained 23.4 per cent sulphur, or 89,774 tons, of which 70,100 tons was made into about 150,000 tons of acid with a specific gravity of 60° B. In 1912 the same plant produced 192,084 tons of acid, or at the rate of almost 600 tons of acid daily. The capacity of the plant is about 700 tons per day. The total acid production in the United States in 1912 was 3,462,000 long tons of acid with a specific gravity of 50° B. (62.18 per cent pure). Of this production about 270,000 tons was derived as a by-product from copper smelting and 292,748 tons as a by-product from zinc smelting, a total of 562,748 tons derived as by-products from metallurgical works.

METHOD EMPLOYED AT COPPERHILL PLANT.

The manner in which the gas used for acid making is produced differs somewhat at the two plants. At the Copperhill plant the sulphide ores are smelted by pyritic smelting in blast furnaces with excess air, the excess being blown in at the tuyères. This excess of air is such that the gases issuing from the top of the furnaces contain 8 to 10 per cent oxygen, the proportion required in acid making. In consequence of this excess air, the gases from the furnace have the high temperature of about 916° F., and contain practically no elemental sulphur in the form of vapor. They enter the niter house at approximately the same temperature. The amount of air used by the furnaces is about 59,000 cubic feet per ton of furnace charge, exclusive of coke.

METHOD EMPLOYED AT ISABELLA PLANT.

At the Isabella plant the sulphide ores are smelted pyritically in blast furnaces, but with only the volume of air normally required to carry on the smelting. In consequence the top of the furnace is comparatively cool, and the gases contain no excess oxygen but considerable elemental sulphur. The gases normally contain 7 to 8 per cent sulphur dioxide, 4 to 5 per cent carbon dioxide, 0.5 to 1 per cent free oxygen, and about 1 per cent carbon monoxide and appreciable sulphur vapor. These gases pass into a dust chamber and flue and thence to the acid plant. The flue leading from the dust chamber is connected with an air pipe from the blowers that furnish air for the blast furnaces. This pipe supplies the air that must be added to the furnace gases to give them the oxygen content necessary in acid making. At the point where the air enters the flue the sulphur vapor burns to sulphur dioxide. The combustion of the sulphur at this point raises the temperature of the gases to 900° to

1,000° F., the temperature necessary in the niter house, and the addition of air reduces the sulphur dioxide content of the gas to about 4.5 to 5 per cent. In other respects the process of making acid at the two plants is very similar. The gases discharged from the acid works are practically free from sulphur dioxide, containing normally about 0.2 per cent, which is not sufficient to cause damage.

THE PRODUCTION OF ELEMENTAL SULPHUR.

THE HALL PROCESS.

DIRECT APPLICATION OF PROCESS.

The Hall process is the invention of William A. Hall, and designs to remove sulphur directly from sulphide ores in the form of elemental sulphur, thus preventing the formation of sulphur dioxide. Proposal has been made to apply the process to the roasting of sulphide ores in furnaces like the McDougal or Wedge multiple-hearth furnaces. Essentially, it consists of submitting the ore to closely controlled oxidation in the presence of water vapor, resulting in the conversion of the sulphides of the metals into oxides, and the liberation of the combined sulphur in the form of elemental sulphur which is collected in water or by other suitable methods. In the process the roasting furnace is fitted with burners for the combustion of oil gas or any gas of high calorific power, although ordinary producer gas may be used. The furnace is tightly inclosed, and the admission of air is accurately controlled. By a proper adjustment of the burners an oxidizing, a neutral, or a reducing atmosphere may be attained in the furnace. It is stated that a furnace operated in this manner attains the same capacity as if it were roasting ore in the usual manner, proportionately to the sulphur eliminated.

PRODUCTS AND CHEMISTRY OF THE PROCESS.

The product of the furnace consists of the oxides of the metals and some undecomposed sulphides, the amount of the latter depending upon how much sulphur it is desired to leave in the calcines or roasted product. The chemistry of the process,[a] pyrite being taken as the sulphide mineral, is probably as follows: One atom of sulphur in the pyrite is readily distilled as elemental sulphur in a neutral or reducing atmosphere. The iron sulphide remaining, at a temperature between 700° and 900° C., which is attained by the combustion of gas or oil at the burners, is converted by the action of water vapor probably into magnetic iron oxide and sulphureted hydrogen. The sulphureted hydrogen is converted to elemental sulphur and water vapor by the limited proportion of atmospheric oxygen present, aided perhaps by the presence of some sulphur dioxide.

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[a] The chemistry of the process is at present under investigation, and the statements here made are provisional only and subject to correction.

The reaction taken as a whole is exothermic; that is, it liberates heat to the extent of 753 calories per kilogram of iron sulphide entering into the reaction, provided the final products of the reaction are assumed to be magnetic oxide and sulphur. Water vapor is necessary in the process for the purpose of furnishing hydrogen to temporarily combine with the sulphur during the decomposition of the iron sulphide by oxygen. It is apparent from the reactions that considerable atmospheric oxygen is needed, so that the iron may be oxidized. Theoretically the product may consist of either ferric oxide or magnetic oxide, and it is this fact that permits a considerable adjustment of conditions in the furnace, for even if a mixture of these oxides be produced the result is still satisfactory. If the conditions in the furnace are maintained too strongly reducing, there is not sufficient oxygen to decompose all of the hydrogen sulphide formed and some escapes with the gases from the furnace. If, on the other hand, the conditions are maintained too strongly oxidizing, the excess oxygen over that required for the iron oxidizes sulphur to sulphur dioxide, and this will be present in the gases coming from the furnaces. The success of the process, therefore, depends upon a close adjustment of the volume of air entering at the burners. The water vapor or steam needed may be injected at the burner, or a sufficient amount may result from the combustion of the hydrogen in the fuel gas that is being used. It has also been noted that organic sulphur compounds form to some extent during the roasting, but little information on this phase of the process is at present available.

The sulphur escapes from the furnace as vapor of sulphur in the gases, and it is stated that if these gases be submitted to the action of water, as, for instance, in centrifugal washers, such as the Thiesen or Feld washer, used to wash iron blast-furnace gases, the sulphur may be readily removed in the water, from which it can be settled out. It is also possible that the electrical-precipitation process may be applied, for recent experiments with the Feld washer show that the washing method presents unusual difficulties. It is stated that sulphur can be produced by this process at a cost of $3 to $5 per ton, but recent experiments seem to show this estimate to be too low. During 1913 sulphur was quoted at about $22 per ton, New York. The fuel consumption in the process is said not to be more than 10 per cent of the weight of the ore roasted. The process is being investigated experimentally on a large scale in the copper-smelting plant at Coram, Cal.

PROSPECTS FOR SUCCESSFUL USE OF PROCESS.

The process is one of considerable promise, but it can not be looked upon, even if successful, as solving the smoke problem as a whole. It may solve the problem locally, dependent upon conditions. One of the great advances in the metallurgy of copper within the last 10 years is the general adoption of the process of smelting sulphide ores in the raw state in the blast furnaces, the fuel value of the iron sulphide being utilized. This plan has decidedly lessened the consumption of coke and the cost of smelting. At present whenever sulphide-copper ores are of the proper physical character—that is, in not too fine a state of division—they are smelted to copper matte in the blast furnace without any preliminary roasting. On the other hand, sulphide ores that break very fine in the mine, or sulphide concentrate, must first be roasted for the elimination of a large part of the sulphur, after which it is smelted in reverberatory furnaces. The Hall process would find its application to those ores that must first be roasted. The California copper-smelting plants in Shasta County, at one of which the process is now being applied, smelt the ores largely in blast furnaces. Whether the grade of ore will permit any other form of smelting, such as roasting by the Hall process and reverberatory smelting of the resultant calcines, is a question that remains to be settled. It is, however, true that the cost of coke is high ($11 per ton), whereas crude oil for roasting and reverberatory smelting is cheap (2½ cents per gallon). Recent work at Coram on the Hall process shows, on steady runs of a week or more, an oil consumption of 20 gallons per ton of Balaklala ore. One 18-foot McDougal roasting furnace had a capacity of 24 tons on ore of such fineness as to pass a ½-inch screen. The sulphur content of the calcines was 6 to 8 per cent. The same furnace had a capacity of 32 tons on ore crushed to pass a 10-mesh screen, the calcine containing 3 to 5 per cent sulphur.

THE THIOGEN PROCESSES.

The Thiogen Co. has two processes, both based on the same general chemical reactions, but differing widely in their application and possibilities.

CHEMISTRY OF "DRY PROCESS."

The so-called "dry process" is based on the fact that when sulphur-dioxide gas of suitable concentration is mixed with hydrocarbon gas or practically any refining gas, and the mixture of such gases is passed over a heated basic sulphide such as calcium sulphide (CaS) or barium sulphide (BaS), a reaction takes place in accordance with the following equations:

$$2CaS+3SO_2=2CaSO_3+3S$$
$$2CaSO_3+2CH_2=2CaS+2CO_2+2H_2O$$

In this general reaction the base used plays a purely catalytic or cycle rôle. If pure gas were used the result of the reaction would be in accordance with the equation:

$$2CH_2{}^{a}+3SO_2=2CO_2+2H_2O+3S$$

The last reaction does not take place, however, to any appreciable extent in the absence of a base.

Essential to the commercial success of this process are several conditions, as follows:

The sulphur-dioxide gas must have a concentration of not less than 8 per cent by volume. Below this concentration the reaction is incomplete, and although the gas is completely decomposed there is formed a large amount of gaseous organic compounds by the combinations of the sulphur formed with the various constituents of the hydrocarbon gas used. With a gas carrying about 11 or 12 per cent sulphur dioxide by volume, the tendency to the formation of these organic sulphur compounds is greatly decreased, and other conditions being correct sulphur is formed in nearly theoretical proportions and of a high degree of purity.

The gases, both sulphur bearing and hydrocarbon, must be free from mechanical impurities, such as flue dust or soot, as any solids carried to the basic material soon form a coating over it and destroy its activity.

The reaction starts at a temperature close to 750° C., and should not be allowed to go above 900° C., as the sulphide has a corrosive action at the higher temperatures.

Any free oxygen in the gas must be combined with the hydrocarbons before the reaction will approach completion. If the proportion of free oxygen present is large, the heat developed from its combination with the hydrocarbon gas presents a difficult mechanical problem both in its disposition and its control.

On account of the various controlling and limiting factors, this process seems applicable only to gases of high and uniform sulphur-dioxide concentration, such as roaster gases and by-passed sintering-machine gases, unless a part of the final sulphur product is continuously used to raise the concentration of sulphur dioxide in the gases before treatment.

PROCEDURE IN "DRY PROCESS."

The general method of procedure in the "dry process" is as follows:

The hot gas is taken from its source, such as a roaster, and enters directly into a steel combination heat radiator and dust collector

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[a] The formula CH_2 is here used to represent gases of the olefin group of hydrocarbons, of which ethylene or olefiant gas (C_2H_4) is the type.

filled with sets of vertical pipes through which the gases, after having been cleaned and cooled, are returned and partly preheated. The proportion of cool gas passing through the transfer tubes is controlled by means of a by-pass, and in this way the exit temperature of the hot, uncleaned gases from the radiator is never allowed to fall below 280° C.

The uncleaned gases at this temperature are taken into a water-jacketed, lead-coil combination cooler and scrubber and are completely cooled and cleaned of all mechanical impurities and solids. Sulphuric acid is used for the scrubbing.

The gas then passes through a filter to remove all acid mist carried over from the scrubber. From the filter the dry, clean gas enters the suction side of a positive blower and is passed through the tubes in the steel radiator, as before mentioned.

The roaster gas is next mixed with the hydrocarbon gas in the correct proportions, and the combined gases passed to the contact chamber. This chamber is double walled and divided into four outside chambers and one large inner chamber. The inner chamber is filled with the contact material (CaS) and is also traversed by a large number of fire-clay pipes. By a peculiar arrangement the mixed gases enter in parallel the two outer chambers, and then have to pass through the clay pipes into the second chambers, and from there downward to the base of the contact chamber, which the gases enter from both sides. A great amount of heat is developed in the contact chamber by the reaction, and this heat is radiated through the clay tubes and preheats the mixed gases to the reaction temperature before they enter the base of the contact chamber. The transference of heat also serves to keep down the temperature of the contact chamber, which would otherwise be excessively hot.

The hot exit gases from the contact chamber, now carrying sulphur vapor, are passed through the flues of a boiler-type condenser. The heat from the gases is used to generate steam, and this steam is kept at a pressure of about 30 pounds, insuring the correct temperature in the tubes for the sulphur to condense into its liquid form and flow freely.

The final waste gases, free from sulphur, are discharged from a small stack.

CHEMISTRY OF "WET PROCESS."

The so-called "wet process" of the Thiogen Co. is based on the fact that when a basic sulphide such as calcium sulphide or barium sulphide, either in finely divided water suspension or in solution, is treated with sulphur dioxide gas free from carbon dioxide the following reaction takes place in the cold:

$$2Ca(\text{or}Ba)S+3SO_2=2Ca(\text{or}Ba)SO_3+3S.$$

The absorption and reduction of the sulphur dioxide gas in a water suspension of calcium sulphide was first tried, and with gases free from carbon dioxide was accomplished satisfactorily with only a slight loss of sulphur as hydrogen sulphide gas. However, instead of the reaction proceeding strictly according to the above equation, there was formed a large amount of thiosulphates, dithionates, trithionates, and tetrathionates, and bisulphites, trisulphites, and tetrasulphites of calcium, all of which were soluble up to a certain concentration. The formation of such compounds resulted unfavorably in two ways, as follows:

(1) The water or liquor of suspension could not be thrown away, as to do so would have entailed a large loss of sulphur and calcium.

(2) The specific gravity of the solution, after having been used as a liquor of suspension, with repeated additions of fresh calcium sulphide after each filtration of the precipitate, gradually raised, unless kept well cooled, to about 1.3, owing to the large proportion of sulphur compounds of calcium in solution. At this gravity the settling of the sludge (calcium sulphite and sulphur) was slow and unsatisfactory. Filtration, likewise, was unsatisfactory, as the thiosulphates in solution crystallized slowly and tended to completely clog the filters by precipitating in the meshes of the filter itself. For these reasons, calcium was abandoned, temporarily at least, and barium used.

Barium sulphide is rapidly soluble in water, whereas calcium sulphide is not. If any thiosulphates or other sulphur compounds of barium are formed in addition to the barium sulphite they are practically insoluble and precipitate rapidly along with the main products of the reaction, barium sulphite and sulphur.

The specific gravity of the solution rises only slightly above 1.00, and the settling of the solids is so rapid and complete as to do away with the necessity for any filtration except for the final removal of moisture from the settled precipitates themselves.

The presence of carbon dioxide in the gases to be treated causes the formation of hydrogen sulphide in large proportions, and for this reason it was found advisable to absorb the sulphur dioxide out of the gas in water and to treat the resulting solution with finely powdered barium sulphide in the dry state. The action is rapid and complete, and, as before stated, the precipitates settle rapidly, allowing the clear overflow liquor to be immediately used again for absorption.

PROCEDURE IN "WET PROCESS."

The general method of procedure in the "wet process" is as follows:

The gas containing sulphur dioxide is taken from its source and thoroughly cooled and scrubbed as in the "dry process." It is then passed through three absorbing towers in series. The towers are filled with hollow tiles especially designed, and present a large absorption surface; the gas passes through slowly and builds up little pressure. The absorbing liquor enters at the top of the last tower and comes down in series through the three towers so that the final flow of the liquor is through the incoming gas of highest sulphur dioxide concentration.

The liquor carrying the sulphur dioxide in solution enters the suction side of a centrifugal pump. Part of the discharge of this pump is taken off and enters the top of a vertical cone-shaped incorporator, into which also the finely powdered barium sulphide is admitted by means of a closely adjusted screw feed. The incorporator discharges into the inlet of a small contrifugal pump, which in turn discharges into the inlet of the large pump and is there mixed with the main stream of liquor from the absorbing towers.

The discharge from the large pump passes through a system of cone-bottomed agitating tanks and through a system of settling tanks in series. The overflow from the last settling tank returns to the absorbing towers. Each settling tank is designed to hold a day's run of solids (sulphur and barium sulphite). The tanks have filter bottoms and draw-off outlets on the sides. One tank is cut out of the system each day, and the supernatent liquor, after it has settled to the lowest point, is drawn off. The filter pump is then started and the precipitate is filtered. When filtration is completed a discharge door in the bottom of the tank is opened and an excavator discharges the filtered precipitates on a conveyor belt below which carries them to the dryer.

The drying is done with waste gases and at a low temperature, and the dried precipitates then go to a sulphur vaporizer which is built on the order of a muffle roaster with cast-iron floors. In this device the sulphur is volatilized and removed by means of a current of sulphur dioxide gas circulated through the vaporizer, then through a condenser, and back again through the vaporizer. The sulphur is condensed as a liquid, or flowers of sulphur may be obtained by varying the method of condensation of the vapor.

The solid material discharged from the base of the vaporizer consists of barium sulphite. At a temperature of 650° C. this is easily reduced to barium sulphide by treatment with hydrocarbon gas or carbon. The reaction is greatly accelerated by using finely crushed materials and by elevating the temperature above the point mentioned.

The reducer consists of a muffle roaster, the temperature on the floors of which is kept above 650° C. by direct firing through the muffles. The reducer may be operated under down draft, and hydrocarbon gas may be circulated over the floors on which the heated barium sulphide is rabbled. Coke or coal may be mixed with the barium sulphite and the reduction attained in this manner.

Heavy fuel oil may be mixed with the barium sulphite, and the reducer operated under down draft. If this is done, the reduction is performed partly by the residual carbon from the oil and partly by the vapor and gas given off by the decomposition of the oil. Any residual gas emerging from the lower floor may be burned directly in the muffles and the total oxygen combining power of the oil approximated without the use of a gas producer.

The "wet process" is applicable to any concentration of sulphur dioxide gas, as the excess oxygen is not absorbed by the water, and reducing fuel, which is the main item as to cost, is used in direct proportion to the sulphur produced.

APPLICATION OF "WET PROCESS."

The absorbing power of water varies almost directly with the concentration of sulphur dioxide in the gas. For this reason a highly concentrated gas is cheaper to handle, as the production of sulphur increases for the same absorption and circulation costs for power, labor, and installation.

The Thiogen process in its present form, as described above, has not as yet (May, 1914) been applied on a commercial scale. Experiments with an earlier form of the process were performed on a large scale at the smelter at Campo Seco, Cal.

PROPOSED PROCESSES FOR THE REMOVAL OF SULPHUR DIOXIDE.

USE OF LIME IN PRESENCE OF WATER.

If gases containing sulphur dioxide be brought into intimate contact with lime in the presence of water, the sulphur dioxide unites with the lime to form calcium sulphite. This can be accomplished by passing the gases containing the dioxide through towers filled with crushed limestone or with brush-wood layers, heavily sprinkled with lime, or through towers against a descending spray of milk of lime. According to Richards,[a] the Ashio copper smelter in Japan employs this method to neutralize sulphur dioxide in its smelter smoke. All of the sulphur trioxide is destroyed and the sulphur dioxide reduced from 0.8 to 0.5 per cent. However, the mechanical details of the process for the handling of enormous quantities of gas have never been worked out, nor has it been stated how the product, calcium sulphite, could be used locally. This method has been applied on a small scale at zinc smelters in Silesia, Germany, but holds forth no promise under present conditions in this country.

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[a] Richards, J. W., The Japan excursion of the American Institute of Mining Engineers: Met. and Chem. Eng., vol. 10, January, 1912, p. 21.

POSSIBILITY OF ABSORPTION IN WATER.

It has been suggested that sulphur dioxide may be removed by absorption in water. It is a well-known fact that the concentrated gas is highly soluble in water. At 32° F. and at standard atmospheric pressure 1 volume of water will dissolve 79.8 volumes of sulphur dioxide gas, which is equivalent to 1 cubic foot of water, or 62.5 pounds, dissolving 14.25 pounds of sulphur dioxide. The solubility decreases rapidly with rising temperature, and at 104° F. and standard pressure only 18.18 volumes of sulphur dioxide is dissolved by 1 volume of water. These data refer to pure sulphur dioxide gas.

PROPORTION OF SULPHUR DIOXIDE DISSOLVED BY WATER.

The proportion of sulphur dioxide gas dissolved by water from a mixture of gases, such as smelter smoke, depends on the partial pressure of the sulphur dioxide gas, which in turn depends on the proportion of the unit volume it occupies. As an example, if smelter smoke containing 3 per cent of sulphur dioxide (which is high) be submitted to the action of water at 32° F. and the atmospheric pressure is 650 mm., 1 volume of water will dissolve only 2.04 volumes of sulphur dioxide. If the smelter smoke contain only 0.75 per cent of sulphur dioxide, the water will dissolve only 0.56 volume of the gas. In actual experiments made by W. C. Ebaugh in Salt Lake City, it was found that smelter smoke containing about 1.5 per cent of sulphur dioxide, when submitted to the action of water at 95° to 102° F., gave rise to solutions containing 0.33 volume of sulphur dioxide per volume of water.

It is therefore plainly evident that not much may be expected from the endeavor to remove sulphur dioxide from ordinary metallurgical smoke by spraying it with water or passing it through water in in some manner. However, Ingalls[a] states this method is perfectly feasible for gases containing a considerable percentage of sulphur dioxide (4 per cent), and that it has been applied in practice in Silesia, Germany, at certain zinc smelters. The smoke would have to be submitted to a large number of consecutive washings, and the quantity of water necessary would be excessive.

REMOVAL OF SULPHUR DIOXIDE BY REFRIGERATING SMOKE.

Another suggestion is to freeze out the sulphur dioxide from smelter smoke by submitting the smoke to refrigeration. One proposal of this kind suggests that the smoke be submitted to such a low temperature that the sulphur dioxide will be precipitated either as liquid sulphur dioxide or as frozen dioxide. The impossibility of this plan has already been discussed under the properties of sulphur dioxide. A second proposal is to refrigerate the smoke in a manner similar to that employed at a few iron blast furnaces[a] for drying of air used in the furnaces. The air is passed through refrigerating chambers, where it deposits all moisture in excess of that which it can carry at the freezing point. If smelter smoke containing sulphur dioxide and water vapor is refrigerated, the product is ice that contains a certain proportion of a hydrate of sulphur dioxide. With water highly charged with sulphur dioxide this crystallohydrate of sulphur dioxide and water, of the composition SO_2+7H_2O, freezes out first. It contains 33.7 per cent sulphur dioxide and 66.3 per cent ice. If this ice containing the sulphur dioxide be melted and the resultant acid water heated in a vacuum, sulphur dioxide will be liberated in a concentrated form, which can be readily liquefied. The relation between sulphur dioxide and water, in reference to pressure, temperature, and concentration has been fully worked out by Roozeboom.[b] The cryohydrate, or the solution of water and sulphur dioxide having the lowest freezing point contains 92.1 per cent water and 7.9 per cent- sulphur dioxide, freezing at —2.6° C. The difference between the freezing point of this cryohydrate and that of water is too small to permit, in a commercial operation, the obtaining of a concentrated liquid by the freezing of dilute solutions of sulphur dioxide and water.

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[a] Ingalls, W. R., The metallurgy of zinc and cadmium, 1905, p. 166.

COMMERCIAL APPLICATION OF REFRIGERATION PROCESS DOUBTFUL.

No data of experiments on a commercial scale are available as to the concentration of sulphur dioxide attainable in the ice formed from ordinary smelter smoke, but it seems questionable that this concentration can be high, when the small amount of sulphur dioxide actually dissolved by water from smelter smoke is considered. One of the difficulties of the process would be the disposal of the liquid sulphur dioxide. Manifestly it is impossible to discharge it into streams or onto the land, and unless a ready market be available it would seem that the process could not be applied commercially. The principles on which the process is based seem sound theoretically, but data on such necessary details as the amount of water needed and the attainable concentration of sulphur dioxide in the ice are entirely lacking.

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[a] The Gayley process.

[b] Roozeboom, H. W. B., Sur l'hydrate de l'acide sulfureux; Rec. trav. chim. Pays-Bas, vol. 3, 1994, p. 29; Sur la dissociation des hydrates de l'acide sulfureux, du chlore et du brome: vol. 4, 1885, p. 65; Sur les points triples et multiples, envisagés comme points de transition: vol. 6, 1887, p. 319; Studien über chemisches Gleichgewicht: Zeitschr. physikal. Chem., vol. 2, 1888, p. 450; quoted by Nernst, —, Theoretic chemistry, 1911, p. 622, and by Findlay, —, The phase rule, 1906, p. 169, and by Landolt and Börnstein, Physikalisch-chemische Tabellen, 1912, p. 493.

USE OF FINELY GROUND SLAG OR SLAG WOOL.

Westby and Sorenson have proposed a process whereby hot smelter smoke containing appreciable amounts of sulphur dioxide is brought into intimate contact with finely ground ferruginous slag, such as is produced in copper or lead smelting, resulting in the formation of iron sulphur salts, which dissolve in water, with the consequent elimination of the sulphur dioxide from the smoke stream. The slag, which is a waste product of the smelting, may be used either in a finely ground form or as slag wool, a product produced by blowing air or steam against a slag stream so that it is spun out into fine threads. In the experimental work the smoke stream from a roasting furnace was admitted into a wooden tower set in a water seal. The tower was provided with numerous sloping baffle plates. The water and finely ground slag charged from a hopper at the top spread over these baffles or shelves, the slag accumulating on the shelves, thus exposing large areas of material. The ascending smoke stream came into contact with this slag and the descending water sprays, resulting in the rapid chemical action of the sulphur dioxide on the iron silicates of the slag, with the formation of such salts as ferrous and ferric sulphites, ferrous sulphate, thionate of iron, and sulphur salts of other bases, such as lime. Some of these salts are dissolved by the water, whereas others remain as insoluble constituents with the undissolved slag residue.

RESULTS EXPECTED OF PROCESS.

In practice the solutions were to be discharged from the water seal and run into ponds, where oxidation of the iron sulphur salts would take place, precipitating iron oxides and basic sulphates, or they were to be run over waste lands where they would be absorbed by the soil and any acids present be neutralized by the lime constituents of that soil, so that the ground water would not be rendered impure. The slag used usually contained 0.2 to 0.4 per cent copper. In the slag wool this copper was in the form of oxide. In the finely ground slag the copper was partly in the form of oxide and partly in the form of sulphide. That part present as oxide of copper would dissolve and pass out with the solutions, whereas that present as sulphide would remain with the slag residues in the tower. The dissolved copper was to be precipitated from the solutions by heating them with steam, which would cause the thionates of iron present to react with the soluble copper salts to form insoluble copper salts. The slag residues from the tower containing the insoluble copper, owing to their relatively small bulk, might contain sufficient copper to warrant their resmelting in reverberatory furnaces. If the location of the smelting plant should justify it, such by-products as iron sulphate and zinc sulphate might be recovered from the solutions.

The process has been tried only experimentally, and, under favorable conditions, as much as 85 to 90 per cent of the sulphur dioxide is said to be removable from the smoke stream. It would seem that the process is applicable only to gases comparatively high in sulphur dioxide, containing, say, more than 4 per cent, and to comparatively small volumes of smoke. Its application to great volumes of gas comparatively low in sulphur dioxide seems hardly feasible.

DILUTION OF THE SULPHUR DIOXIDE BY AIR.

The dilution of smelter smoke by air to such a point that the concentration of sulphur dioxide is low enough to be considered harmless appears at present the only universally applicable remedy for the sulphur dioxide nuisance. Although this fact is generally known and its significance is fully realized by metallurgists, it would seem that in a number of instances the possibilities of the method have not been fully utilized.

USE OF HIGH CHIMNEYS NOT SATISFACTORY.

It was formerly rather generally believed that if smelter smoke was discharged from high chimneys natural diffusion would so act on the smoke stream that when the gases reached the ground, the sulphur-dioxide would be so diluted as to be practically harmless. However, experience, both in this country and in Europe, does not warrant this assumption. It seems rather that the diffusive action is comparatively slow, and that frequently masses of smelter smoke are wafted to the surface, still in such concentrated condition as to be harmful. High stacks tend to protect the immediate vicinity around the smelting plant, and may cause a milder but rather widespread damage at some distance from the plant. Of course in considering whether damage will be done, it is necessary to know the initial concentration of the sulphur dioxide as it issues from the stack. If this be low, say less than 1.50 per cent, the great probabilitiy is that with a high stack diffusion will be sufficient to render the discharge practically harmless. If, however, the concentration of sulphur dioxide is greater than this, there is a possible chance of damage being done. Court decisions in California and Utah have set 0.75 per cent as the maximum permissible sulphur dioxide content of smoke streams discharged by smelter stacks. This figure is not based on scientific experiment, but was adopted as a general figure to conform to certain opinions held at the time the decisions were rendered.

USE OF FANS GIVES PROMISE.

Recently the feasibility of diluting the smoke stream with air by the use of large fans and of then discharging the diluted smoke from a number of low stacks rather than from one high stack has received considerable attention. One case in point is the manner of discharging smelter smoke in use in the plant at Kennett, Cal., which has been described. There fans are in use for the special purpose of diluting the smoke stream with air. The copper-smelting plant at Coram, Cal., before its shutdown, also employed fans for the dilution of the smoke stream.

It has recently been stated that the Ashio Copper Co., in Japan, will use a similar system.[a] At this plant the problem of fume damage is important, and the proposed method of the disposal of the smelter smoke was adopted after careful study of conditions the world over. It is proposed to proceed as follows: (1) To cool the smoke stream by radiation to 100° C. or less; (2) to settle the fine dust by passing the smoke stream through a chamber 210 feet long at a velocity of 3 feet per second, the dust chamber to be hung with bamboo strips instead of wires; (3) to dilute the smoke stream to 0.3 per cent sulphur dioxide or less with fresh air by means of fans, and to scatter the diluted smoke by the use of three or four stacks rather than one stack.

LEGAL ASPECT OF THE SMELTER-SMOKE PROBLEM.

Although the smelter-smoke problem has been in the courts for a long time in a minor way, it has been prominent only in the past 10 years. During that time, however, and at present it is attracting nation-wide attention. The cases that have been before the courts may be classified geographically under five heads, as follows: (1) Utah, centering about Salt Lake City; (2) southeastern Tennessee, near the Georgia and North Carolina boundary line; (3) Montana, centering about Butte and Anaconda; (4) Shasta County, Cal.; (5) near San Francisco, Cal., in Contra Costa and Solano counties. There are cases of minor importance in other sections of the country, but those in the five localities mentioned serve well to illustrate the legal aspects of the smelter-smoke problem.

UTAH CASES.

The country about Salt Lake City contains rich agricultural lands, particularly along the Jordan River. Many years ago, during the early mineral development of Utah, various smelters were erected in the midst of this section, and grew into large plants. There was always more or less trouble between individual farmers and certain smelting companies which was adjusted by the payment of damages. The early smelters treated lead ores. Later, copper smelters were built and the complaints about smoke became more insistent. In 1901, and immediately before that, certain farmers obtained heavy damages from one of the smelting companies on account of injury to crops. Damage did not cease, and in 1904, after lengthy conferences between the farmers of the vicinity and various smelting corporations, which led to no results, suits were started against the smelting companies.

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[a] For description of this system see editorial, Dust chambers at the Ashio smelter, Japan: Min. Sci. Press, vol. 109, January, 1914, p. 13.

In the case of James Godfrey et al., representing about 400 farmers, versus the American Smelting & Refining Co., the United States Smelting Co., the Bingham Consolidated Co., and the Utah Consolidated Co., the Federal court issued an injunction in November, 1906, designed to prevent any of the companies from treating ore containing more than 10 per cent sulphur in the natural state, or permitting the escape of arsenic from the stacks. This decision was upheld on appeal. The Bingham Consolidated Co. and the Utah Consolidated Co. then abandoned their smelters. The American Smelting & Refining Co., by permission of the court, entered into contract with the farmers to rebuild its flue and dust-chamber system and to add a bag house for the filtering of blast-furnace gases at its Murray plant, and was permitted to resume operations. The United States Smelting Co., at Midvale, also entered into an agreement with the farmers by permission of the court, and built elaborate bag houses for the filtering of gases from all of its furnaces in the lead department. The agreements reached called for (*a*) the removal of all free sulphuric acid in the smelter smoke; (*b*) the removal of all solids from the smoke by means of a bag house; (*c*) that after the removal of solids and sulphuric acid, the gases discharged into the atmosphere should contain not more than 0.75 per cent of sulphur dioxide; (*d*) that such sulphur dioxide as escaped should not damage crops and live stock or cause discomfort or inconvenience to persons. Under this agreement the United States Smelting Co. resumed operations in its lead department in April, 1908. The copper department of the smelter has not as yet resumed operations, although granted permission to do so. All of the copper smelting of the Salt Lake Valley is now done at Garfield and at Tooele, westward from Salt Lake City, in a region where smoke can seemingly do less damage than in the immediate vicinity of Salt Lake City. Although at present there is not much complaint about smoke near Salt Lake City, it is doubtful whether the matter is permanently settled.

TENNESSEE CASES.

The copper smelters of the Tennessee Copper Co. and of the Ducktown Copper, Sulphur & Iron Co., are at Copperhill and Isabella, respectively, in the extreme southeastern corner of Tennessee, less than a mile from Georgia and only a few miles from North Carolina. The country is beautiful, mountainous, and heavily wooded, although in places it is well cultivated. The prevailing winds for the greater part of the year blow to the south. The country immediately surrounding Copperhill and Ducktown is entirely denuded and bare, the damage having been wrought largely during earlier days when heap roasting was extensively practiced. There was always more or less trouble in the district, particularly in northern Georgia. In May, 1907, the State of Georgia obtained from the Supreme Court of the United States the permission to call for a permanent injunction restraining the Tennessee Copper Co. and the Ducktown Copper, Sulphur & Iron Co. from operating their plants. This injunction, however, was not put into effect because the companies were constructing sulphuric acid plants for the treatment of the smelter smoke. This method was put into successful operation, and former difficulties have largely disappeared. However, the sulphuric acid plants are not large enough to fully accommodate the tonnage that might be smelted, and the Tennessee Copper Co. is forced during the active crop-growing season to curtail its tonnage in order not to produce damaging smoke. That some damage was perhaps still being done was reflected in a news statement dated October 25, 1913, that the Ducktown mine owners had contracted with the State of Georgia to pay annually $16,500 to Georgia farmers.

MONTANA CASES.

The great smelting interests of the State of Montana center about Anaconda, 29 miles west of Butte, Great Falls, 120 miles northeast of Butte, and East Helena, 50 miles northeast of Butte. The Montana smelter cases relate solely to the Anaconda location. The tremendous devastation wrought by sulphurous gases in the immediate vicinity of Butte, in the early days of smelting operations is well known. Damage was also done in the vicinity of Anaconda in the Deer Lodge Valley before the great copper smelter of the Anaconda Copper Mining Co. moved to its present site. It was claimed that this last plant also caused damage, and this complaint culminated in the suit of F. J. Bliss versus the Anaconda Copper Mining Co. and the Washoe Copper Mining Co.

A perpetual injunction was sought on account of damage by fume. In the first decision of the Federal court in January, 1909, the injunction was refused, although Bliss was awarded nominal damages, the court finding that some damage had been done by fume in the immediate vicinity of the smelter, but that no damage had resulted from sulphur dioxide. It decreed a rehearing of the case to determine whether the arsenic in the fume could be reduced in quantity. In May, 1911, the United States Circuit Court of Appeals confirmed the first decision, the records in the case being the most voluminous known in a suit of this kind, comprising more than 25,000 pages. Appeal was then had to the Supreme Court of the United States, which, in November, 1913, finally dismissed the suit. The United States Government, in 1910, also entered suit against the Anaconda Copper Mining Co. on account of damage by smoke to the national forests. This suit was not proceeded with, but a commission to investigate the matter was appointed in 1911, by agreement between the Department of Justice and the attorneys for the Anaconda Co. This commission consists of J. A. Holmes, Director of the Bureau of Mines, L. D. Ricketts, an eminent metallurgical engineer, and John Hays Hammond, the well-known mining engineer. Chemists and engineers employed by this commission are working in conjunction with employees of the Anaconda Copper Mining Co., seeking a solution of the smelter-smoke problem, and it is understood that good progress is being made.

SHASTA COUNTY, CAL., CASES.

The copper-smelting plants of Shasta County, Cal., are situated in the mountains immediately above the head of the great Sacramento Valley. The country immediately surrounding the various smelting plants has been entirely denuded by former operations. The section can not be called an agricultural one, but the Sacramento Valley lying below it is fertile and highly cultivated, and the complaint against the smelters arises chiefly from the farmers of that vicinity. The first plant to be closed down was that of the Mountain Copper Co., which was enjoined by the United States Government for doing smoke damage to forest reserves. Suit was brought by the Shasta County Farmers' Protective Association against the Mammoth Copper Co. at Kennett, a subsidiary concern of the United States Smelting, Refining & Mining Co., and the Balaklala Consolidated Copper Co. at Coram, seeking to enjoin these companies from operating. The case was brought to trial and the United States circuit court of appeals, in February, 1910, arranged for agreements between smelters and farmers whereby the smelters were permitted to operate provided that they fulfilled conditions similar to those outlined for the United States Smelting Co. in Utah. In confirmation with this agreement the Mammoth Copper Co. put in a bag house and diluting fans, and the Balaklala Co. installed the electrical precipitation process and diluting fans, and the companies continued to operate. Some time later the Balaklala Consolidated

Copper Co. decided to close its plant as damage to crops was still found to continue. The Mammoth Copper Co. is still operating, being able to satisfy the conditions imposed. The difficulty, however, is by no means settled, and it is stated (December, 1913) that the Farmers' Protective Association expects to proceed again against the smelting company. In November, 1913, the governor of California appointed a commission consisting of the secretary of the State board of health, the horticultural commissioner, and the State veterinarian to investigate the smelter fumes. It is hoped that this commission will assist in solving the problem. In 1910 the Government suggested to the officials of the Bully Hill Smelter, in this vicinity, that they begin vigorous experimental work for the abatement of the smoke nuisance on account of damage then being done to forest reserves. The company thereupon closed its plant in order to be better able to work out its problems and devise new methods.

Since the Government is interested in this section of California by virtue of the forest reserves, the secretary of the commission that has charge of the Montana smelter-smoke cases, J. A. Holmes, has several times visited this district.

CASES NEAR SAN FRANCISCO.

A number of smelting and acid plants are located in Contra Costa County, on San Pablo Bay, near San Francisco. These are the Selby Smelting & Lead Co., the General Chemical Co., and the Mountain Copper Co. During the past 9 or 10 years there has been difficulty in this district owing to claims of damage by farmers in Contra Costa and Solano Counties. Suits in the local and State courts were frequent. Statutes seeking to regulate the disposal of smelter smoke were passed by county supervisors. The situation is complicated by the existence in the vicinity of other manufacturing plants, which give rise to odors and gases of a disagreeable nature. It seems, however, that the smelting plants receive most of the blame, possibly unjustly.

In 1908 the Selby Smelting & Lead Co. was enjoined by the superior court of the State from operating the smelter during the period from March 15 to October 15 of each year. In this time the trade winds carry the smoke westward across the Karquines Straight to Benicia and the adjoining farm lands. Since then the company has installed one bag house for its blast-furnace gases and another for those of the Dwight-Lloyd sintering machines and has put in the electrical-precipitation process for the condensation of acid fume from the parting kettles. It has also diverted ores high in sulphur to the General Chemical Co.'s plant at Bay Point, where the sulphur is utilized in the manufacture of sulphuric acid, the roasted ore being returned to the Selby smelter. The introduction of these improvements has much ameliorated the conditions that formerly existed. The complaints from the farmers, however, have not entirely ceased, and in March, 1913, the county supervisors of Solano County and the officials of the Selby Smelting & Lead Co. agreed on the appointment of a commission of three experts to investigate and decide future questions of annoyance and nuisance resulting from the escape of smelter fumes. This commission consists of J. A. Holmes, Director of the Bureau of Mines; E. C. Franklin, of the department of chemistry, Stanford University; and Ralph A. Gould, a chemical engineer of San Francisco. It has conducted a long and thorough investigation and in November, 1914, submitted its report, which is to be published as soon as practicable.

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Bulletin 85

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, Director

ANALYSES OF MINE AND CAR SAMPLES OF COAL COLLECTED IN THE FISCAL YEARS 1911 TO 1913

BY

ARNO C. FIELDNER, HOWARD I. SMITH, ALBERT H. FAY,
and SAMUEL SANFORD

WASHINGTON
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ANALYSES OF MINE AND CAR SAMPLES OF COAL COLLECTED IN THE FISCAL YEARS 1911 TO 1913.

By ARNO C. FIELDNER, HOWARD I. SMITH, ALBERT H. FAY, and SAMUEL SANFORD.

SAMPLING AND ANALYTICAL METHODS.

By ARNO C. FIELDNER.

INTRODUCTION.

In connection with investigations relating to fuels belonging to or for the use of the United States Government, the causes of accidents in coal mines, the geologic relations of coal beds, and the character of the coal and the value of the fuel in the public lands, many mine samples of coal are analyzed each year in the laboratories of the Bureau of Mines. The systematic collection and analysis of coal samples in connection with investigations of fuels was begun in 1904 at the Government fuel-testing plant at the Louisiana Purchase Exposition, and was under the direction of the United States Geological Survey for six years. Descriptions of the coal samples collected in connection with these investigations between the beginning of this work, July 1, 1904, and the transfer of the fuel-testing and mine-accident investigations to the Bureau of Mines, July 1, 1910, were compiled and published in Bureau of Mines Bulletin 22.[a]

The present bulletin presents analyses and descriptions of samples collected during the fiscal years 1911 to 1913. In order that the material in this bulletin may be used to supplement that presented in Bulletin 22, the same plan of geographical classification has been followed, the analyses and descriptions of the samples being grouped in alphabetical order according to the State, county, and town near which the mines or prospects sampled are situated.

---

[a] Lord, N. W., and others: Analyses of coals in the United States, with descriptions of mine and field samples collected between July 1, 1904, and June 30, 1910. Bull. 22, Bureau of Mines, 1912. 1,200 pp. (in two parts).

ACKNOWLEDGMENTS.

Special acknowledgment is due to G. O. Smith, Director of the United States Geological Survey, and to M. R. Campbell, geologist in charge of the economic geology of fuels, for furnishing field descriptions of those samples collected by the geologists of the survey, and for revising the nomenclature of coal beds and geologic formations.

The mine samples from Missouri were collected under the direction of H. A. Buehler, director of the Missouri Geological Survey.

The collection of samples by the fuel-inspection service of the Bureau of Mines was under the direction of G. S. Pope, engineer. Samples designated by the letter W preceding the laboratory number were analyzed in the Washington laboratory of the bureau under the direction of J. D. Davis, chemist.

The collection of mine samples and the gathering of mine data by the field engineers of the bureau were done under the direction of G. S. Rice, chief mining engineer of the bureau, who has also reviewed the descriptions and comments on the methods of field sampling. The descriptions of the mines and the sections of the coal beds were compiled by H. I. Smith and A. H. Fay, mining engineers, and Samuel Sanford, engineer. The chemical analyses were made under the direction of A. C. Fieldner, chemist, who was assisted by D. I. Brown, assistant chemist; C. K. Glycart, A. E. Hall, Harold Isenberg, W. A. Mueller, G. G. Oberfell, F. D. Osgood, G. E. Postma, W. A. Selvig, C. A. Taylor, G. G. Taylor, G. E. Webster, A. M. Wedd, and H. G. Woodward, junior chemists; W. E. Surbled, laboratory assistant; Max Hecht and Morris Block, laboratory helpers. The junior chemists and laboratory helpers were engaged on coal analyses for varying lengths of time, ranging from one month to two years. The computing and the tabulating of the analyses at the Pittsburgh laboratory were under the direction of Lauson Stone, engineer.

COLLECTION OF SAMPLES.

Samples designated by the letter A in the table of analyses were collected by representatives of the Bureau of Mines, in connection with the investigation of fuels belonging to, or for the use of, the Government; and the investigation of accidents in coal mines. Under this designation are also included a number of samples collected by the Missouri Geological Survey in cooperation with the Bureau of Mines.

The samples designated by the letter B in the table of analyses were collected by the United States Geological Survey in connection with the study of the geologic relations of coal beds and the classification of public lands.

METHOD OF MINE SAMPLING FOLLOWED BY BUREAU OF MINES.

The method of collecting mine samples that is practiced by the Bureau of Mines has been described in detail in a previous publication.[a] It involves selecting a representative face of the bed to be sampled; cleaning the face; making a cut across it from roof to floor, and rejecting or including impurities in this cut according to a definite plan as they are included or excluded in mining operations; reducing this gross sample, by crushing and quartering, to about 3 pounds; and immediately sealing the 3-pound sample in an air-tight container for shipment to the laboratory.

COLLECTION OF CAR SAMPLES.

The carload lots of coal shipped to Pittsburgh for test were sampled by taking definite quantities of coal at regular intervals from a car as it was unloaded, and by reducing to convenient size (about 50 pounds) the gross sample thus obtained.

METHOD OF SAMPLING FOLLOWED BY THE UNITED STATES GEOLOGICAL SURVEY.

In collecting mine samples the Geological Survey follows essentially the same method of sampling as that used by the Bureau of Mines. However, in sampling outcrops and prospect holes or country banks when mining is not in progress, the geologist can not imitate the miner in rejecting or including impurities in the sample, and hence the sample from the cut across the bed includes all partings or binders less than three-eighths of an inch thick and every concretion or "sulphur ball" having a maximum diameter of less than 2 inches and a thickness of less than one-half inch. All other impurities in the bed are excluded from the sample. Obviously an arbitrary and uniform system of rejecting impurities is necessary for sampling outcrops, prospects, and undeveloped mines.

RELATION OF MINE SAMPLES TO COMMERCIAL SHIPMENTS.

In making statements, on the basis of the analyses of mine samples, in regard to the quality of coal shipped from a mine due allowance must be made for the larger proportion of impurities that may be included in the commercial operation of the mine. It is difficult to take a mine sample in which impurities are rejected in exactly the same manner as is done by the miner. The practice of different miners will vary, especially if rigid inspection at the tipple is not enforced. In some mines, for instance, where the coal bed has friable partings or has a soft, flaky roof or floor, the inclusion of some foreign matter is unavoidable. Hence the analysis of the mine sample usually indicates a better grade of coal, as regards ash content and heating value, than the actual commercial shipments, and for this reason the mine sample should be considered as representing the coal that can only be produced under the most favorable conditions of mining and preparation.

---

[a] Holmes, J. A., The sampling of coal in the mine: Technical Paper 1, Bureau of Mines, 1911, 18 pp.

In commercial shipments that are sampled at their destination the moisture content may be either more or less than that in the mine samples, the relative proportions depending on the amount of bed moisture, the size of the coal, and the weather conditions during transit.

Coals containing 5 per cent or more of moisture tend to lose moisture while in transit. Slack coal usually contains more moisture than the mine sample. Low-moisture coals shipped in open cars may gain or lose moisture, depending on weather conditions.[a]

The calorific value, referred to moisture-free and ash-free coal, of samples taken from shipments at destination, tends to be slightly lower than that of the fresh mine samples from the same mine. The deterioration is caused mainly by the freshly exposed surfaces of coal absorbing oxygen from the air. The rate of deterioration varies with the different types of coal and depends on a number of factors, chief of which are: (1) Size of coal, (2) proportion of surface exposed to circulating air, (3) duration of exposure, (4) temperature and humidity.

It is therefore difficult to assign any definite values for deterioration of coal while in transit.[b] A number of mine and car samples tested by the United States Geological Survey and the Bureau of Mines showed the following average losses:

Average loss in moisture-free and ash-free calorific value of car sample as compared with that of mine sample.

Kind of coal.	Per cent.
Semibituminous, New River and Pocahontas	0.1
Bituminous, Appalachian field	.3
Bituminous, Illinois, Indiana, and Missouri	.8
Subbituminous and lignite	1.3

---

[a] For a comparison of the moisture, ash, and sulphur contents of mine and car samples tested at the United States fuel-testing plant, St. Louis, Mo., see Burrows, J. S., The importance of uniform and systematic coal-mine sampling: U. S. Geol. Survey Bull. 316, 1907, pp. 490-560. For a discussion of moisture in coal deliveries, see Pope, G. S., Sampling coal deliveries: Bull. 63, Bureau of Mines, 1913, p. 12.

[b] For data on deterioration of coal see Porter, H. C., and Ovitz, F. K., Deterioration and spontaneous combustion of coal in storage, a preliminary report: Tech. Paper 16, Bureau of Mines, 1912, 14 pp.; The escape of gas from coal: Tech. Paper 2, Bureau of Mines, 1911, 14 pp., 1 fig.; Parr, S. W., and Wheeler, W. F., The weathering of coal: University of Illinois Eng. Exp. Station Bull. 38, 1909.

ANALYTICAL METHODS.

The methods of analysis and the details of apparatus used by the Bureau of Mines in analyzing coal are fully described in Bulletin 22 [a] and Technical Paper 8 [b] of the Bureau of Mines. They are briefly summarized here, certain changes in minor details, which are fully discussed in Technical Paper 76,[c] being described.

PREPARATION OF LABORATORY SAMPLES.

Immediately after the sealed 3-pound can in which the sample is received at the laboratory has been opened, the contents are transferred to a weighed sheet-metal pan, spread out to a depth of 1 inch, and at once weighed. The pan containing the sample is placed in a large drying oven in which a temperature of 30° to 35° C. is maintained. Through this oven a current of warm air is made to flow by means of an ordinary desk fan mounted on top of the oven, and the sample is dried until the loss in weight between two successive weighings, made 6 to 12 hours apart, does not exceed 0.5 per cent. The total loss of weight is reported as "air-drying loss."

Immediately after the last weighing has been made the entire sample is quickly pulverized to 10-mesh size by being passed through a roll crusher. After having been mixed, the 10-mesh material is reduced with a riffle sampler to 500 grams. This 500-gram part is at once transferred to the porcelain jar of an Abbe ball mill, which is sealed air-tight and rotated at the rate of 60 revolutions per minute until the coal is pulverized to 60-mesh size.

This 60-mesh coal is emptied on a one-half-inch screen, which is vigorously shaken to detach the coal from the pebbles. The sample (approximately 500 grams) is then reduced to the final laboratory sample of approximately 60 grams by successive halvings with a small riffle sampler. The final sample is put through the 60-mesh sieve and at once transferred to a 4-ounce wide-mouthed bottle, which is securely closed with a well-fitting rubber stopper. Any particles of coarse material remaining on the sieve are carefully rubbed down to 60-mesh size on a bucking board and are thoroughly mixed with the sample in the bottle.

This air-dried sample is used in all analytical determinations.

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[a] Lord, N. W., and others, Analyses of coals in the United States, with descriptions of mine and field samples collected between July 1, 1904, and June 30, 1910: Bull. 22, Bureau of Mines, 1913, pp. 10-26.

[b] Stanton, F. M., and Fieldner, A. C., Methods of analyzing coal and coke, Tech. Paper 8, Bureau of Mines, 1913, 42 pp.

[c] Fieldner, A. C., Notes on the sampling and analysis of coal: Tech. Paper 76, Bureau of Mines, 1914, pp. 14-58.

DETERMINATION OF MOISTURE.

The residual moisture in the air-dried sample is determined by heating 1 gram in a shallow porcelain capsule, seven-eighths of an inch deep and 1¾ inches in diameter, for one hour at 105° C. in a constant-temperature oven through which a current of dry preheated air is rapidly passing. The air is dried by being passed through concentrated sulphuric acid. The covered capsule is cooled in a desiccator over sulphuric acid and then weighed. The loss in weight is called "moisture at 105°" in the air-dried coal.

ASH DETERMINATION.

The same sample is used for determining ash as was previously used for determining moisture. A porcelain capsule containing the sample is placed in a cool muffle and the temperature is gradually raised to approximately 750° C. The ignition in the muffle is continued, with occasional stirring of the ash, until all particles of carbon have disappeared. After the capsule containing the ash has been cooled in a desiccator it is weighed and ignition is repeated until constant weight (0.0005 gram or less) has been attained.

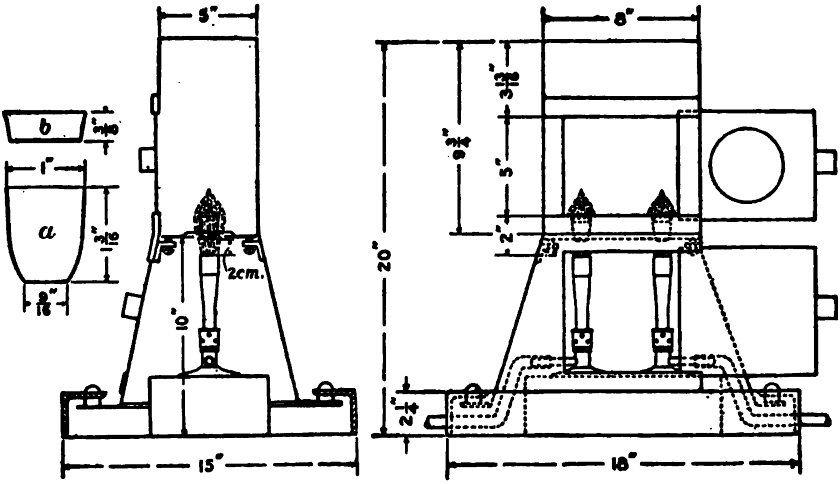

FIGURE 1.—Apparatus for determining volatile matter.

The ash content as determined by this method represents the ignited mineral residue or "uncorrected ash."

DETERMINATION OF VOLATILE MATTER.

APPARATUS.

A 1-gram sample of the fine (60-mesh) coal is weighed into a bright, well-burnished, 10-gram (10 c. c.) platinum crucible (*a*, fig. 1), with a close-fitting capsule cover, *b*. The crucible and contents are heated to a temperature of 950° C. upon a platinum or nichrome triangle for seven minutes in the full flame of a No. 3 Meker burner. The gas orifice of the burner is adjusted to give a free-burning flame from 16 to 18 cm. high. The crucible should be so placed in the triangle that its bottom is 2 cm. above the top of the burner. To protect the crucible from the effects of drafts it is put in a sheet-iron chimney of special design. The maximum temperature, as measured by a thermocouple whose hot junction is buried in the coal and in contact with the inside surface of the bottom, should be 950° C. After being heated exactly seven minutes, the crucible is cooled and weighed. The loss in weight minus the weight of moisture determined at 105° C. times 100 equals the percentage of volatile matter.

MODIFIED METHOD FOR LIGNITES AND SUBBITUMINOUS COALS.

Lignites and coals high in moisture are heated at a low temperature until all the moisture is driven out in order to avoid mechanical losses from material thrown out of the crucible by the rapid escape of steam and volatile matter.[a] This preliminary heating for four to six minutes is followed by the usual 7-minute application of the full flame.

It should be clearly understood that the volatile matter does not represent any definite compound in the coal. The method of determination is wholly conventional, and any variation in temperature or rate of heating will change the amount of volatile matter determined.

USE OF THE ELECTRIC FURNACE.

Since February 26, 1913, all volatile-matter determinations at the bureau's laboratory in Pittsburgh have been made in specially designed electric furnaces of the vertical type (fig. 2). Two of these furnaces are connected in series and are controlled by a single rheostat. A current of 8 to 9 amperes is required to maintain the temperature of each furnace at 950° C. Under these conditions the drop in potential across the two furnaces is 60 volts. The life of the heating element under daily operation has been six to eight months. Each furnace contains a platinum platinum-rhodium thermocouple that is connected to a millivoltmeter through a double-pole, double-throw switch. The position of the hot junction in each furnace is so adjusted that the direct reading of the temperature scale of the millivoltmeter is the same as the maximum temperature attained inside the crucible as determined by another standardized couple whose hot junction is in contact with the bottom of the interior of the

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[a] Lord, N. W.: Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo. U. S. Geol. Survey Bull. 323 (reprinted as Bull. 28, Bureau of Mines), pp. 6-7.

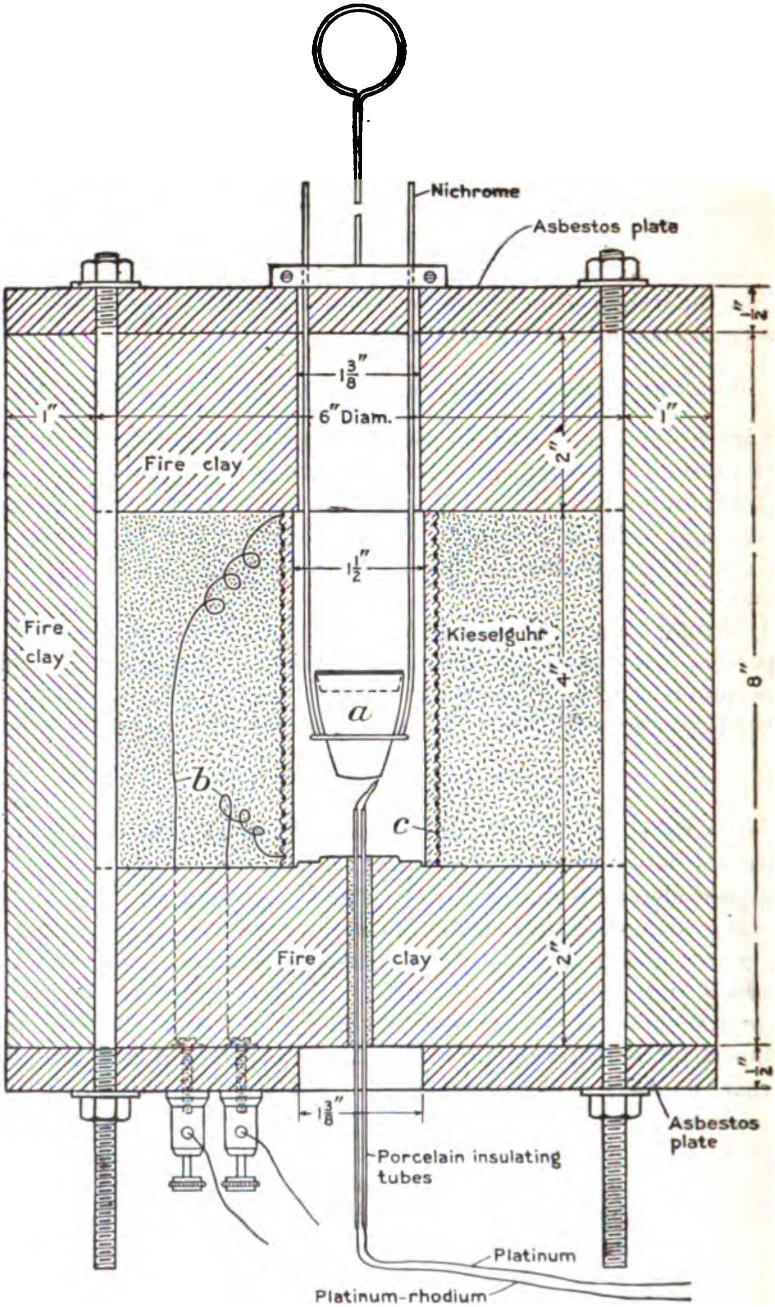

FIGURE 2.—Electric furnace for volatile-matter determination.

crucible. In making this comparison a nickel cover perforated for admitting the thermocouple leads is substituted for the usual platinum cover of the crucible. To avoid possible contamination of the thermocouple, 1 gram of ignited silica or alumina is used instead of coal.

DETERMINATION OF FIXED CARBON.

The fixed carbon content of the sample is determined by calculation—by subtracting the sum of the percentages of moisture, ash, and volatile matter from 100.

SULPHUR DETERMINATION.

Sulphur is determined by the Eschka method. A 1.3737-gram part of the air-dried coal is mixed with 6 grams of Eschka mixture in a No. 1 porcelain crucible and gradually heated in a muffle, with free access of air, until all the carbon has been consumed. The sulphur is extracted with hot water, completely oxidized to sulphate with bromine water, and precipitated and weighed as barium sulphate.

DETERMINATION OF CARBON AND HYDROGEN.

Carbon and hydrogen are determined by the usual method of combustion in a current of oxygen. A 0.2-gram part of the air-dried coal is burned in a 25-burner combustion furnace of the Glazer type or in an electrically heated furnace of the Heraeus type. Complete oxidation is insured by passing the products of combustion over red-hot copper oxide. A layer of lead chromate, following the copper oxide, removes the sulphur. The water and carbon dioxide are absorbed and weighed in calcium chloride and potassium hydroxide solution respectively. No correction is made for carbon or hydrogen from inorganic matter in the coal.

NITROGEN DETERMINATION.

Nitrogen is determined by the Kjeldahl-Gunning method. One gram of the air-dried coal is digested with 30 c. c. of concentrated sulphuric acid, 0.5 gram of metallic mercury, and 5 grams of potassium sulphate, until the carbon has been completely oxidized and all the nitrogen has been converted to ammonium sulphate. After dilution with water and precipitation of the mercury by the addition of potassium sulphide, an excess of sodium hydroxide is added and the ammonia determined by distillation.

DETERMINATION OF CALIFORIC VALUE.

The heat of combustion is determined with the bomb calorimeter. One gram of the air-dried coal is completely burned in compressed oxygen gas and the total heat evolved is absorbed in a weighed quantity of water in which the bomb is immersed. The rise in temperature of the water is measured with a thermometer that is graduated in hundredths of 1° C., and can be read, by means of a cathetometer, to 0.002° C.

Corrections are made for "radiation losses," oxidation of nitrogen to aqueous nitric acid, and oxidation of sulphur dioxide to aqueous sulphuric acid. The calorific value obtained in this manner is the total heat of combustion with water vapor condensed to liquid water at the temperature of the calorimeter, that is, 20° to 25° C.

INTERPRETATION OF ANALYTICAL RESULTS.[a]

The coal analyses reported in this bulletin are grouped in the usual manner, as follows:

1. The proximate analysis, including results of determinations of moisture, volatile matter, fixed carbon, and ash.
2. The ultimate analysis, including results of determinations of carbon, hydrogen, nitrogen, oxygen, sulphur, and ash.
3. The calorific value, heating value, or heat of combustion.

The analysis of each sample is given in three conditions, as follows: (1) As received at the laboratory, (2) computed to a moisture-free condition, and (3) computed to a moisture-free and ash-free condition.

The analysis of the sample "as received" (condition 1) represents the actual sample as received at the laboratory and, as regards a mine sample, represents the coal as it occurs in the mine. The "moisture-free" analysis (condition 2) represents the relative composition and heating value of the dry coal; this form of analysis is convenient for comparing similar coals of variable moisture content. The "moisture-free and ash-free" analysis represents approximately the relative composition and calorific value of the dry organic or combustible matter. This form of analysis is only an approximation because the ash does not have the same weight as the inorganic or incombustible matter in the coal. However, the error does not exceed 1.5 per cent in comparing coals that do not vary much in the character and amount of ash and sulphur.

PROXIMATE ANALYSIS.

The proximate analysis of coal originated in response to the industrial demand for laboratory tests of the relative amounts of certain compounds, either present in the coal or derived from it, that affect its use as a fuel. These compounds are grouped by the proximate analysis as follows:

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[a] See also, Fieldner, A. C., Notes on the sampling and analysis of coal: Tech. Paper 76, Bureau of Mines, 1914, pp. 23-57.

(1) Water or moisture; (2) mineral impurities that remain in a somewhat altered condition as ash when the coal is burned; (3) organic or combustible matter, which is approximately represented by the volatile matter and fixed carbon.

MOISTURE CONTENT.

The moisture in coal consists of (1) extraneous moisture, which comes from external sources, such as underground waters trickling over the face of the coal bed, through the coal bed, and, in places going to the dip, from the coal shot down lying in water, condensation from saturated mine air, or, as regards car samples, from atmospheric precipitation or from water used for washing the coal, and (2) inherent moisture, which is one of the products of the original vegetable matter from which coal is derived.

Analysis does not differentiate between extraneous and inherent moisture, because in air-drying coal not only is the extraneous or superficial moisture removed but also more or less of the inherent moisture, the amount depending on the fineness of the coal and the humidity of the air. The percentage loss of weight on air drying should be regarded merely as an indication of the relative tendency of different coals to lose moisture, after removal from the mine, on exposure to comparatively dry atmospheres.

The loss of weight on air drying at 30–35° C., combined with the loss of weight occurring when the air-dried sample is heated in a current of dry air at a temperature of 105° C., is considered equal to the total moisture in the coal as received at the laboratory.

ASH.

Ash is the incombustible residue left from complete combustion of the coal. It is derived from the inorganic matter in the coal, and is composed largely of compounds of silica, alumina, lime, and iron, together with smaller quantities of magnesia, titanium, and alkali compounds. The silica, alumina, and titanic oxide are derived from sand, clay, shale, and "slate"; the iron oxide mainly from iron pyrite; and the lime and magnesia from their corresponding carbonates and sulphates.

The ash-forming constituents consist of (1) "inherent" or "intrinsic" impurities that are present in an intimate mixture with the coal substance, and are derived either from the original vegetable material or from external sources during the accumulation of coal-forming plant remains; (2) impurities formed either during the laying down of the coal bed or subsequently, which occur in the form of partings, "veins," and nodules of clay, shale, pyrite, and calcite; and (3) impurities that become mechanically mixed with the coal in the process of mining, such as fragments of roof and floor.

INHERENT IMPURITIES.

The percentage of ash derived from the intimately mixed impurities may be determined by selecting a sample of clean lumps of homogeneous coal, or, better, by placing the coarsely crushed coal in a solution of calcium or zinc chloride with a specific gravity of 1.35. The coal will float and the heavier extraneous impurities will sink. In this way a sample of clean coal may be prepared for analysis. In some samples the percentage of "intrinsic" ash is so low that the ash must have come exclusively from the original coal-forming vegetation. Certain coals have deposited in their minute joints or cleavage planes thin flakes of calcite, gypsum, silica, or clay, and some have veinlets of pyrite. These impurities are not intimately mixed with the coal substance, but can not be separated from the coal by any commercial methods of preparation. In some coals, also, microscopic crystals of pyrite are so distributed that washing does not appreciably lower the sulphur content.

EXTRANEOUS IMPURITIES.

The percentage of ash derived from extraneous impurities varies considerably, depending on the number and size of the partings in the seam, the possibility of separating them from the coal, and the care with which the coal is mined. Such impurities of this character, when in the form of "slate" or bone, may be removed to some degree by suitable methods of washing, picking, or screening. The possibility of improving the quality of coal by washing may be determined by the laboratory float-and-sink test already described.

Coal ash, as determined, usually weighs less than the organic matter from which it is produced because of the loss of volatile constituents during ignition. Shale and clay lose their water of hydration; the carbonates are more or less decomposed, giving off carbon dioxide, and the iron pyrite is changed to ferric oxide, giving off sulphur dioxide, either to the atmosphere or to the free calcium oxide that has been formed from the carbonate. In coals containing calcium carbonate a large proportion of the sulphur may be retained in the ash as calcium sulphate.

VOLATILE MATTER AND FIXED CARBON.

The volatile matter and fixed carbon represent approximately the relative proportions of gaseous and solid combustible matter that are obtained from coal by heating it in a closed vessel. The volatile matter consists chiefly of the combustible gases hydrogen, carbon monoxide, and methane and other hydrocarbons, and some noncombustible gases, as carbon dioxide and water vapor. The volatile matter does not include the water that can be removed from the coal by heating it at 105° C.

From the standpoint of the analyst, 950° to 1,000° C. appears to be the best temperature for the determination of volatile matter, as most duplicate determinations made within those limits agree within 0.5 per cent.

The residue of coke left in the crucible after the ash has been deducted is reported as "fixed carbon." The fixed carbon does not represent the total carbon in the coal, for a considerable part of the carbon is expelled as volatile matter, being in combination with hydrogen as hydrocarbon or with oxygen as carbon monoxide and carbon dioxide. Furthermore, fixed carbon is not pure carbon. The carbonized residue contains, in addition to the ash-forming constituents, several tenths per cent of hydrogen and oxygen, from 0.4 to 1.0 per cent nitrogen, and approximately half the sulphur that was in the coal. The terms "volatile matter" or "volatile combustible matter" and "fixed carbon" do not represent definite compounds that existed in the coal before heating. The method of determination is arbitrary, and results are comparable only when the temperature and rate of heating are the same. In general the sum of the fixed carbon and ash may be taken as representing the relative amount of coke yield from coking coals. However, the conditions under which the laboratory determination is made are so different from the conditions in the commercial manufacture of coke or gas that no close agreement can be expected. The slow rate of heating in the industrial processes yields several per cent more coke than is obtained in the laboratory determination.

SULPHUR.

In the proximate analysis of coal the sulphur is included in the volatile matter, fixed carbon, and ash. If the ash contains no lime or alkali oxides, all the sulphur is distributed between the volatile matter and fixed carbon, approximately half the sulphur being retained by the fixed carbon and the remainder escaping with the volatile matter.

FORMS OF OCCURRENCE IN COAL.

Sulphur occurs in coal (1) as pyrite or marcasite, (2) as sulphate of iron, lime, and alumina, and (3) in combination with the coal substance as organic compounds.

In some coal beds balls, lenses, and bands of pyrite are easily distinguishable in the face of the bed, and may be largely removed by breaking up the lumps of coal containing them and by suitable washing or picking. In other beds the pyrite is disseminated throughout the lumps of coal in veinlets or in such small particles that it is not practicable to separate them from the lump coal or even to remove them from the screenings by washing.

Sulphur combined as sulphates of iron, lime, and alumina may be found in weathered coal. On exposure to air pyrite and marcasite tend to absorb oxygen and form iron sulphate. Sulphur combined as sulphate has no heating value.

Next to pyritic sulphur the most prominent form of sulphur in coal is that which occurs in some organic combination with the coal substance. There is no direct method of quantitatively determining the relative amounts of pyritic and "organic" sulphur. However, as regards many coals, if all the sulphate sulphur and an amount of sulphur sufficient to form pyrite equivalent to all the iron in the coal be deducted from the total sulphur, there yet remains a considerable amount of sulphur.

ULTIMATE ANALYSIS.

In the ultimate analysis the composition of coal is expressed in percentages of ash, sulphur, carbon, hydrogen, nitrogen, and oxygen. The sum of these constituents is taken as equal to 100 per cent. As there is no simple direct method for the determination of oxygen, it is estimated by subtracting the sum of the other five constituents from 100. This method throws the summation of all the errors incurred in the other determinations upon the oxygen.

The determination of carbon, hydrogen, and nitrogen, although requiring experience and a considerable degree of skill on the part of the chemist, is not subject to the arbitrary conditions that must be maintained in the proximate analysis. As the chemist is dealing with definite chemical elements, he should be able to make duplicate determinations, agreeing within the following limits: Carbon, 0.3 per cent; hydrogen, 0.07 per cent; nitrogen, 0.05 per cent.

CALIFORIC VALUE.

DIRECT DETERMINATION IN THE BOMB CALORIMETER.

The most accurate method of determining the total heating value is by combustion in a bomb calorimeter. The instrument should be carefully standardized by burning substances of known calorific value, such as the standard samples of cane sugar, naphthalene, and benzoic acid, that are now being furnished by the Bureau of Standards. The standardization should be conducted under exactly the same conditions and with the same thermometer that is used in the tests. The use of calibrated thermometers is essential. In carefully conducted calorimetric work the probable error should not exceed 0.3 per cent, which corresponds to about 50 B. t. u. in a high-grade coal.

This degree of accuracy is higher than can be obtained in the usual methods of sampling, the error introduced by which oftentimes increases the total error to 100 B. t. u.

CALCULATION OF CALORIFIC VALUE BY DULONG'S FORMULAS.

Before the bomb calorimeter came into general use the heating value of a coal was frequently calculated from the ultimate analysis by one of Dulong's formulas. These formulas, which are occasionally used in the bureau's work to verify results with the bomb calorimeter, are as follows:

(1) Calorific value in calories per gram $=8080C+34460(H-\frac{O}{8})+2250S$.

(2) Calorific value in B. t. u. per pound $=14544C+62028(H-\frac{O}{8})+4050S$.

There is fairly close agreement between the calorific values calculated by Dulong's formula and those determined by the calorimeter. In coals of higher rank than subbituminous the agreement is usually within 1.5 per cent.

DESCRIPTIONS OF SAMPLES.

The samples of coal and the mines from which they were taken are described in the same general manner as the corresponding information was presented in Bulletin 22. The descriptions have been compiled from published reports or from the note books of the engineers or geologists who collected the samples and necessarily vary in completeness. The purpose of the descriptions is to present such information as bears on the significance of the analyses and to give supplementary details, such as methods of mining and clearing coal, that may be of assistance to Government engineers or purchasing agents and to other persons in preparing specifications and awarding contracts for fuel.

The names of the coal beds, geologic formations, mining districts, and coal fields are those used by the United States Geological Survey. In the descriptions of the sections of coal beds, the word "shale" has been substituted for the "slate" of collectors and "bony coal" or "bone" for "bone coal."

CLASSIFICATION OF COAL.

The usage of the Survey has been followed also in the classification of the coals. The following statement on coal classification is made by Campbell [a] in a recent report.

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[a] Campbell, M. R., Analyses of coal samples from various fields in the United States: U. S. Geol. Survey Bull. 541, 1914.

The classification of coal into various grades, such as bituminous, semibituminous, and lignite, is arbitrary and not at all satisfactory, but it is in common use in the United States, and it seems desirable to give the class for each sample, because from the analysis one could not always determine this important point. The classes generally used in the United States are as follows: Anthracite, semianthracite, semibituminous, bituminous, subbituminous, and lignite.

Anthracite.—Anthracite coal is generally well known, but in a systematic classification it is generally defined as a hard coal having a fuel ratio (fixed carbon divided by volatile matter) of not less than 10. Most of this coal comes from the anthracite fields of eastern Pennsylvania, but small areas are known in some of the Western States.

Semianthracite.—Semianthracite coal has a fuel ratio ranging from 6 or 7 to 10. There is only a small amount of this coal in the United States.

Semibituminous.—Semibituminous coal is of great commercial importance, but is not widely distributed. Its fuel ratio ranges from 3 to 6 or 7. It is the best steam coal in the country, and some of it can be utilized in the manufacture of coke. The centers of production are the Pocahontas and New River fields of Virginia and West Virginia, Georges Creek field of Maryland, Windber field of Pennsylvania, and the western end of the Arkansas field in the vicinity of Fort Smith. Though small areas containing this grade of coal have been found in Washington and Colorado, the amount of such coal in these fields is small.

Bituminous.—Bituminous coal is the most important grade in the country and includes most of the coals east of the Rocky Mountains. In the Western States there are large areas of bituminous coal, such as the Trinidad-Raton field of Colorado and New Mexico; the Grand Hogback field of Colorado; the Book Cliffs field of Utah; the Rock Springs, Kemmerer, and Black Hills fields of Wyoming; the Great Falls field of Montana; and many districts of Washington. This grade furnishes most of the coking coal of the country, and it is largely sold for steam raising and for domestic use.

Subbituminous.—The term "subbituminous" has been adopted by the United States Geological Survey for what has generally been called "black lignite." The latter term is objectionable, for the reason that the coal is not lignitic in the sense of being woody, and the use of the term seems to imply that the coal is little better than the brown, woody lignite of North Dakota, whereas many of the coals of this class closely approach the lowest grade of bituminous coal. In fact, it is extremely difficult to separate this class from the one below and the one above. It is generally distinguished from lignite by its color and freedom from apparent woody texture and from bituminous coal by the slacking it undergoes when exposed to the weather. As the latter is an important difference in commercial use, it has been adopted by the Geological Survey as the criterion for the separation of subbituminous and bituminous coals.

Subbituminous coal is found in most of the western fields, being well known in the field about Boulder and Denver and in North Park, Colo.; Gallup, N. Mex.; Hanna, Douglas, Sheridan, and the Bighorn Basin, Wyo.; Red Lodge and Musselshell, Mont., and in many of the districts of Washington and Oregon.

Lignite.—As used by the Geological Survey the term "lignite" is restricted to the coals that are distinctly brown and generally woody. They are intermediate in quality between peat and subbituminous coal. Lignite is abundant in the North in eastern Montana and North Dakota and in the northwest corner of South Dakota. In the South it is present in all of the Gulf States, but it has been developed commercially only in Texas.

TABULATED ANALYSES.

On the following pages are given the analyses of the samples of coal collected during the period covered by this report.

In the first column of the table, locations of samples are, for brevity, not worded as in the detailed descriptions (pp. 147–405). The word "cut" denotes thickness of coal sampled.

Table of chemical analyses.

Locality, bed, etc.	Mine notes[a] (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.[b]	Kind.[c]	Condition.[d]	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
ALABAMA.																
JEFFERSON COUNTY.																
Littleton, near, 20 miles northwest of Birmingham; Banner mine, Big or Mary Lee bed (rib near mouth, room 4, right entry 5, 2,000 feet east of shaft No. 1, 8-foot 7½-inch bed, 6-foot 11¼-inch cut).	147	12083	A	1	2.48	26.77	59.40	11.35	0.86	....	....	....	....	1.4	....	....
				2	....	27.45	60.91	11.64	.88	....	....	....	....	....	....	....
				3	....	31.07	68.93	....	1.00	....	....	....	....	....	....	....
Same (rib near mouth of room 2, left entry 8, 2,400 feet east of shaft No 1, 10-foot 4-inch bed, 7-foot 6-inch cut).	147	12084	A	1	2.29	26.73	59.84	11.14	.74	....	....	....	....	1.3	....	....
				2	....	27.36	61.24	11.40	.76	....	....	....	....	....	....	....
				3	....	30.88	69.12	....	.86	....	....	....	....	....	....	....
Same (right rib of main slope at mouth of left entry 9, 2,700 feet east of shaft No. 1, 10-foot bed, 7-foot 4-inch cut).	147	12085	A	1	2.42	26.89	60.04	10.65	.87	....	....	....	....	1.5	....	....
				2	....	27.56	61.53	10.91	.89	....	....	....	....	....	....	....
				3	....	30.94	69.06	....	1.00	....	....	....	....	....	....	....
Same (composite of samples 12083, 12084, and 12085).	147	12086	A	1	2.53	26.94	59.48	11.06	.79	4.80	74.44	1.59	7.33	1.4	7,381	13,286
				2	....	27.64	61.02	11.34	.81	4.64	76.38	1.63	5.20	....	7,573	13,631
				3	....	31.18	68.82	....	.91	5.23	86.15	1.84	5.87	....	8,542	15,376
Morris, 4 miles east of; Indio mine, Jefferson bed (face of room 7, off left south entry 5, 2,000 feet southeast of shaft, 4-foot 5-inch bed, 2-foot 7-inch cut).	148	16683	A	1	1.96	30.97	59.03	8.04	3.26	....	....	....	....	1.1	7,705	13,869
				2	....	31.59	60.21	8.20	3.33	....	....	....	....	....	7,859	14,146
				3	....	34.41	65.59	....	3.63	....	....	....	....	....	8,561	15,410
Same (face of room 4, off right north entry 7, 2,500 feet northeast of shaft, 4-foot 8-inch bed, 2-foot 9-inch cut).	148	16684	A	1	2.23	30.96	59.91	6.90	2.56	....	....	....	....	1.4	7,828	14,090
				2	....	31.67	61.27	7.06	2.62	....	....	....	....	....	8,006	14,411
				3	....	34.08	65.92	....	2.82	....	....	....	....	....	8,614	15,506
Same (entrance to room 4, off right south entry 6, 4-foot ½-inch bed, 2-foot 4½-inch cut).	148	16685	A	1	1.95	30.88	59.40	7.77	2.45	....	....	....	....	1.1	7,736	13,925
				2	....	31.49	60.59	7.92	2.50	....	....	....	....	....	7,890	14,202
				3	....	34.20	65.80	....	2.72	....	....	....	....	....	8,569	15,424
Same (in main south aircourse, at left south entry 6, 4-foot 1-inch bed, 2-foot 7-inch cut).	148	16686	A	1	1.83	30.54	61.11	6.52	2.68	....	....	....	....	.9	7,873	14,171
				2	....	31.11	62.25	6.64	2.73	....	....	....	....	....	8,019	14,434
				3	....	33.32	66.68	....	2.92	....	....	....	....	....	8,589	15,460
Same (composite of samples 16683, 16684, 16685, and 16686).	148	16687	A	1	1.95	30.66	60.04	7.35	2.75	4.93	77.71	1.70	5.56	1.1	7,757	13,963
				2	....	31.27	61.23	7.50	2.80	4.80	79.26	1.73	3.91	....	7,911	14,240
				3	....	33.81	66.19	....	3.03	5.19	85.69	1.87	4.22	....	8,553	15,396

[a] Figures in column No. 2 refer to page of this bulletin where may be found the description of the section in the mine where the sample was taken.

[b] Laboratory numbers with a prefix "W" represent samples analyzed in the Washington laboratory of the bureau; all others were analyzed at the Pittsburgh laboratory. An asterisk preceding the laboratory number indicates that the volatile matter was determined by the modified method, see page 7.

[c] The kind of sample is denoted by letter, as follows: A=mine sample collected by an inspector of the Bureau of Mines; B=mine sample collected by a geologist of the United States Geological Survey; C=car sample taken at the fuel-testing plant.

[d] The form of analysis is denoted by number, as follows: 1=sample as received; 2=dried at a temperature of 105° C.; 3=moisture and ash free.

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
ALABAMA—Continued.																
ST. CLAIR COUNTY.																
Coal City; Coal Branch mine, Coal City bed (first left room off entry 26, 3,000 feet north of slope mouth, 5-foot bed, 4-foot 10-inch cut).	149	12440	B	1	2.79	32.92	60.28	4.01	1.10	....	....	....	....	1.8	8,053	14,495
				2	....	33.86	62.01	4.13	1.13	....	....	....	....	....	8,284	14,911
				3	....	35.32	64.68	....	1.18	....	....	....	....	....	8,641	15,554
Same (end of entry 34, 2,800 feet north of slope mouth, 3-foot 10-inch cut).	149	12441	B	1	1.97	34.32	58.21	5.50	1.30	....	....	....	....	1.2	8,002	14,404
				2	....	35.01	59.38	5.61	1.33	....	....	....	....	....	8,163	14,693
				3	....	37.09	62.91	....	1.41	....	....	....	....	....	8,648	15,566
Same (end of room 32, 3,000 feet northeast of slope mouth, 2-foot 10-inch cut).	149	12442	B	1	2.41	33.72	60.65	3.22	.81	....	....	....	....	1.4	8,214	14,785
				2	....	34.55	62.15	3.30	.83	....	....	....	....	....	8,417	15,151
				3	....	35.73	64.27	....	.86	....	....	....	....	....	8,704	15,667
Same (composite of samples 12440, 12441, and 12442).	149	12443	B	1	2.38	33.31	60.00	4.31	1.07	5.27	81.85	1.46	6.04	1.5	8,138	14,648
				2	....	34.12	61.46	4.42	1.10	5.13	83.85	1.50	4.00	....	8,337	15,007
				3	....	35.70	64.30	....	1.15	5.37	87.72	1.57	4.19	....	8,722	15,700
Coal City; Vulcan No. 4 mine, Broken Arrow bed (room 21, off left heading 2, 750 feet southeast of mine mouth, 4-foot 1½-inch bed, 3-foot 3½-inch cut).	149	12436	B	1	3.76	29.07	58.67	8.50	1.46	....	....	....	....	2.9	....	....
				2	....	30.21	60.96	8.83	1.52	....	....	....	....	....	....	....
				3	....	33.14	66.86	....	1.67	....	....	....	....	....	....	....
Same (in mine, 1,100 feet east of mine mouth, 4-foot 9-inch bed, 3-foot 8-inch cut).	149	12437	B	1	3.54	29.04	59.20	8.22	1.36	....	....	....	....	2.7	....	....
				2	....	30.11	61.37	8.52	1.41	....	....	....	....	....	....	....
				3	....	32.91	67.09	....	1.54	....	....	....	....	....	....	....
Same (room 15, left heading 2, 600 feet south of mine mouth, 3-foot 5-inch cut).	149	12438	B	1	4.42	27.72	56.57	11.29	1.43	....	....	....	....	3.7	....	....
				2	....	29.00	59.19	11.81	1.50	....	....	....	....	....	....	....
				3	....	32.88	67.12	....	1.70	....	....	....	....	....	....	....
Same (composite of samples 12436, 12437, and 12438).	149	12439	B	1	3.94	28.14	58.59	9.33	1.46	5.10	75.71	1.32	7.08	3.1	7,498	13,496
				2	....	29.29	61.00	9.71	1.52	4.85	78.81	1.37	3.74	....	7,805	14,049
				3	....	32.44	67.56	....	1.68	5.37	87.28	1.52	4.15	....	8,644	15,559
SHELBY COUNTY.																
Straven; sec. 7, T. 21 S., R. 3 W.; Straven mine, Upper Straven bed (near end of slope, 2,000 feet north of slope mouth; 4-foot 10½-inch bed, 3 foot ½-inch cut).	150	14391	B	1	3.06	35.64	53.33	7.97	.59	....	....	....	....	1.0	7,318	13,172
				2	....	36.77	55.01	8.22	.61	....	....	....	....	....	7,549	13,588
				3	....	40.06	59.94	....	.66	....	....	....	....	....	8,225	14,580

Same (position and bed same as sample 14391, 13½-inch cut).	150	14392	B	1	3.61	36.80	53.15	6.44	1.10	....	....	....	....	1.4	7,503	13,505
				2	....	38.18	55.14	6.68	1.14	....	....	....	....	....	7,784	14,011
				3	40.91	59.09	....	1.22	....	....	....	....	....	....	8,341	15,014
TUSCALOOSA COUNTY.																
Abernant; sec. 18, T. 20 S., R. 6 W.; Abernant mine, Jagger bed (face of right entry 13, 6-foot 7½-inch bed, 5-foot 6-inch cut).	150	14717	A	1	2.62	24.18	64.11	9.09	.64	4.72	77.52	1.48	6.55	1.8	7,627	13,729
				2	....	24.83	65.84	9.33	.66	4.55	79.61	1.52	4.33	....	7,832	14,098
				3	....	27.39	72.61	....	.73	5.02	87.80	1.68	4.77	....	8,638	15,548
ALASKA.																
ALASKA PENINSULA.																
Chignik Bay; Warner's claim	151	12742	A	1	2.33	34.82	46.75	16.10	.65	4.82	66.58	.81	11.04	....	6,627	11,928
				2	....	35.65	47.87	16.48	.67	4.67	68.17	.83	9.18	....	6,785	12,213
				3	....	42.68	57.32	....	.80	5.59	81.62	.99	11.00	....	8,124	14,623
BERING RIVER.																
Barrett Creek, three-fourths of a mile above mouth of, or junction with Clear Creek; first bed in series (5-foot bed).	151	12716	A	1	3.63	15.37	73.95	7.05	1.18	3.84	77.85	1.64	8.44	....	7,287	13,116
				2	....	15.95	76.73	7.32	1.22	3.57	80.78	1.70	5.41	....	7,562	13,612
				3	....	17.21	82.79	....	1.32	3.85	87.16	1.83	5.84	....	8,159	14,686
Three-fourths of a mile above mouth of; second bed (100 feet, measured across dip, up the creek, above point where sample 12716 was taken, 2-foot bed).	151	12709	A	1	2.05	14.10	72.60	11.25	1.02	3.78	77.59	1.44	4.92	....	7,358	13,244
				2	....	14.39	74.12	11.49	1.04	3.62	79.21	1.47	3.17	....	7,512	13,522
				3	....	16.26	83.74	....	1.17	4.09	89.49	1.66	3.59	....	8,487	15,277
Three-fourths of a mile above mouth of; third bed (15 to 25 feet, measured across strike, above second bed).	151	12707	A	1	1.55	12.67	65.03	20.75	1.14	3.49	69.53	1.35	3.74	....	6,577	11,839
				2	....	12.87	66.05	21.08	1.16	3.37	70.62	1.37	2.40	....	6,680	12,024
				3	....	16.31	83.69	....	1.47	4.27	89.48	1.74	3.04	....	8,464	15,235
Three-fourths of a mile above mouth of; fourth bed (350 feet along the creek below the fifth bed, or 100 to 125 feet across strike, and probably 100 to 150 feet across strike above third bed, noted from memory).	152	12708	A	1	1.80	14.05	81.72	2.43	1.13	4.05	86.67	1.66	4.06	....	8,254	14,858
				2	....	14.31	83.22	2.47	1.15	3.92	88.26	1.69	2.51	....	8,405	15,129
				3	....	14.67	85.33	....	1.18	4.02	90.49	1.73	2.58	....	8,618	15,512
Near junction with Clear Creek; east side of creek; from bed about 350 feet up the creek and about 125 to 150 feet, across the strike, from fourth bed on creek.	152	12714	A	1	2.88	14.24	72.83	10.05	.67	3.68	76.50	1.62	7.48	....	7,209	12.976
				2	....	14.66	74.99	10.35	.69	3.46	78.77	1.67	5.06	....	7,423	13,361
				3	....	16.35	83.65	....	.77	3.86	87.86	1.86	5.65	....	8,280	14,904
Same (between foot-wall or floor and the band of shale in middle of bed; 3-foot cut).	152	12710	A	1	1.65	10.87	29.48	58.00	.33	2.20	31.93	.92	6.62	....	3,053	5,496
				2	....	11.05	29.98	58.97	.34	2.05	32.47	.94	5.23	....	3,104	5,587
				3	....	26.93	73.07	....	.83	5.00	79.14	2.29	12.74	....	7,565	13,617
Three-fourths of a mile from mouth of, or junction with Clear Creek (20 to 30 feet, across strike, from samples 12710 and 12714).	152	12718	A	1	2.73	13.77	58.12	25.38	.76	3.27	61.54	1.35	7.70	....	5,802	10,443
				2	....	14.16	59.75	26.09	.78	3.05	63.27	1.39	5.42	....	5,965	10,737
				3	....	19.16	80.84	....	1.06	4.13	85.60	1.88	7.33	....	8,071	14,528
Three-fourths of a mile from mouth of (150 feet upstream; from samples 12710 and 12718, 50 feet across strike).	152	12712	A	1	2.13	13.47	78.22	6.18	.68	3.95	82.89	1.74	4.56	....	7,879	14,183
				2	....	13.76	79.93	6.31	.69	3.79	84.70	1.78	2.73	....	8,051	14,492
				3	....	14.69	85.31	....	.74	4.05	90.40	1.90	2.91	....	8,593	15,467
Branch of Clear Creek, 1,000 feet upstream from Barrett Creek (about 3,000 feet northeast of point where sample 12712 was taken).	153	12711	A	1	1.48	12.82	76.72	8.98	1.35	3.77	80.96	1.64	3.30	....	7,697	13,854
				2	....	13.01	77.88	9.11	1.37	3.66	82.17	1.66	2.03	....	7,812	14,062
				3	....	14.31	85.69	....	1.51	4.03	90.40	1.83	2.23	....	8,595	15,471
Carbon Mountain: No. 1 opening; 10 to 12 foot bed	153	12732	A	1	5.65	15.05	76.10	3.20	.66	3.43	79.28	1.70	11.73	....	7,223	13,001
				2	....	15.95	80.66	3.39	.70	2.97	84.03	1.80	7.11	....	7,656	13,781
				3	....	16.51	83.49	....	.72	3.07	86.98	1.86	7.37	....	7,925	14,265

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
ALASKA—Continued.																
BERING RIVER—continued.																
Clear Creek, 4½ miles above mouth of; below main falls (natural coke, 6 to 8 inches thick).	153	12713	A	1	1.18	10.02	84.50	4.30	1.59	3.23	86.37	1.46	3.05	....	7,969	14,344
				2	....	10.14	85.51	4.35	1.61	3.14	87.40	1.48	2.02	....	8,064	14,515
				3	....	10.60	89.40	....	1.68	3.28	91.38	1.55	2.11	....	8,431	15,176
Same (coal 6 to 8 inches thick, next coke)	153	12715	A	1	1.58	11.95	84.72	1.75	.95	3.82	88.69	1.42	3.37	....	8,346	15,022
				2	....	12.14	86.08	1.78	.97	3.70	90.12	1.44	1.99	....	8,480	15,264
				3	....	12.36	87.64	....	.99	3.77	91.75	1.47	2.02	....	8,633	15,539
Same (coal from main part of bed)	153	12717	A	1	1.75	13.62	79.98	4.65	1.19	3.93	84.62	1.49	4.12	....	8,002	14,403
				2	....	13.86	81.41	4.73	1.21	3.81	86.13	1.52	2.60	....	8,144	14,659
				3	....	14.55	85.45	....	1.27	4.00	90.40	1.60	2.73	....	8,548	15,387
Cunningham claim (first crosscut going in)	153	12739	A	1	1.01	16.44	71.04	11.51	.69	4.06	78.61	1.68	3.45	....	7,576	13,637
				2	....	16.61	71.76	11.63	.70	3.99	79.41	1.70	2.57	....	7,653	13,775
				3	....	18.80	81.20	....	.79	4.52	89.86	1.92	2.91	....	8,660	15,588
Glacier Creek; anthracite bowlder found on creek	153	12734	A	1	2.68	7.37	88.37	1.58	.54	3.09	89.96	1.26	3.57	....	8,171	14,708
				2	....	7.57	90.81	1.62	.55	2.87	92.43	1.29	1.24	....	8,396	15,113
				3	....	7.69	92.31	....	.56	2.92	93.96	1.31	1.25	....	8,535	15,363
Hartline No. 1 mine	154	12724	A	1	6.45	16.75	73.45	3.35	.57	3.58	76.99	1.64	13.87	....	6,963	12,534
				2	....	17.90	78.52	3.58	.61	3.06	82.29	1.75	8.71	....	7,443	13,397
				3	....	18.56	81.44	....	.63	3.17	85.34	1.81	9.05	....	7,719	13,894
Hartline No. 2 mine	154	12726	A	1	3.48	11.67	78.20	6.65	.78	3.46	80.27	1.54	7.30	....	7,458	13,424
				2	....	12.09	81.02	6.89	.81	3.18	83.17	1.60	4.35	....	7,727	13,909
				3	....	12.98	87.02	....	.87	3.42	89.32	1.72	4.67	....	8,299	14,938
Hartline No. 3 mine	154	12725	A	1	4.75	13.72	63.31	18.22	.62	3.14	65.93	1.32	10.77	....	6,011	10,820
				2	....	14.40	66.47	19.13	.65	2.74	69.22	1.39	6.87	....	6,311	11,360
				3	....	17.81	82.19	....	.80	3.39	85.60	1.72	8.49	....	7,804	14,047
Hartline No. 4 mine	154	12727	A	1	7.03	16.32	71.48	5.17	.59	3.49	75.56	1.61	13.58	....	6,838	12,309
				2	....	17.55	76.89	5.56	.63	2.91	81.27	1.73	7.90	....	7,355	13,239
				3	....	18.58	81.42	....	.67	3.08	86.06	1.83	8.36	....	7,788	14,018
Hartline No. 5 mine	154	12728	A	1	1.65	8.03	68.62	21.70	.55	3.08	69.33	1.49	3.85	....	6,456	11,620
				2	....	8.16	69.78	22.06	.56	2.95	70.49	1.52	2.42	....	6,564	11,815
				3	....	10.47	89.53	....	.72	3.78	90.44	1.95	3.11	....	8,422	15,160

Hartline No. 6 mine	154	12723	A	1	4.78	14.22	78.61	2.39	.58	3.69	81.67	1.59	10.08	....	7,460	13,428
				2	....	14.93	82.56	2.51	.61	3.32	85.77	1.67	6.12	....	7,834	14,101
				3	....	15.31	84.69	....	.63	3.41	87.97	1.71	6.28	....	8,035	14,463
Hunts mine	154	12729	A	1	1.41	8.64	69.07	20.88	.77	3.05	70.51	1.38	3.41	....	6,612	11,902
				2	....	8.76	70.06	21.18	.78	2.93	71.52	1.40	2.19	....	6,707	12,073
				3	....	11.11	88.89	....	.99	3.72	90.74	1.78	2.77	....	8,509	15,316
Mount Ann; top of mountain	154	12733	A	1	3.65	9.20	84.58	2.57	.60	3.20	86.49	1.11	6.03	....	7,879	14,182
				2	....	9.55	87.78	2.67	.62	2.90	89.77	1.15	2.89	....	8,178	14,720
				3	....	9.81	90.19	....	.64	2.98	92.23	1.18	2.97	....	8,402	15,124
Taken on way up mountain	154	12720	A	1	5.13	9.17	60.87	24.83	.49	2.57	61.88	.99	9.24	....	5,488	9,878
				2	....	9.67	64.16	26.17	.52	2.11	65.23	1.04	4.93	....	5,785	10,413
				3	....	13.10	86.90	....	.70	2.86	88.35	1.41	6.68	....	7,836	14,105
Near top of mountain	154	12719	A	1	3.40	9.25	76.05	11.30	.51	2.95	77.92	1.04	6.28	....	7,061	12,710
				2	....	9.58	78.72	11.70	.53	2.66	80.66	1.08	3.37	....	7,310	13,158
				3	....	10.85	89.15	....	.60	3.01	91.35	1.22	3.82	....	8,279	14,902
Mount Hamilton; ½ mile south of U. S. L. M.; McDonald mine, unnamed bed; Christopher group, near south line, top of ridge.	155	12730	A	1	8.60	13.50	68.87	9.03	.56	3.96	66.57	1.32	18.56	....	6,111	11,000
				2	....	14.77	75.35	9.88	.61	3.28	72.83	1.44	11.96	....	6,686	12,035
				3	....	16.39	83.61	....	.68	3.64	80.81	1.60	13.27	....	7,419	13,354
Same (Face of entry, head of entry)	155	12722	A	1	.96	16.69	69.37	12.98	1.30	3.98	76.43	1.41	3.90	....	7,476	13,456
				2	....	16.85	70.04	13.11	1.31	3.91	77.17	1.42	3.08	....	7,549	13,588
				3	....	19.39	80.61	....	1.51	4.50	88.81	1.63	3.55	....	8,688	15,638
Same (in mine)	155	12731	A	1	1.03	15.97	65.02	17.98	2.38	3.58	72.21	1.25	2.60	....	6,958	12,525
				2	....	16.14	65.69	18.17	2.40	3.51	72.96	1.26	1.70	....	7,030	12,654
				3	....	19.72	80.28	....	2.93	4.29	89.16	1.54	2.08	....	8,591	15,464
St. Mary's Creek; Hartline No. 7 mine	155	12721	A	1	2.55	9.45	76.60	11.40	.93	3.35	78.26	1.62	4.44	....	7,263	13,073
				2	....	9.70	78.60	11.70	.95	3.15	80.31	1.66	2.23	....	7,453	13,415
				3	....	10.99	89.01	....	1.08	3.57	90.95	1.88	2.52	....	8,441	15,194
Trout Creek, Cunningham claims on; prospect tunnel No. 4 (chute between timber sets 19 and 20, 76 feet from drift mouth, 10 feet from entry; 5-foot 4-inch cut).	155	15355	A	1	1.18	17.41	78.39	3.02	.66	4.47	86.62	1.82	3.41	....	8,335	15,003
				2	....	17.62	79.32	3.06	.67	4.39	87.65	1.84	2.39	....	8,434	15,181
				3	....	18.18	81.82	....	.69	4.53	90.42	1.90	2.46	....	8,701	15,662
Same (rib of right crosscut between timber sets 18 and 19, 72 feet from drift mouth and 17 feet from entry, 4-foot 1-inch cut).	155	15356	A	1	.98	16.40	72.34	10.28	.61	4.21	79.71	1.64	3.55	....	7,705	13,869
				2	....	16.56	73.06	10.38	.62	4.14	80.50	1.66	2.70	....	7,781	14,006
				3	....	18.48	81.52	....	.69	4.62	89.82	1.85	3.02	....	8,682	15,628
Same (rib at face of right crosscut, between timber sets 11 and 12, 44 feet from drift mouth, 8-foot 3½-inch cut).	155	15357	A	1	2.68	16.25	76.30	4.77	.63	4.45	83.85	1.68	4.62	....	8,090	14,562
				2	....	16.70	78.40	4.90	.65	4.26	86.16	1.73	2.30	....	8,312	14,962
				3	....	17.56	82.44	....	.68	4.48	90.60	1.82	2.42	....	8,740	15,732
Same (right crosscut between timber sets 25 and 26, 100 feet from drift mouth, 11 feet from entry, 6-foot 7-inch cut).	155	15358	A	1	1.51	16.32	76.26	5.91	.65	4.28	84.04	1.72	3.40	....	8,059	14,506
				2	....	16.57	77.43	6.00	.66	4.17	85.33	1.75	2.09	....	8,182	14,728
				3	....	17.63	82.37	....	.70	4.44	90.77	1.86	2.23	....	8,704	15,667
Prospect tunnel No. 5 (chute to left, between timber sets 21 and 22, 84 feet from drift mouth, 17 feet up chute, 5-foot 3-inch cut).	155	15359	A	1	1.22	16.36	77.94	4.48	.65	4.42	85.64	1.66	3.15	....	8,207	14,773
				2	....	16.56	78.90	4.54	.66	4.33	86.70	1.68	2.09	....	8,309	14,956
				3	....	17.35	82.65	....	.69	4.54	90.83	1.76	2.18	....	8,705	15,669
Same (left crosscut between timber sets 13 and 14, 52 feet from drift mouth, 5 feet from entry, 5-foot 3-inch cut).	155	15360	A	1	1.38	16.48	78.38	3.76	.67	4.38	85.11	1.82	4.26	....	8,266	14,879
				2	....	16.71	79.48	3.81	.68	4.29	86.30	1.85	3.07	....	8,382	15,088
				3	....	17.37	82.63	....	.71	4.46	89.72	1.92	3.19	....	8,714	15,685

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
ALASKA—Continued.																
MATANUSKA.																
Anthracite Ridge, outcrop on ridge	156	12745	A	1	3.59	6.83	64.29	25.29	0.19	2.03	65.91	0.59	5.99	....	5,704	10,268
				2	....	7.08	66.69	26.23	.20	1.69	68.36	0.61	2.91	....	5,916	10,650
				3	....	9.60	90.40	....	.27	2.29	92.67	0.83	3.94	....	8,020	14,437
Outcrop on ridge	156	12750	A	1	1.72	37.43	39.55	21.30	.41	4.20	61.68	1.36	11.06	....	5,966	10,738
				2	....	38.09	40.24	21.67	.42	4.08	62.76	1.38	9.69	....	6,070	10,926
				3	....	48.63	51.37	....	.54	5.21	80.13	1.76	12.36	....	7,750	13,950
Outcrop on ridge	156	12753	A	1	1.12	13.03	75.67	10.18	.64	3.72	78.95	1.88	4.63	....	7,534	13,562
				2	....	13.18	76.52	10.30	.65	3.64	79.84	1.90	3.67	....	7,619	13,714
				3	....	14.69	85.31	....	.72	4.06	89.01	2.12	4.09	....	8,494	15,289
Near middle of south face of ridge. Barometer reading 4,100 (less 400 feet).	156	12746	A	1	2.85	7.75	77.34	12.06	.61	2.66	78.35	1.25	5.07	....	7,073	12,732
				2	....	7.98	79.61	12.41	.63	2.41	80.65	1.29	2.61	....	7,280	13,105
				3	....	9.11	90.89	....	.72	2.75	92.08	1.47	2.98	....	8,312	14,962
South slope, near center, barometer reading 4,500 (less 400).	157	12757	A	1	3.05	10.06	71.01	15.89	.35	2.55	74.03	0.80	6.38	....	6,629	11,932
				2	....	10.37	73.24	16.39	.36	2.28	76.36	0.83	3.78	....	6,838	12,308
				3	....	12.40	87.60	....	.43	2.73	91.33	0.99	4.52	....	8,178	14,720
Chickaloon Creek; at bend of creek, ½ mile above Watson's camp (2-foot 9-inch cut, upper bench).	157	12752	A	1	1.21	19.14	56.01	23.64	.58	3.88	64.17	1.35	6.38	....	6,237	11,226
				2	....	19.37	56.70	23.93	.59	3.80	64.95	1.37	5.36	....	6,313	11,363
				3	....	25.46	74.54	....	.78	5.00	85.38	1.80	7.04	....	8,299	14,938
Same (position and bed same as No. 12752, 4-foot 10-inch cut, lower bench).	157	12764	A	1	1.10	19.40	60.11	19.39	.52	4.07	68.96	1.23	5.83	....	6,727	12,108
				2	....	19.62	60.77	19.61	.53	3.99	69.73	1.24	4.90	....	6,802	12,244
				3	....	24.41	75.59	....	.66	4.96	86.74	1.54	6.10	....	8,461	15,230
Tunnel No. 4, at Watson's camp (west crosscut off entry, 111 feet from entry mouth)	158	12758	A	1	0.80	19.75	67.75	11.70	.58	4.41	77.33	1.73	4.25	....	7,558	13,604
				2	....	19.91	68.30	11.79	.58	4.35	77.96	1.74	3.58	....	7,619	13,714
				3	....	22.57	77.43	....	.66	4.93	88.38	1.97	4.06	....	8,638	15,548
Prospect	158	12761	A	1	0.51	21.91	75.13	2.45	.83	4.92	86.27	1.79	3.74	....	8,496	15,293
				2	....	22.02	75.52	2.46	.83	4.88	86.71	1.80	3.32	....	8,539	15,370
				3	....	22.57	77.43	....	.85	5.00	88.90	1.85	3.40	....	8,754	15,757
Tunnel B (3 feet 3 inches, upper bench)	158	12760	A	1	0.74	21.81	70.45	7.00	.44	4.59	81.93	1.41	4.63	....	7,954	14,317
				2	....	21.97	70.98	7.05	.44	4.54	82.54	1.42	4.01	....	8,014	14,426
				3	....	23.64	76.36	....	.47	4.88	88.80	1.53	4.82	....	8,621	15,518

Same. (3 feet 10 inches, lower bench)	158	12756	A	1	0.60	21.23	69.09	9.08	.45	4.53	80.23	1.33	4.38	....	7,808	14,055
				2	....	21.36	69.51	9.13	.45	4.49	80.71	1.34	3.88	....	7,855	14,139
				3	....	23.51	76.49	....	.50	4.94	88.82	1.47	4.27	....	8,644	15,559
Coal Creek; prospect	158	12744	A	1	1.75	31.30	54.05	12.90	.41	5.10	71.32	1.56	8.71	....	7,139	12,851
				2	....	31.86	55.01	13.13	.42	5.00	72.59	1.59	7.27	....	7,266	13,080
				3	....	36.67	63.33	....	.48	5.76	83.56	1.83	8.37	....	8,364	15,056
Prospect	158	12748	A	1	2.45	37.67	56.25	3.63	.40	5.59	75.91	2.00	12.47	....	7,697	13,855
				2	....	38.62	57.66	3.72	.41	5.45	77.82	2.05	10.55	....	7,890	14,202
				3	....	40.11	59.89	....	.43	5.66	80.82	2.13	10.96	....	8,195	14,751
(No data available)	159	12743	A	1	2.50	36.00	50.61	10.89	.41	5.15	69.79	1.71	12.05	....	7,066	12,719
				2	....	36.92	51.91	11.17	.42	4.99	71.58	1.75	10.09	....	7,247	13,045
				3	....	41.56	58.44	....	.47	5.62	80.58	1.97	11.36	....	8,158	14,684
Matanuska River. O'Brien claims, ½ mile east of place where samples 12755 and 12762 were taken, 100 feet below top of bluff, S. 15° W. from east end of Anthracite Ridge.	159	12763	A	1	1.08	24.04	66.78	8.10	.55	4.83	78.35	1.57	6.60	....	7,746	13,942
				2	....	24.30	67.51	8.19	.56	4.76	79.20	1.59	5.70	....	7,830	14,094
				3	....	26.47	73.53	....	.61	5.18	86.26	1.73	6.22	....	8,528	15,350
O'Brien claims, 1 mile south of Matanuska River, north side of bluff, S. SW. from east end of Anthracite Ridge (upper half of cut).	159	12755	A	1	2.08	25.20	64.24	8.48	.53	4.79	76.90	1.55	7.75	....	7,554	13,598
				2	....	25.73	65.61	8.66	.54	4.66	78.53	1.58	6.03	....	7,714	13,885
				3	....	28.17	71.83	....	.59	5.10	85.97	1.73	6.61	....	8,445	15,201
Same (lower half of cut)	159	12762	A	1	1.30	24.50	64.72	9.48	.50	4.82	77.32	1.46	6.42	....	7,612	13,701
				2	....	24.82	65.57	9.61	.51	4.74	78.34	1.48	5.32	....	7,712	13,882
				3	....	27.46	72.54	....	.56	5.24	86.67	1.64	5.89	....	8.532	15,358
Near and on west side of same gulch where sample 12763 was taken, 35 to 55 feet above same.	160	12747	A	1	1.72	24.36	58.97	14.95	.46	4.46	70.78	1.42	7.93	....	6,992	12,585
				2	....	24.79	60.00	15.21	.47	4.34	72.02	1.44	6.52	....	7,114	12,805
				3	....	29.24	70.76	....	.55	5.12	84.94	1.70	7.69	....	8,390	15,102
Near point where No. 12763 was taken, in east face of gulch 50 to 75 feet above same (across strata), about 50 feet from top of bluff.	160	12749	A	1	1.73	27.60	67.43	3.24	.56	5.01	82.31	1.65	7.23	....	8,117	14,610
				2	....	28.09	68.61	3.30	.57	4.90	83.76	1.68	5.79	....	8,260	14,868
				3	....	29.05	70.95	....	.59	5.07	86.62	1.74	5.98	....	8,542	15,376
In the west face of bluff on creek flowing north by O'Brien's cabin, 300 yards or more up stream (south) from place where sample 12754 was taken.	161	12751	A	1	1.67	24.96	67.39	5.98	.48	4.80	79.79	1.76	7.19	....	7,849	14,128
				2	....	25.38	68.54	6.08	.49	4.69	81.15	1.79	5.80	....	7,982	14,368
				3	....	27.02	72.98	....	.52	4.99	86.40	1.91	6.18	....	8,498	15,296
Same (½ mile south of Matanuska River, and short distance upstream from O'Brien cabin).	161	12754	A	1	2.70	25.72	64.46	7.12	.50	4.69	76.15	1.66	9.88	....	7,436	13,385
				2	....	26.43	66.25	7.32	.51	4.51	78.26	1.71	7.69	....	7,642	13,756
				3	....	28.52	71.48	....	.55	4.87	84.44	1.85	8.29	....	8,246	14,843
West side and near top of bluff (facing east) across the gulch from place where No. 12751 was taken.	162	12759	A	1	3.12	25.18	65.77	5.93	.42	4.65	77.26	1.71	10.03	....	7,460	13,428
				2	....	25.99	67.89	6.12	.43	4.44	79.75	1.77	7.49	....	7,700	13,860
				3	....	27.68	72.32	....	.46	4.73	84.95	1.89	7.97	....	8,202	14,764
Young Creek; longitude 148° 42′ 12″ west, latitude 61° 47′ 40″ north (outcrop near creek, freshly exposed surface, 12-foot 8-inch bed).	162	11382	B	1	10.57	33.80	50.57	5.06	.23	5.41	65.16	1.13	23.01	6.1	6,236	11,225
				2	....	37.80	56.54	5.66	.26	4.74	72.86	1.26	15.22	....	6,973	12,551
				3	....	40.07	59.93	....	.28	5.02	77.23	1.34	16.13	....	7,391	13,304
COLORADO.																
ADAMS COUNTY.																
Bennett, 25 miles northeast of; sec. 12, T. 1 S., R. 61 W.; Thomas mine (at working face, 4-foot bed 100 feet south from bottom of 110-foot shaft).	163	*13141	B	1	35.00	27.39	30.23	7.38	.31	6.56	41.71	.74	43.30	26.9	3,879	6,982
				2	....	42.14	46.51	11.35	.48	4.11	64.17	1.14	18.75	....	5,968	10,742
				3	....	47.53	52.47	....	.54	4.64	72.38	1.29	21.15	....	6,732	12,118

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
COLORADO—Continued.																
BOULDER COUNTY.																
Lafayette; Simpson mine, Simpson bed (room 1, off entry 125; 11-foot bed, 6-foot 10-inch cut).	163	*15165	A	1	20.71	31.82	43.98	3.49	0.45	6.04	57.80	1.20	31.02	10.3	5,523	9,941
				2	....	40.13	55.47	4.40	.57	4.72	72.90	1.51	15.90	....	6,966	12,539
				3	....	41.98	58.02	....	.60	4.94	76.25	1.58	16.63	....	7,286	13,115
DELTA COUNTY.																
Bowie, 1 mile from; T. 13 S., R. 91 W.; King mine, King bed (face of room 5, off left entry 3, 3,600 feet from mine mouth; 10-foot 2-inch bed and cut).	164	*15213	A	1	5.08	35.93	54.35	4.64	.60	....	....	....	....	2.3	....	....
				2	....	37.85	57.26	4.89	.62	....	....	....	....	....	....	....
				3	....	39.80	60.20	....	.66	....	....	....	....	....	....	....
Dominguez, 2½ miles northeast of; Wells Gulch, Gunnison Valley; sec. 17, T. 4 S., R. 3 E.; Wells Gulch mine (in main entry, about 200 feet from mouth of mine, 3-foot 11-inch bed, 1-foot 8-inch cut).	165	11106	B	1	3.45	38.81	51.75	5.99	.98	....	....	....	....	.5	7,281	13,107
				2	....	40.20	53.60	6.20	1.01	....	....	....	....	....	7,541	13,574
				3	....	42.86	57.14	....	1.08	....	....	....	....	....	8,039	14,471
EL PASO COUNTY.																
Calhan, 9 miles south of; sec. 18, T. 13 S., R. 62 W.; Mosby mine, Mosby bed (at face of entry, 30 feet from shaft bottom, 4-foot 8-inch bed, 4-foot 4-inch cut).	165	*10732	B	1	33.13	25.95	27.03	13.89	.30	6.48	37.25	.68	41.40	20.8	3,444	6,199
				2	....	38.81	40.42	20.77	.45	4.19	55.70	1.02	17.87	....	5,150	9,270
				3	....	48.98	51.02	....	.57	5.29	70.30	1.29	22.55	....	6,500	11,700
Pikeview, 1 mile north of; sec. 18, T. 13 S., R. 66 W.; Pikeview mine; "A" bed (face of north entry 2, 4,000 feet northwest of shaft, 10-foot 6-inch bed, 7-foot 5-inch cut.)	166	12099	A	1	26.20	29.67	37.67	6.46	.30	6.13	49.36	.66	37.09	7.8	4,640	8,352
				2	....	40.20	51.05	8.75	.41	4.36	66.88	.89	18.71	....	6,287	11,317
				3	....	44.06	55.94	....	.45	4.78	73.29	.98	20.50	....	6,890	12,402
Ramah, 3 miles southwest of; sec. 27, T. 11 S., R. 61 W.; Purdon mine, Purdon bed (face of workings, about 700 feet from entry, 6-foot 9½-inch bed, 6-foot 6-inch cut).	166	*10741	B	1	33.66	23.46	24.67	18.21	.32	6.31	33.21	.54	41.41	25.2	3,059	5,506
				2	....	35.36	37.19	27.45	.48	3.87	50.06	.81	17.33	....	4,611	8,300
				3	....	48.74	51.26	....	.66	5.33	69.00	1.12	23.89	....	6,356	11,441
FREMONT COUNTY.																
Chandler; sec. 22, T. 19 S., R. 70 W.; Chandler mine, Canon bed (face of east entry 9, 4-foot 8-inch bed, 4-foot 2-inch cut).	167	13393	A	1	9.70	38.08	46.19	6.03	.46	....	....	....	....	5.2	....	....
				2	....	42.17	51.15	6.68	.51	....	....	....	....	....	....	....
				3	....	45.19	54.81	....	.55	....	....	....	....	....	....	....

Same (face of main slope, 4-foot 9-inch bed and cut)	167	13394	A	1	10.74	37.92	44.36	6.98	.42	......	......	......	......	6.4	......	......
				2	......	42.48	49.70	7.82	.47	......	......	......	......	......	......	......
				3	......	46.08	53.92	......	.51	......	......	......	......	......	......	......
Same (composite of samples 13393 and 13394).....	167	13395	A	1	10.27	38.25	44.99	6.49	.42	5.38	65.45	1.08	21.18	5.8	6,342	11,416
				2	......	42.63	50.14	7.23	.47	4.74	72.94	1.20	13.42	......	7,068	12,722
				3	......	45.95	54.05	......	.51	5.11	78.62	1.29	14.47	......	7,619	13,714
Rockvale; secs. 25 and 36, T. 19 S., R. 70 W., and secs. 30 and 31, T. 19 S., R. 69 W.; Rockvale mine, Rockvale bed (face of room 39, off third dip, about 6,000 feet south of shaft, 3-foot 9½-inch bed and cut).	167	11938	A	1	8.10	35.22	49.63	7.05	.59	......	......	......	......	2.8	......	......
				2	......	38.32	54.01	7.67	.64	......	......	......	......	......	......	......
				3	......	41.50	58.50	......	.69	......	......	......	......	......	......	......
Same (face of room 31, off fourth dip, 6,200 feet south of shaft, 3-foot 9-inch bed and cut).	167	11939	A	1	8.97	33.78	45.24	12.01	.60	......	......	......	......	3.7	......	......
				2	......	37.11	49.70	13.19	.66	......	......	......	......	......	......	......
				3	......	42.75	57.25	......	.76	......	......	......	......	......	......	......
Same (composite of samples 11938 and 11939).....	167	11941	A	1	8.41	33.70	48.27	9.62	.59	5.28	64.09	1.18	19.24	3.3	6,272	11,290
				2	......	36.79	52.71	10.50	.64	4.75	69.97	1.29	12.85	......	6,848	12,326
				3	......	41.11	58.89	......	.72	5.31	78.18	1.44	14.35	......	7,651	13,772
GARFIELD COUNTY.																
Newcastle, 2 miles east of; sec. 1, T. 6 S., R. 91 W.; Vulcan mine, Allen bed (face of east entry on third level, 1,250 feet southeast of main opening, 10-foot 6-inch bed and cut).	168	12325	A	1	7.13	39.71	48.09	5.07	.41	......	......	......	......	3.6	......	......
				2	......	42.76	51.78	5.46	.44	......	......	......	......	......	......	......
				3	......	45.23	54.77	......	.47	......	......	......	......	......	......	......
Same (face of Allen entry on second level, 1,700 feet from mouth of entry, 10-foot bed and cut).	168	12326	A	1	......	......	......	......	......	......	......	......	......	......	......	......
				2	......	44.39	49.93	5.68	.51	......	......	......	......	......	......	......
				3	......	47.06	52.94	......	.54	......	......	......	......	......	......	......
Same (composite of samples 12325 and 12326).....	168	12327	A	1	7.12	40.75	46.94	5.19	.45	5.65	70.77	1.56	16.38	3.6	7,012	12,622
				2	......	43.88	50.53	5.59	.48	5.23	76.20	1.68	10.82	......	7,550	13,590
				3	......	46.48	53.52	......	.51	5.54	80.71	1.78	11.46	......	7,997	14,395
Same (face of room 15, 250 feet up from entry and 1,000 feet southeast of mine mouth, 10-foot bed and cut).	168	12420	A	1	4.45	42.05	49.56	3.94	.44	5.43	72.57	1.72	15.90	1.3	7,294	13,129
				2	......	44.01	51.87	4.12	.46	5.17	75.95	1.80	12.50	......	7,634	13,741
				3	......	45.90	54.10	......	.48	5.39	79.22	1.88	13.03	......	7,962	14,332
South Cañon; South Cañon mine, Wheeler bed (in main entry, 800 feet from portal, 5-foot 9-inch cut).	169	15166	A	1	7.40	38.99	48.12	5.49	.54	......	......	......	......	3.1	6,729	12,112
				2	......	42.11	51.96	5.93	.58	......	......	......	......	......	7,267	13,081
				3	......	44.76	55.24	......	.62	......	......	......	......	......	7,725	13,905
Same (face of main entry, 1,600 feet from portal, 17-foot 8-inch bed, 7-foot 8-inch cut).	169	15167	A	1	8.31	36.29	44.74	10.66	.52	5.19	62.87	1.43	19.33	3.4	6,231	11,216
				2	......	39.58	48.79	11.63	.57	4.66	68.57	1.56	13.01	......	6,796	12,233
				3	......	44.79	55.21	......	.65	5.27	77.59	1.77	14.72	......	7,690	13,842
GUNNISON COUNTY.																
Somerset, near; Somerset mine (face of room 46, off west entry 6, 10,000 feet north of entry mouth, 12-foot bed, 11-foot 9-inch cut).	170	12322	A	1	5.72	37.45	48.11	8.72	.39	......	......	......	......	1.6	......	......
				2	......	39.72	51.03	9.25	.41	......	......	......	......	......	......	......
				3	......	43.77	56.23	......	.45	......	......	......	......	......	......	......
Same (face of west entry 9, 4,500 feet north of entry mouth, 8-foot bed, 7-foot 9-inch cut).	170	12323	A	1	5.40	37.71	47.21	9.68	.48	......	......	......	......	1.7	......	......
				2	......	39.86	49.91	10.23	.51	......	......	......	......	......	......	......
				3	......	44.40	55.60	......	.57	......	......	......	......	......	......	......
Same (composite of samples 12322 and 12323).....	170	12324	A	1	5.61	37.60	47.57	9.22	.43	5.31	69.68	1.40	13.96	1.7	6,908	12,434
				2	......	39.83	50.40	9.77	.46	4.97	73.82	1.48	9.50	......	7,318	13,172
				3	......	44.14	55.86	......	.51	5.51	81.81	1.64	10.53	......	8,111	14,600

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
COLORADO—Continued.																
HUERFANO COUNTY.																
Oakview, 7 miles west of La Veta; sec. 10, T. 29 S., R. 68 W.; Oakdale mine (face of room 11, off second "cross," off main entry, 13-foot 10-inch bed and cut).	170	13526	A	1	6.79	38.59	45.78	8.84	0.56	....	....	....	....	2.6	....	....
				2	....	41.40	49.12	9.48	.60	....	....	....	....	....	....	....
				3	....	45.73	54.27	....	.66	....	....	....	....	....	....	....
Same (face of main entry, 6-foot 6-inch cut).......	170	13527	A	1	8.56	38.67	43.65	9.12	.57	....	....	....	....	4.6	....	....
				2	....	42.29	47.74	9.97	.62	....	....	....	....	....	....	....
				3	....	46.97	53.03	....	.69	....	....	....	....	....	....	....
Same (composite of samples 13526 and 13527).....	170	13529	A	1	7.80	38.50	44.80	8.90	.53	5.17	65.48	1.09	18.83	3.6	6,412	11,542
				2	....	41.76	48.59	9.65	.57	4.66	71.02	1.18	12.92	....	6,954	12,517
				3	....	46.22	53.78	....	.63	5.16	78.60	1.31	14.30	....	7,697	13,855
Same (sample of natural coke, face of north entry 4, 3-foot 6-inch cut).	170	13528	A	1	12.84	9.35	67.30	10.51	.44	....	....	....	....	11.5	6,365	11,457
				2	....	10.73	77.21	12.06	.50	....	....	....	....	....	7,303	13,145
				3	....	12.20	87.80	....	.57	....	....	....	....	....	8,304	14,947
JACKSON COUNTY.																
Coalmont; NW. ¼ SW. ¼ sec. 24, T. 7 N., R. 81 W.; Riach mine, Riach bed (at breast, about 225 feet east of opening, 77-foot bed, 7-foot cut).	171	12601	B	1	22.81	36.21	34.08	6.90	.70	6.30	51.74	1.14	33.22	5.9	5,004	9,007
				2	....	46.91	44.15	8.94	.91	4.88	67.03	1.48	16.76	....	6,483	11,669
				3	....	51.52	48.48	....	1.00	5.36	73.61	1.63	18.40	....	7,120	12,816
Higho, 8 miles northwest of; SW. ¼ SE. ¼ sec. 24, T. 8 N., R. 82 W.; Mitchell mine, Mitchell bed (275 feet northeast of entry mouth, 8-foot cut).	171	* 12539	B	1	20.72	33.65	35.67	9.96	.83	6.32	51.27	1.27	30.35	4.1	5,024	9,043
				2	....	42.45	44.99	12.56	1.06	5.07	64.67	1.60	15.05	....	6,337	11,407
				3	....	48.55	51.45	....	1.20	5.80	73.96	1.83	17.21	....	7,247	13,045
4 miles northwest of, 18 miles northwest of Walden; SE. ¼ SE. ¼ sec. 31, T. 10 N., R. 81 W.; Monahan mine, Monahan bed (face of slope, 125 feet east of mine mouth, 4-foot 6-inch bed and cut.)	172	*12540	B	1	19.70	32.77	44.06	3.47	.68	6.39	57.70	1.49	30.27	4.8	5,716	10,289
				2	....	40.81	54.87	4.32	.85	5.23	71.85	1.86	15.89	....	7,118	12,812
				3	....	42.65	57.35	....	.89	5.47	75.09	1.94	16.61	....	7,439	13,390
Walden, 5 miles northeast of; NE. ¼ NE. ¼ sec. 18, T. 9 N., R. 78 W.; McCallum mine, Sudduth bed (face of entry about 50 feet from mouth of entry, 35-foot bed, 7-foot cut).	172	* 12774	B	1	19.18	33.48	41.05	6.29	.27	5.75	56.34	.82	30.53	5.2	5,353	9,635
				2	....	41.42	50.80	7.78	.33	4.48	69.71	1.01	16.69	....	6,623	11,921
				3	....	44.92	55.08	....	.36	4.86	75.59	1.10	18.09	....	7,182	12,928

8½ miles from; NW. ¼ NE. ¼ sec. 35, T. 9 N., R. 78 W.; Marr mine, Sudduth bed (inside mouth of entry, water beyond, 53-foot bed, 6-foot cut).	172	*12415	B	1	16.72	33.02	46.67	3.59	.16	5.63	60.20	.70	29.72	7.4	5,755	10,359
				2	....	39.65	56.04	4.31	.19	4.53	72.29	.84	17.84	....	6,911	12,440
				3	....	41.43	58.57	....	.20	4.73	75.54	.88	18.65	....	7,222	13,000
7½ miles from; NW. ¼ SW. ¼ sec. 15, T. 9 N., R. 78 W.; Sudduth mine, Sudduth bed (face of main entry about 70 feet in, 50-foot bed, 20-foot cut).	172	*12414	B	1	19.97	32.54	42.49	5.00	.59	5.82	57.05	.78	30.76	8.2	5,419	9,751
				2	....	40.66	53.09	6.25	.74	4.50	71.28	.97	16.26	....	6,741	12,188
				3	....	43.37	56.63	....	.79	4.80	76.03	1.03	17.35	....	7,223	13,001
10½ miles southeast of; SW. ¼ NE. ¼ sec. 14, T. 8 N., R. 78 W.; Winscom mine, Winscom bed (in entry, about 125 feet northeast of mine mouth, 4-foot 5-inch bed, 2-foot 7-inch cut).	173	*12775	B	1	18.06	34.49	35.32	12.13	.80	5.82	52.44	1.11	27.70	5.2	5,153	9,275
				2	....	42.09	43.11	14.80	.98	4.65	64.00	1.35	14.22	....	6,289	11,320
				3	....	49.40	50.60	....	1.15	5.46	75.12	1.58	16.69	....	7,381	13,286
JEFFERSON COUNTY.																
Morrison, 2½ miles from, 6 miles from Golden; secs. 23, 24, 25, 26, T. 4 S., R. 70 W.; White Ash mine, Jumbo bed (30 feet up from second level, 75 feet from shaft, 16-foot bed, 10-foot 6-inch cut).	173	*16615	A	1	28.67	30.69	34.85	5.79	.75	6.35	47.94	.69	38.48	15.8	4,533	8,159
				2	....	43.02	48.86	8.12	1.05	4.43	67.21	.97	18.22	....	6,355	11,439
				3	....	46.82	53.18	....	1.14	4.82	73.15	1.06	19.83	....	6,917	12,451
LA PLATA COUNTY.																
Durango, 1½ miles southwest of; T. 35 N., R. 10 W.; San Juan mine, Mesaverde formation (face of main entry, 1,200 feet west of mine mouth, 3-foot 1-inch bed, 3-foot cut).	174	14770	A	1	3.83	34.42	55.29	6.46	.68	....	....	....	....	2.7	....	....
				2	....	35.79	57.49	6.72	.71	....	....	....	....	....	....	....
				3	....	38.37	61.63	....	.76	....	....	....	....	....	....	....
Same (face of room 6, off first dip entry south, 300 feet from mine mouth, 5-foot 2-inch bed, 5-foot cut).	174	14771	A	1	2.24	34.06	56.49	7.21	.74	....	....	....	....	1.2	....	....
				2	....	34.84	57.78	7.38	.76	....	....	....	....	....	....	....
				3	....	37.62	62.38	....	.82	....	....	....	....	....	....	....
Same (composite of samples 14770 and 14771).....	174	14772	A	1	3.01	34.77	55.31	6.91	.74	5.50	76.36	1.50	8.99	2.0	7,619	13,714
				2	....	35.85	57.03	7.12	.76	5.33	78.73	1.55	6.51	....	7,855	14,139
				3	....	38.60	61.40	....	.82	5.74	84.77	1.67	7.00	....	8,457	15,223
Hesperus, 16 miles west of Durango; sec. 14, T. 35 N., R. 11 W.; Hesperus mine, Mesaverde formation (face of room 3, off right entry 4; 4,000 feet from mine mouth, 5-foot 2½-inch bed, 5-foot 2-inch cut).	175	14773	A	1	7.57	38.01	47.75	6.67	.70	....	....	....	....	4.2	....	....
				2	....	41.12	51.66	7.22	.76	....	....	....	....	....	....	....
				3	....	44.32	55.68	....	.82	....	....	....	....	....	....	....
Same (room 5, off left entry 5; 4,000 feet west of mine mouth, 4-foot 7-inch bed, 4-foot 6-inch cut).	175	14774	A	1	6.59	38.94	49.31	5.16	.62	....	....	....	....	3.4	....	....
				2	....	41.69	52.79	5.52	.66	....	....	....	....	....	....	....
				3	....	44.12	55.88	....	.70	....	....	....	....	....	....	....
Same (composite of samples 14773 and 14774).....	175	14775	A	1	7.08	38.10	48.97	5.85	.60	5.68	70.85	1.47	15.55	3.8	7,040	12,672
				2	....	41.00	52.70	6.30	.65	5.26	76.25	1.58	9.96	....	7,576	13,637
				3	....	43.76	56.24	....	.69	5.61	81.37	1.69	10.64	....	8,085	14,553
Perins, 5 miles southeast of Durango; secs. 11 and 14, T. 35 N., R. 10 W.; Perins Peak mine, Mesaverde formation (room 3, off main entry, 3,200 feet from mine mouth, 5-foot 1-inch bed and cut).	175	14776	A	1	6.43	39.00	50.88	3.69	.80	....	....	....	....	4.4	....	....
				2	....	41.68	54.38	3.94	.85	....	....	....	....	....	....	....
				3	....	43.39	56.61	....	.88	....	....	....	....	....	....	....
Same (face of room 13, off main back entry, 5-foot 5-inch bed and cut).	175	14777	A	1	4.05	39.52	49.04	7.39	4.02	....	....	....	....	2.4	....	....
				2	....	41.19	51.11	7.70	4.19	....	....	....	....	....	....	....
				3	....	44.63	55.37	....	4.54	....	....	....	....	....	....	....
Same (composite of samples 14776 and 14777).....	175	14778	A	1	5.31	38.98	50.03	5.68	2.11	5.66	72.83	1.58	12.14	3.4	7,342	13,216
				2	....	41.17	52.83	6.00	2.23	5.35	76.91	1.67	7.84	....	7,754	13,957
				3	....	43.80	56.20	....	2.37	5.69	81.82	1.78	8.34	....	8,249	14,848

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
COLORADO—Continued.																
LAS ANIMAS COUNTY.																
Cokedale; sec. 25, T. 33 S., R. 65 W.; Cokedale mine, Cokedale bed (face of room 8, off west entry 4, 6-foot 1-inch bed, 5-foot 4-inch cut).	176	11768	A	1	2.39	25.07	54.80	17.74	0.52					1.6		
				2		25.68	56.15	18.17	.53							
				3		31.38	68.62		.65							
Same (face of room 21, off west entry 1, 6-foot 10½-inch bed, 4 foot 9½-inch cut).	176	11769	A	1	2.74	26.77	56.22	14.27	.56					2.0		
				2		27.52	57.81	14.67	.58							
				3		32.25	67.75		.68							
Same (face of room 10, off west entry 6, 6-foot 1-inch bed, 5-foot 3-inch cut).	176	11770	A	1	2.12	24.85	53.39	19.64	.47					1.6		
				3		25.39	54.54	20.07	.48							
				3		31.76	68.24		.60							
Same (composite of samples 11768, 11769, and 11770).	176	11771	A	1	2.33	25.82	54.58	17.27	.52	4.62	69.14	1.07	7.38	1.7	6,854	12,337
				2		26.44	55.88	17.68	.53	4.46	70.79	1.10	5.44		7,018	12,632
				3		32.12	67.88		.64	5.42	86.00	1.34	6.60		8,525	15,345
Delagua, 20 miles north of Trinidad; sec. 15, T. 31 S., R. 65 W.; Delagua mine, Delagua bed (face of room 1, off east entry 24, off north entry 3, 5-foot 8-inch bed and cut).	177	11435	A	1	3.19	35.00	53.22	8.59	.57					1.9		
				2		36.16	54.97	8.87	.59							
				3		39.68	60.32		.65							
Same (face of east entry 18, off north entry 4, 6-foot 6-inch bed and cut).	177	11436	A	1	3.53	34.23	52.05	10.19	.47					2.3		
				2		35.48	53.96	10.56	.49							
				3		39.67	60.33		.55							
Same (face of room 5, west entry 3, off north entry 4, 5-foot 8-inch bed, 5-foot 4-inch cut).	177	11437	A	1	3.14	33.37	50.96	12.53	.50					2.1		
				2		34.45	52.61	12.94	.52							
				3		39.57	60.43		.60							
Same (composite of samples 11435, 11436, and 11437).	177	11443	A	1	3.31	34.34	52.03	10.32	.55	5.09	71.57	1.38	11.09	2.1	7,063	12,713
				2		35.51	53.82	10.67	.57	4.88	74.02	1.43	8.43		7,305	13,149
				3		39.75	60.25		.64	5.46	82.86	1.60	9.44		8,177	14,719
Forbes; secs. 15 and 16, T. 32 S., R. 64 W.; Forbes No. 9 (or Cox) mine, Walsen bed (face of first crosscut off west entry 1, 2,800 feet southwest of slope opening, 4-foot 4-inch bed and cut).	177	11926	A	1	2.14	30.50	56.99	10.37	.65					1.2		
				2		31.17	58.23	10.60	.66							
				3		34.87	65.13		.74							

Same (face of west entry 6, 3,800 feet southwest of slope opening, 3-foot 9-inch bed and cut).	177	11927	A	1	2.29	30.53	56.30	10.88	.60					1.5		
				2		31.24	57.63	11.13	.61							
				3		35.15	64.85		.69							
Same (room 6, off west entry 4, 3,200 feet southwest of slope opening, 3-foot 11-inch bed and cut).	178	11929	A	1	1.96	30.22	57.54	10.28	.59					1.1		
				2		30.82	58.69	10.49	.60							
				3		34.43	65.57		.67							
Same (composite of samples 11926, 11927, and 11929).	178	11930	A	1	2.15	30.67	56.64	10.54	.61	5.01	75.31	1.31	7.22	1.3	7,512	13,522
				2		31.34	57.89	10.77	.62	4.87	76.97	1.34	5.43		7,677	13,819
				3		35.12	64.88		.69	5.46	86.26	1.50	6.09		8,604	15,487
Hastings; sec. 25, T. 32 S., R. 64 W.; Hastings mine, Hastings bed (pillar between south entries 7 and 8, 8,300 feet from mouth of slope, 10-foot 5-inch bed, 9-foot 5-inch cut).	178	12028	A	1	2.85	32.65	55.26	9.24	.62	5.16	74.39	1.35	9.24	1.6	7,368	13,262
				2		33.61	56.88	9.51	.64	4.99	76.57	1.39	6.90		7,584	13,651
				3		37.14	62.86		.71	5.51	84.62	1.54	7.62		8,381	15,086
Same (in south entry 7, at fifth crosscut, 8,200 feet from mouth of slope).	178	12031	A	1	10.12	26.06	41.99	21.83	.51					8.9		
				2		28.99	46.72	24.29	.57							
				3		38.29	61.71		.75							
Berwind (?) bed (face of slope, 5-foot 1-inch bed and cut).	178	12029	A	1	2.52	32.21	48.25	17.02	.92	4.92	66.47	1.20	9.47	1.7	6,661	11,990
				2		33.04	49.50	17.46	.94	4.76	68.19	1.23	7.42		6,834	12.301
				3		40.03	59.97		1.14	5.77	82.61	1.49	8.99		8,279	14,902
Morley; Morley mine, Engle-Starkville bed (face of room 2, third plane, off right entry 4, 5-foot 3-inch bed and cut).	179	13361	A	1	2.13	32.68	53.57	11.62	.69					1.6		
				2		33.39	54.74	11.87	.71							
				3		37.89	62.11		.81							
Same (face of left entry 5, 6-foot 1-inch bed and cut).	179	13362	A	1	3.60	31.71	45.67	19.02	1.12					3.1		
				2		32.89	47.38	19.73	1.16							
				3		40.97	59.03		1.45							
Same (face of right entry 4, 3,000 feet from mine mouth, 5-foot 1-inch bed and cut).	179	13363	A	1	4.25	31.91	47.31	16.53	.56					3.7		
				2		33.33	49.41	17.26	.58							
				3		40.28	59.72		.70							
Same (composite of samples 13361, 13362, and 13363).	179	13364	A	1	3.32	31.98	49.09	15.61	.82	4.93	67.66	1.23	9.75	2.8	6,865	12,357
				2		33.08	50.77	16.15	.85	4.72	69.98	1.27	7.03		7,100	12,780
				3		39.45	60.55		1.01	5.63	83.46	1.51	8.39		8,467	15,241
Primero; secs. 23, 24, 25, and 26, T. 33 S., R. 66 W.; First East Primero mine, Primero bed (pillar of right entry 6, between rooms 1 and 2, off entry B-8, off blind entry 3, 6-foot 1-inch bed and cut).	180	W31143	A	1	1.9	30.8	51.5	15.8	.80						7,011	12,620
				2		31.4	52.5	16.1	.80						7,144	12,860
				3		37.4	62.6		.95						8,517	15,330
Same (pillar of room 5, off west entry B-9, off blind entry 1, 5-foot 1-inch bed and cut).	180	W31144	A	1	1.4	33.0	53.9	11.7	.50						7,411	13,340
				2		33.5	54.6	11.9	.50						7,517	13,530
				3		38.0	62.0		.57						8,533	15,360
Same (chain pillar of east entry B-9, 6-foot 5-inch bed and cut).	180	W31145	A	1	1.6	29.6	55.4	13.4	.50						7,233	13,020
				2		30.1	56.3	13.6	.50						7,356	13,240
				3		34.8	65.2		.58						8,511	15,320
Same (face of room 4, off east entry B-10, 5-foot 11-inch bed and cut).	180	W31146	A	1	1.4	31.7	55.0	11.9	.45						7,428	13,370
				2		32.2	55.7	12.1	.45						7,539	13,570
				3		36.6	63.4		.51						8,578	15,440
Same (face of room 3, off east entry 11, 7-foot 4-inch bed and cut).	181	W31147	A	1	2.0	29.0	56.1	12.9	.60						7,256	13,060
				2		29.6	57.2	13.2	.60						7,406	13,330
				3		34.1	65.9		.69						8,533	15,360
Same (chain pillar of east entry B-10, off blind entry 3, 7-foot bed and cut).	181	W31149	A	1	1.0	28.3	56.5	14.2	.55						7,233	13,020
				2		28.6	57.1	14.3	.55						7,306	13,150
				3		33.4	66.6		.64						8,528	15,350

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
COLORADO—Continued.																
LAS ANIMAS COUNTY—continued.																
Primero; secs. 23, 24, 25, and 26, T. 33 S., R. 66 W.; First East Primero mine, Primero bed (composite of samples 31143-31149).	181	14060	A	1	1.53	30.78	54.37	13.32	0.53	4.98	73.10	1.71	6.36	0.6	7,221	12,998
				2		31.26	55.21	13.53	.54	4.88	74.23	1.74	5.08		7,333	13,199
				3		36.15	63.85		.62	5.64	85.85	2.01	5.88		8,481	15,266
Same (taken from cars while being loaded at the mine).	182	W31131	C	1	1.2	30.5	55.7	12.6	.60						7,311	13,160
				2		30.9	56.3	12.8	.60						7,400	13,320
				3		35.4	64.6		.69						8,483	15,270
Same (taken from same cars as sample 31131).....	182	W31132	C	1	1.2	30.5	55.7	12.6	.50						7,328	1,3190
				2		30.9	56.3	12.8	.50						7,417	1,3350
				3		35.4	64.6		.57						8,506	1,5310
Same (taken from four cars while being loaded at the mine).	182	W31133	C	1	1.0	30.1	55.1	13.8	.50						7,228	1,3010
				2		30.4	55.7	13.9	.50						7,300	1,3140
				3		35.3	64.7		.58						8,478	1,5260
Same (taken from same cars as sample 31133).....	182	W31134	C	1	1.0	30.7	54.8	13.5	.50						7,233	1,3020
				2		31.0	55.4	13.6	.50						7,311	1,3160
				3		35.9	64.1		.58						8,567	1,5240
Same (taken from four cars while being loaded at the mine).	182	W31135	C	1	1.0	30.3	58.0	10.7	.50						7,367	1,3260
				2		30.6	58.6	10.8	.50						7,444	1,3400
				3		34.3	65.7		.56						8,344	1,5020
Same (taken from same cars as sample 31135).....	182	W31136	C	1	1.0	30.4	56.1	12.5	.50						7,356	1,3240
				2		30.7	56.7	12.6	.50						7,428	1,3370
				3		35.1	64.9		.57						8,500	1,5300
Same (taken from four cars while being loaded at the mine).	182	W31137	C	1	1.1	29.9	56.0	13.0	.55						7,317	13,170
				2		30.2	56.7	13.1	.55						7,394	13,310
				3		34.8	65.2		.63						8,511	15,320
Same (taken from same cars as sample 31137).....	182	W31138	C	1	1.1	30.3	55.2	13.4	.50						7,283	13,110
				2		30.6	55.8	13.6	.50						7,367	13,260
				3		35.4	64.6		.58						8,522	15,340
First North Primero mine, Primero bed (room 5, off back entry 3, off entry A-12, 7-foot 11-inch bed and cut).	181	W31148	A	1	1.6	27.1	59.1	12.2	.50						7,367	13,260
				2		27.5	60.1	12.4	.50						7,489	13,480
				3		31.4	68.6		.57						8,557	15,390
Same (pillar of room 7, off entry A-12, 7-foot bed, 6-foot 11-inch cut).	181	W31150	A	1	1.5	31.3	53.3	13.4	.55						7,217	12,990
				2		32.3	54.1	13.6	.55						7,328	13,190
				3		37.4	62.6		.64						8,483	15,270

Same (face of entry B-9, 7-foot 2-inch bed, 7-foot ½-inch cut).	181	W31151	A	1	1.9	30.6	54.7	12.8	.55						7,317	13,170
				2		31.2	55.8	13.0	.55						7,456	13,420
				3		35.9	64.1		.63						8,572	15,430
Same (composite of samples W31148, W31150, and W31151).	181	14059	A	1	1.42	32.31	53.33	12.94	.50	5.04	73.46	1.71	6.35	.5	7,267	13,081
				2		32.78	54.09	13.13	.51	4.95	74.52	1.73	5.16		7,372	13,270
				3		37.73	62.27		.59	5.70	85.78	1.99	5.94		8,486	15,275
Same (taken from four cars while being loaded at the mine).	182	W31033	C	1	2.1	29.5	54.7	13.7	.60						7,150	12,870
				2		30.1	55.9	14.0	.60						7,306	13,150
				3		35.0	65.0		.70						8,494	15,290
Same (taken from same cars as sample 31033).....	182	W31034	C	1	1.1	30.8	54.0	14.1	.50						7,189	12,940
				2		31.1	54.6	14.3	.50						7,272	13,090
				3		36.3	63.7		.58						8,483	15,270
Same (taken from four cars while being loaded at the mine).	182	W31035	C	1	1.2	31.4	55.3	12.1	.59						7,389	13,300
				2		31.8	56.0	12.2	.50						7,478	13,460
				3		36.2	63.8		.57						8,517	15,330
Same (taken from same cars as sample 31035).....	182	W31036	C	1	1.1	31.2	55.4	12.3	.50						7,344	13,220
				2		31.6	56.0	12.4	.50						7,428	13,370
				3		36.1	63.9		.57						8,478	15,260
First West Primero mine, Primero bed (face of north entry 2, 4-foot 9-inch bed, 4-foot 5½-inch cut).	181	W31039	A	1	2.2	30.7	51.8	15.3	.65						7,011	12,620
				2		31.4	53.0	15.6	.65						7,167	12,900
				3		37.2	62.8		.77						8,489	15,280
Same (face of room 10, off north entry 1, 5-foot 4-inch bed and cut).	181	W31040	A	1	1.9	31.0	50.7	16.4	.60						6,933	12,480
				2		31.6	51.7	16.7	.60						7,072	12,730
				3		37.9	62.1		.72						8,489	15,280
Same (face of room 9, off south entry 2, 4-foot 9-inch bed, 4-foot 7-inch cut).	181	W31041	A	1	2.0	31.1	52.3	14.6	.55						7,072	12,730
				2		31.7	53.4	14.9	.55						7,217	12,990
				3		37.3	62.7		.65						8,483	15,270
Same (face of room 5, off right entry 1, off first west air course, 5-foot bed, 4-foot 7½-inch cut).	181	W31042	A	1	2.1	31.1	54.0	12.8	.60						7,283	13,110
				2		31.8	55.1	13.1	.60						7,439	13,390
				3		36.6	63.4		.69						8,561	15,410
Same (face of room 17, off right entry 2, off first west air course, 5-foot bed and cut).	181	W31043	A	1	2.8	30.4	51.2	15.6	.55						6,922	12,460
				2		31.3	52.7	16.0	.55						7,117	12,810
				3		37.3	62.7		.65						8,472	15,250
Same (composite of samples W31039, W31040, W31041, W31042, and W31043).	181	14058	A	1	2.02	30.38	52.80	14.80	.60	4.94	71.36	1.66	6.64	1.0	7,052	12,694
				2		31.01	53.89	15.10	.61	4.82	72.83	1.69	4.95		7,197	12,955
				3		36.53	63.47		.72	5.68	85.79	1.99	5.82		8,477	15,259
Same (taken from 4 cars while being loaded at the mine.	182	W31037	C	1	1.8	29.6	51.9	17.2	.50						6,900	12,420
				2		30.0	52.6	17.4	.50						6,994	12,590
				3		36.3	63.7		.61						8,467	15,240
Same (taken from same cars as sample 31037)....	182	W31038	C	1	1.2	30.1	51.5	17.2	.60						6,900	12,420
				2		30.5	52.1	17.4	.60						6,989	12,580
				3		36.9	63.1		.73						8,461	15,230
Same (taken from car while being loaded at the mine).	182	W31139	C	1	1.2	29.8	52.0	17.0	.65						6,944	12,500
				2		30.2	52.6	17.2	.65						7,028	12,650
				3		36.5	63.5		.79						8,483	15,270
Same (taken from same car as sample 31139)......	182	W31140	C	1	1.1	29.2	52.7	17.0	.60						6,928	12,470
				2		29.5	53.3	17.2	.60						7,006	12,610
				3		35.6	64.4		.72						8,461	15,230
Same (taken from 4 cars while being loaded at the mine).	182	W31141	C	1	1.4	28.9	53.0	16.7	.60						6,933	12,480
				2		29.3	53.8	16.9	.60						7,033	12,660
				3		35.3	64.7		.72						8,461	15,230

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
COLORADO—Continued.																
LAS ANIMAS COUNTY—continued.																
Primero; secs. 23, 24, 25, and 26, T. 33 S., R. 66 W.; First West Primero mine, Primero bed (taken from same cars as sample 31141).	182	W31142	C	1	1.3	28.6	52.3	17.8	0.60	......	......	......	......	......	6,828	12,290
				2	......	29.0	53.0	18.0	.60	......	......	......	......	......	6,922	12,460
				3	......	35.4	64.6	......	.73	......	......	......	......	......	8,439	15,190
Primero mine, Primero bed (face of room 1 off second blind back entry, 5-foot 10-inch bed and cut).	180	12025	A	1	2.36	31.07	56.65	9.92	.59	5.06	74.54	1.65	8.24	1.6	7,503	13,505
				2	......	31.82	58.02	10.16	.60	4.92	76.34	1.69	6.29	......	7,685	13,833
				3	......	35.42	64.58	......	.67	5.48	84.97	1.88	7.00	......	8,554	15,397
Same (face of first west slope, 1,500 feet from mouth).	180	12027	A	1	3.98	27.96	50.50	17.56	.51	......	......	......	......	3.0	......	......
				2	......	29.12	52.59	18.29	.53	......	......	......	......	......	......	......
				3	......	35.64	64.36	......	.65	......	......	......	......	......	......	......
Starkville; Starkville mine, Engle-Starkville bed (rib of west entry 8, 30 feet outby slant from new entry, 7-foot 2⅛-inch bed, 6-foot 9⅛-inch cut).	182	11134	A	1	2.07	29.15	55.56	13.22	.57	4.64	71.77	1.15	8.65	.8	7,139	12,850
				2	......	29.77	56.73	13.50	.58	4.50	73.28	1.17	6.97	......	7,290	13,122
				3	......	34.42	65.58	......	.67	5.20	84.72	1.35	8.06	......	8,428	15,170
Same (south entry 3, near second crosscut off new entry, 2-foot 11½-inch cut).	182	11135	A	1	1.79	28.67	55.33	14.21	.62	4.69	71.76	1.15	7.57	.8	7,074	12,733
				2	......	29.19	56.34	14.47	.63	4.57	73.07	1.17	6.09	......	7,203	12,965
				3	......	34.13	65.87	......	.74	5.34	85.43	1.37	7.12	......	8,422	15,160
Same (entry I-5 between rooms 26 and 27, on pillar, 15,000 feet southwest of mine mouth, 5-foot 9-inch bed and cut).	182	12196	A	1	1.81	31.16	55.49	11.54	.60	......	......	......	......	.7	......	......
				2	......	31.73	56.52	11.75	.61	......	......	......	......	......	......	......
				3	......	35.95	64.05	......	.69	......	......	......	......	......	......	......
Same (face of south entry 1, 1,200 feet southeast of mine mouth, 4-foot 2-inch cut).	182	12197	A	1	2.97	25.89	49.80	21.34	.54	......	......	......	......	1.8	......	......
				2	......	26.68	51.33	21.99	.56	......	......	......	......	......	......	......
				3	......	34.20	65.80	......	.72	......	......	......	......	......	......	......
Same (composite of samples 12196 and 12197).....	183	12198	A	1	2.50	28.85	52.09	16.56	.55	4.79	69.12	1.10	7.88	1.3	6,881	12,386
				2	......	29.59	53.43	16.98	.56	4.63	70.89	1.13	5.81	......	7,057	12,703
				3	......	35.64	64.36	......	.67	5.58	85.39	1.36	7.00	......	8,500	15,300
Tollerburg; Toller mine, Berwind bed; (face of slope, 2,500 feet from shaft bottom, 7-foot bed and cut).	183	13253	A	1	6.31	30.15	49.45	14.09	.65	5.10	67.46	1.31	11.39	5.5	6,747	12,145
				2	......	32.18	52.78	15.04	.69	4.70	72.00	1.40	6.17	......	7,201	12,962
				3	......	37.88	62.12	......	.81	5.53	84.74	1.65	7.27	......	8,476	15,257
Same (face of north entry 5, 3,000 feet from shaft bottom, 7-foot bed and cut).	183	13254	A	1	5.43	33.01	52.93	8.63	.63	5.46	72.98	1.35	10.95	4.6	7,354	13,237
				2	......	34.90	55.97	9.13	.67	5.14	77.17	1.43	6.46	......	7,776	13,997
				3	......	38.41	61.59	......	.74	5.66	84.93	1.57	7.10	......	8,557	15,403

MESA COUNTY.																
Grand Junction, 1 mile south of; sec. 26, T. 1 S., R. 1 W.; prospect (in entry, about 125 feet from opening, 3-foot 11-inch bed, 1-foot 11-inch cut).	184	11108	B	1	5.13	27.94	38.74	28.19	1.14	....	....	....	....	1.9	5,147	9,265
				2	....	29.45	40.83	29.72	1.20	....	....	....	....	....	5,425	9,765
				3	....	41.90	58.10	....	1.72	....	....	....	....	....	7,719	13,894
Same (in entry, about 125 feet from opening, 2-foot 10-inch bed, 10-inch cut).	184	11109	B	1	5.69	26.34	34.21	33.76	.80	....	....	....	....	2.1	4,648	8,366
				2	....	27.93	36.27	35.80	.85	....	....	....	....	....	4,928	8,870
				3	....	43.50	56.50	....	1.32	....	....	....	....	....	7,676	13,817
5 miles southeast of; sec. 5, T. 2 S., R. 1 E.; prospect (in entry, 100 feet from opening, 2-foot 6-inch bed, cut taken from upper 15 inches).	184	11104	B	1	3.51	39.19	51.33	5.97	1.67	....	....	....	....	.7	7,251	13,052
				2	....	40.62	53.19	6.19	1.73	....	....	....	....	....	7,515	13,527
				3	....	43.30	56.70	....	1.84	....	....	....	....	....	8,011	14,420
Same (position and bed same as sample 11104, cut taken from lower 12 inches).	184	11105	B	1	3.18	29.11	45.29	22.42	1.19	....	....	....	....	.7	5,840	10,512
				2	....	30.06	46.78	23.16	1.23	....	....	....	....	....	6,032	10,858
				3	....	39.12	60.88	....	1.60	....	....	....	....	....	7,850	14,130
MOFFAT COUNTY.																
Axial, 6 miles west of; NW. ¼ sec. 14, T. 4 N., R. 94 W.; Battle Era mine (in main entry, 130 feet northeast of mine mouth; 11-foot 6-inch bed, 11-foot cut).	185	*14528	B	1	14.67	36.65	42.56	6.12	.89	5.54	59.06	1.33	27.06	1.3	5,634	10,141
				2	....	42.95	49.88	7.17	1.04	4.58	69.21	1.56	16.44	....	6,602	11,884
				3	....	46.27	53.73	....	1.12	4.93	74.55	1.68	17.72	....	7,112	12,802
5½ miles northwest of; sec. 13, T. 4 N., R. 94 W.; Ed. Collom mine (75 feet northwest of opening, 13-foot 6-inch bed and cut).	185	*14529	B	1	14.84	38.68	42.66	3.82	.66	5.87	61.49	1.17	26.99	2.0	5,989	10,780
				2	....	45.42	50.09	4.49	.78	4.96	72.21	1.37	16.19	....	7,033	12,659
				3	....	47.55	52.45	....	.82	5.19	75.60	1.43	16.96	....	7,364	13,255
Sec. 14, T. 3 N., R. 93 W.; James mine (in main entry, 100 feet north of opening, 6-foot 4-inch bed and cut).	185	12704	B	1	11.42	40.24	44.47	3.87	.54	5.57	65.27	1.32	23.43	.8	6,310	11,358
				2	....	45.43	50.20	4.37	.61	4.85	73.68	1.49	15.00	....	7,123	12,821
				3	....	47.51	52.49	....	.64	5.07	77.05	1.56	15.68	....	7,449	13,408
1½ miles south of; sec. 2, T. 3 N., R. 93 W.; Joe Collom mine, Collom bed (in mine, 400 feet northwest of mine mouth; 20-foot 1-inch bed, 8-foot 11-inch cut).	186	*14543	B	1	11.91	40.64	45.24	2.21	.32	....	....	....	....	2.1	6,451	11,612
				2	....	46.13	51.36	2.51	.36	....	....	....	....	....	7,323	13,181
				3	....	47.32	52.68	....	.37	....	....	....	....	....	7,511	13,520
2 miles east of; sec. 31, T. 4 N., R. 92 W.; Shafer mine (189 feet northeast of the mine mouth, 14-foot bed, 13-foot 9-inch cut).	186	14909	B	1	13.51	38.50	44.10	3.89	.59	5.80	63.11	1.23	25.38	4.6	6,124	11,023
				2	....	44.51	50.99	4.50	.68	4.97	72.97	1.42	15.46	....	7,081	12,746
				3	....	46.61	53.39	....	.71	5.20	76.41	1.49	16.19	....	7,415	13,347
Craig, 3 miles southeast of; Walker mine (face of main entry, 300 feet west of mine mouth, 8-foot bed).	186	12400	A	1	15.54	37.42	39.56	7.48	.97	....	....	....	....	2.9	5,751	10,352
				2	....	44.31	46.83	8.86	1.15	....	....	....	....	....	6,809	12,256
				3	....	48.62	51.38	....	1.26	....	....	....	....	....	7,471	13.448
MONTEZUMA COUNTY.																
Cortez, one-half mile west of, 12 miles southwest of Dolores; sec. 23, T. 36 N., R. 16 W.; Cortez mine, lowest bed in Dakota formation (in tunnel, 50 feet from mouth, 4-foot 8½-inch bed, 1-foot 7-inch cut).	187	12785	B	1	8.05	32.27	45.50	14.18	.57	4.72	60.29	1.01	19.23	4.5	5,800	10,439
				2	....	35.09	49.49	15.42	.62	4.17	65.57	1.10	13.12	....	6,308	11,354
				3	....	41.49	58.51	....	.73	4.93	77.52	1.30	15.52	....	7,458	13,424
16 miles southwest of; sec. 3, T. 35 N., R. 18 W.; Hamilton prospect, Dakota formation (near surface in open cut, 2-foot 8-inch bed, 1-foot 7-inch cut).	187	12686	B	1	7.74	21.00	58.68	12.58	.46	....	....	....	....	3.2	6,033	10,859
				2	....	22.76	63.60	13.64	.50	....	....	....	....	....	6,539	11,770
				3	....	26.35	73.65	....	.58	....	....	....	....	....	7,572	13,630

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
COLORADO—Continued.																
MONTEZUMA COUNTY—continued.																
Cortez, 10 miles southeast of; SW. ¼ NE. ¼ sec. 33, T. 35 N., R. 16 W.; Jackson mine, lower bed of Mesaverde formation, probably Spencer (face of room 4, off north main entry, 200 feet from opening, 2-foot 10-inch bed and cut).	188	16472	A	1	12.20	41.33	40.38	6.09	0.44	......	......	......	......	4.1	6,309	11,356
				2	......	47.07	45.99	6.94	.50	......	......	......	......	......	7,186	12,935
				3	......	50.58	49.42	......	.54	......	......	......	......	......	7,722	13,900
Same (face of main entry, 170 feet north from opening, 2-foot 7-inch bed and cut).	188	16473	A	1	12.31	42.23	40.51	4.95	.48	......	......	......	......	3.8	6,399	11,518
				2	......	48.16	46.20	5.64	.55	......	......	......	......	......	7,297	13,135
				3	......	51.04	48.96	......	.58	......	......	......	......	......	7,733	13,919
Same (face of south entry, 95 feet from opening, 5-foot 3-inch bed, 2-foot 7-inch cut).	188	16474	A	1	13.79	40.67	39.93	5.61	.49	......	......	......	......	4.9	6,176	11,117
				2	......	47.18	46.31	6.51	.57	......	......	......	......	......	7,164	12,895
				3	......	50.46	49.54	......	.61	......	......	......	......	......	7,663	13,793
Same (composite of samples 16472, 16473, and 16474).	189	16475	A	1	12.77	41.34	40.24	5.65	.50	6.10	63.47	1.34	22.94	4.3	6,310	11,358
				2	......	47.39	46.13	6.48	.57	5.37	72.76	1.54	13.28	......	7,234	13,021
				3	......	50.67	49.33	......	.61	5.74	77.80	1.65	14.20	......	7,735	13,923
1½ miles southeast of; sec. 35, T. 36 N., R. 16 W.; Mowry mine, Dakota formation (near face, 75 feet from mine mouth, 2-foot 8-inch bed, 2-foot 2-inch cut).	189	12586	B	1	4.84	34.56	42.30	18.30	7.56	4.85	59.90	1.11	8.28	3.1	6,147	11,065
				2	......	36.32	44.45	19.23	7.94	4.53	62.95	1.17	4.18	......	6,460	11,628
				3	......	44.97	55.03	......	9.83	5.61	77.94	1.45	5.17	......	7,998	14,396
Same (in main entry, 100 feet from mine mouth, 2-foot 2-inch bed, 1-foot 9-inch cut).	189	16479	A	1	3.89	37.01	46.58	12.52	7.04	4.96	66.19	1.16	8.13	2.4	6,856	12,341
				2	......	38.51	48.46	13.03	7.33	4.71	68.87	1.21	4.85	......	7,134	12,841
				3	......	44.28	55.72	......	8.43	5.42	79.19	1.39	5.57	......	8,203	14,765
7 miles from; sec. 25, T. 35 N., R. 16 W.; Todd mine, lowest bed of Mesaverde formation (face of room 9; 340 feet from entry mouth, 2-foot 10-inch bed and cut).	190	16476	A	1	10.97	40.30	42.28	6.45	.81	......	......	......	......	3.7	6,462	11,632
				2	......	45.26	47.50	7.24	.91	......	......	......	......	......	7,258	13,064
				3	......	48.79	51.21	......	.98	......	......	......	......	......	7,825	14,085
Same (face of room 10; 330 feet from entry mouth, 3-foot bed and cut).	190	16477	A	1	13.07	39.62	39.72	7.59	.65	......	......	......	......	5.1	6,194	11,149
				2	......	45.57	45.70	8.73	.75	......	......	......	......	......	7,125	12,825
				3	......	49.93	50.07	......	.82	......	......	......	......	......	7,806	14,051
Same (composite of samples 16476 and 16477) ...	190	16478	A	1	12.05	39.65	41.27	7.03	.73	5.88	63.43	1.38	21.55	4.4	6,310	11,358
				2	......	45.08	46.93	7.99	.83	5.16	72.12	1.57	12.33	......	7,174	12,913
				3	......	48.99	51.01	......	.90	5.61	78.38	1.71	13.40	......	7,797	14,035

Dolores, 11 miles south of; sec. 20, T. 30 N., R. 15 W.; prospect, Dakota formation (near face of entry, 20 feet from mouth, 4-foot 4-inch bed, 2-foot 6½-inch cut).	190	*12551	B	1	10.29	31.81	49.73	8.17	.53	....	....	....	....	.9	....	....
				2	....	35.46	55.43	9.11	.59	....	....	....	....	....	....	....
				3	....	39.01	60.99	....	.65	....	....	....	....	....	....	....
Mancos, 6 miles north of; sec. 30, T. 37 N., R. 13 W.; Haller mine, Dakota formation (in entry, about 50 feet from mine mouth; 3-foot 10-inch bed, 1-foot 9-inch cut).	191	12504	B	1	4.01	28.65	47.56	19.78	.61	....	....	....	....	2.1	6,063	10,913
				2	....	29.85	49.54	20.61	.64	....	....	....	....	....	6,316	11,369
				3	....	37.60	62.40	....	.81	....	....	....	....	....	7,956	14,321
8 miles northeast of; sec. 21, T. 36 N., R. 12 W.; prospect, Dakota formation (in entry, about 20 feet from mouth, 1-foot 8½-inch bed, 1-foot 2-inch cut).	191	12444	B	1	2.60	23.25	41.28	32.87	.45	....	....	....	....	1.4	....	....
				2	....	23.87	42.38	33.75	.46	....	....	....	....	....	....	....
				3	....	36.02	63.98	....	.69	....	....	....	....	....	....	....
OURAY COUNTY.																
Ridgway, 10 miles northeast of; sec. 22, T. 46 N., R. 7 W.; Lou Creek mine, Lou Creek bed (in entry, 600 feet from entry mouth, 20-foot bed, 6-foot cut).	192	12651	B	1	14.15	37.80	40.98	7.07	.58	....	....	....	....	3.5	....	....
				2	....	44.03	47.73	8.24	.68	....	....	....	....	....	....	....
				3	....	47.98	52.02	....	.74	....	....	....	....	....	....	....
RIO BLANCO COUNTY.																
Meeker, 1½ miles northwest of; sec. 15, T. 1 N., R. 94 W.; Black Diamond mine, Lord bed (174 feet northeast of entry mouth, 6-foot 5-inch cut).	192	12776	B	1	10.82	37.25	43.95	7.98	.49	5.52	63.98	1.32	20.71	2.5	6,232	11,218
				2	....	41.77	49.28	8.95	.55	4.84	71.74	1.48	12.44	....	6,988	12,578
				3	....	45.88	54.12	....	.60	5.32	78.79	1.63	13.66	....	7,675	13,815
10 miles northeast of; sec. 10, T. 3 N., R. 93 W.; Cornrike mine (in mine, 30 feet northeast of mine mouth, 7-foot 6-inch bed and cut).	192	12695	B	1	13.88	42.04	41.87	2.21	.28	6.01	63.91	1.28	26.31	3.7	6,262	11,272
				2	....	48.82	48.61	2.57	.33	5.19	74.21	1.49	16.21	....	7,271	13,088
				3	....	50.11	49.89	....	.34	5.33	76.17	1.53	16.63	....	7,463	13,433
2½ miles west of; SW. ¼ sec. 28, T. 1 N., R. 94 W.; Fairfield mine (300 feet from mouth, 6-foot 4-inch cut).	193	12518	B	1	11.02	38.53	44.25	6.20	.85	5.80	64.20	1.32	21.63	4.0	6,314	11,365
				2	....	43.30	49.73	6.97	.96	5.15	72.15	1.48	13.29	....	7,096	12,773
				3	....	46.54	53.46	....	1.03	5.54	77.55	1.59	14.29	....	7,627	13,729
Three-fourths mile from; sec. 22, T. 1 N., R. 94 W.; Meeker mine, Old Pollard bed (about 500 feet northeast of mine mouth, 6-foot 6-inch cut).	193	12426	B	1	12.66	35.92	45.59	5.83	.44	5.69	64.92	1.25	21.87	6.6	6,330	11,394
				2	....	41.13	52.19	6.68	.50	4.90	74.33	1.43	12.16	....	7,248	13,046
				3	....	44.07	55.93	....	.54	5.25	79.65	1.53	13.03	....	7,767	13,980
3 miles west of; sec. 29, T. 1 N., R. 94 W.; Montgomery mine (in mine, 474 feet northeast of mine mouth, 7-foot 6-inch bed and cut).	193	12777	B	1	12.37	38.63	42.87	6.13	.71	5.38	62.61	1.33	23.84	2.0	5,995	10,791
				2	....	44.08	48.92	7.00	.81	4.58	71.45	1.52	14.64	....	6,842	12,315
				3	....	47.40	52.60	....	.87	4.92	76.83	1.63	15.75	....	7,357	13,242
2½ miles north of; sec. 3, T. 1 N., R. 94 W.; Sulphur Creek mine (in mine, 245 feet north of mine mouth, 5-foot 9-inch bed, 5-foot 7½-inch cut).	194	12773	B	1	10.41	36.56	44.73	8.30	.59	5.40	63.38	1.36	20.97	2.5	6,257	11,263
				2	....	40.81	49.93	9.26	.66	4.73	70.74	1.52	13.09	....	6,984	12,571
				3	....	44.97	55.03	....	.73	5.21	77.96	1.68	14.42	....	7,696	13,853
12 miles northeast of; sec. 30, T. 2 N., R. 92 W.; Wesson mine (in mine, 258 feet northwest of mine mouth, 22-foot 3½-inch bed, 18-foot 2-inch cut).	194	12705	B	1	14.40	38.42	41.17	6.01	.81	5.83	60.64	1.24	25.47	5.9	5,889	10,600
				2	....	44.88	48.10	7.02	.95	4.94	70.84	1.45	14.80	....	6,880	12,384
				3	....	48.27	51.73	....	1.02	5.31	76.19	1.56	15.92	....	7,399	13,318
IDAHO.																
BOISE COUNTY.																
Horseshoe Bend, 20 miles north of Boise; sec. 3, T. 6 N., R. 2 E.; Henry mine (Jensen level, 140 feet from slope mouth).	195	12703	B	1	10.11	38.24	36.07	15.58	.51	....	....	....	....	5.4	5,797	10,435
				2	....	42.54	40.13	17.33	.57	....	....	....	....	....	6,449	11,608
				3	....	51.46	48.54	....	.69	....	....	....	....	....	7,801	14,042
Same (60 feet from slope mouth)................	195	*12391	B	1	13.09	24.10	24.15	38.66	.36	....	....	....	....	3.8	3,504	6,307
				2	....	27.73	27.79	44.48	.41	....	....	....	....	....	4,032	7,258
				3	....	49.95	50.05	....	.74	....	....	....	....	....	7,262	13,072

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
IDAHO—Continued.																
CASSIA COUNTY.																
Oakley, 22 miles southwest of; sec. 26 (unsurveyed), T. 16 S., R. 20 E.; Worthington mine, Worthington bed (200 feet from slope mouth).	195	12643	B	1	34.46	26.41	21.08	18.05	0.63	......	......	......	......	26.1	3,229	5,812
				2	......	40.30	32.16	27.54	.96	......	......	......	......	......	4,927	8,869
				3	......	55.62	44.38	......	1.32	......	......	......	......	......	6,800	12,240
FREMONT COUNTY.																
Haden, 10 miles southwest of; sec. 25, T. 5 N., R. 43 E.; Brown Bear mine (end of north entry, 950 feet from portal; 5-foot 3-inch bed, 5-foot cut).	196	15115	B	1	11.45	37.24	47.01	4.30	.54	5.94	68.09	1.40	19.73	8.3	6,719	12.094
				2	......	42.06	53.08	4.86	.61	5.27	76.89	1.58	10.79	......	7,588	13,658
				3	......	44.21	55.79	......	.64	5.54	80.82	1.66	11.34	......	7,976	14,357
10 miles southwest of; sec. 6, T. 4 N., R. 44 E.; Horseshoe mine (in entry, 200 feet from portal, 11-foot 1-inch bed, 8-foot 6-inch cut).	196	15116	B	1	7.76	39.68	50.38	2.18	.38	......	......	......	......	4.3	7,155	12,879
				2	......	43.02	54.62	2.36	.41	......	......	......	......	......	7,757	13,963
				3	......	44.06	55.94	......	.42	......	......	......	......	......	7,945	14,301
ILLINOIS.																
MADISON COUNTY.																
Collinsville, 1 mile east of; Cantine No. 3 mine, No. 6 bed (rib, 10 feet from face of west entry 10, off north heading, 3,200 feet northwest of shaft, 7-foot 8¼-inch bed, 7-foot 5¼-inch cut).	197	10956	A	1	12.94	35.76	41.92	9.38	3.65	......	......	......	......	6.0	6,096	10,973
				2	......	41.07	48.16	10.77	4.19	......	......	......	......	......	7,002	12,604
				3	......	46.02	53.98	......	4.70	......	......	......	......	......	7,846	14,123
Same (face of first stub on east entry 8, off north heading, 4,200 feet north of shaft, 6-foot 10⅞-inch bed, 6-foot 6⅝-inch cut).	197	10957	A	1	12.31	37.57	41.09	9.03	3.58	......	......	......	......	5.4	6,175	11,115
				2	......	42.84	46.86	10.30	4.08	......	......	......	......	......	7,042	12,676
				3	......	47.76	52.24	......	4.55	......	......	......	......	......	7,850	14,130
Same (rib, 8 feet from face of east entry 7, on south heading, 2,500 feet southeast of shaft, 7-foot 7⅞-inch bed, 7-foot 5⅝-inch cut).	197	10958	A	1	12.81	36.31	41.35	9.53	3.28	......	......	......	......	5.4	6,124	11,023
				2	......	41.64	47.43	10.93	3.76	......	......	......	......	......	7,023	12,641
				3	......	46.75	53.25	......	4.22	......	......	......	......	......	7,885	14,193
Same (composite of samples 10956, 10957, 10958)...	197	10959	A	1	12.70	36.86	41.47	9.47	3.67	5.81	60.91	.99	19.15	5.6	6,105	10,989
				2	......	41.65	47.50	10.85	4.20	5.04	69.77	1.13	9.01	......	6,993	12,587
				3	......	46.72	53.28	......	4.71	5.65	78.26	1.27	10.11	......	7,844	14,119

MONTGOMERY COUNTY.																
Panama, ½ mile north of; Panama No. 1 mine, No. 6 bed (face of room 1, third south stub, off southeast entry 3, about 2,000 feet from shaft).	197	11352	A	1	13.23	34.45	42.37	9.95	3.57	....	....	....	....	7.4	....	....
				2	....	39.70	48.83	11.47	4.11	....	....	....	....	....	....	....
				3	....	44.85	55.15	....	4.64	....	....	....	....	....	....	....
Same (face of room 4, third north stub, off west entry 6, off north entry, 2,000 feet from shaft).	197	11353	A	1	12.99	32.86	40.33	13.82	4.06	....	....	....	....	7.2	....	....
				2	....	37.77	46.35	15.88	4.67	....	....	....	....	....	....	....
				3	....	44.90	55.10	....	5.55	....	....	....	....	....	....	....
Same (composite of samples 11352 and 11353).....	197	11354	A	1	13.31	33.62	41.34	11.73	3.75	5.19	59.07	.95	19.31	7.3	5,860	10,548
				2	....	38.78	47.69	13.53	4.33	4.28	68.14	1.10	8.62	....	6,760	12,168
				3	....	44.85	55.15	....	5.01	4.95	78.80	1.27	9.97	....	7,818	14,072
SALINE COUNTY.																
Harrisburg, 1 mile south of; O'Gara No. 9 mine, No. 5 bed (between rooms 18 and 19, face of west entry 3, off main south entry, 3,000 feet from mine mouth, 7-foot 5-inch bed, 7-foot 3-inch cut).	198	12794	A	1	7.12	34.55	50.68	7.65	2.23	....	....	....	....	4.6	6,934	12,481
				2	....	37.20	54.56	8.24	2.40	....	....	....	....	....	7,466	13,439
				3	....	40.54	59.46	....	2.62	....	....	....	....	....	8,136	14,645
Same (face of second crosscut in room 11, off west entry 3, off main south entry, 6-foot 7½-inch bed, 6-foot 7-inch cut).	198	12795	A	1	7.02	34.58	51.46	6.94	2.07	....	....	....	....	4.1	7,022	12,640
				2	....	37.19	55.35	7.46	2.23	....	....	....	....	....	7,552	13,594
				3	....	40.19	59.81	....	2.41	....	....	....	....	....	8,161	14,690
VERMILION COUNTY.																
Westville, 2 miles south of; sec. 19, T. 18 N., R. 11 W.; Little Vermilion mine, No. 6 bed ("Run-of-mine" shipped to Pittsburgh station for testing purposes).	199	14682	C	1	16.16	34.09	39.19	10.56	1.74	5.71	58.38	1.25	22.36	3.4	5,796	10,433
				2	....	40.66	46.75	12.59	2.08	4.66	69.63	1.49	9.55	....	6,913	12,443
				3	....	46.52	53.48	....	2.38	5.33	79.66	1.70	10.93	....	7,908	14,234
INDIANA.																
SULLIVAN COUNTY.																
Dugger, 1½ miles southwest of; Vandalia No. 10 mine, No. 4 bed (face of room 2, east entry 6, off north entry 2, off main east entry, 5-foot 5½-inch bed, 5-foot 2¾-inch cut).	199	10960	A	1	13.05	34.30	47.61	5.04	.82	6.07	67.34	1.44	19.29	6.3	6,679	12,022
				2	....	39.45	54.75	5.80	.94	5.31	77.45	1.66	8.84	....	7,682	13,828
				3	....	41.88	58.12	....	1.00	5.64	82.22	1.76	9.38	....	8,155	14,679
Same (face of room 12, off east entry 6, off north entry 2, off main east entry, 5-foot 3½-inch bed, 5-foot ½-inch cut).	199	10961	A	1	13.48	32.51	48.38	5.63	1.09	5.94	66.01	1.49	19.84	6.6	6,549	11,788
				2	....	37.58	55.91	6.51	1.26	5.13	76.29	1.72	9.09	....	7,569	13,624
				3	....	40.20	59.80	....	1.35	5.49	81.60	1.84	9.72	....	8,096	14,573
WARRICK COUNTY.																
Elberfeld, 1 mile south of; Elberfeld mine, No. 5 bed (in new manway, off main west entry, 200 feet north of shaft bottom, 4-foot ½-inch bed and cut).	200	16578	A	1	9.69	38.59	41.04	10.68	4.79	5.39	62.36	1.28	15.50	5.7	6,340	11,412
				2	....	42.73	45.44	11.83	5.30	4.77	69.05	1.42	7.63	....	7,020	12,636
				3	....	48.46	51.54	....	6.01	5.41	78.32	1.61	8.65	....	7,962	14,332
KANSAS.																
CRAWFORD COUNTY.																
Franklin, ½ mile west of; Western No. 16 mine, Weir-Pittsburg bed (2,000 feet southwest of shaft, 35-inch bed and cut).	200	17032	A	1	6.69	33.55	47.94	11.82	5.25	....	....	....	....	4.3	6,728	12,110
				2	....	35.96	51.37	12.67	5.63	....	....	....	....	....	7,210	12,978
				3	....	41.18	58.82	....	6.45	....	....	....	....	....	8,256	14,861

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
KANSAS—Continued.																
CRAWFORD COUNTY—continued.																
Fuller, southwest corner of; sec. 12, T. 29 N., R. 25 E.; Sheridan No. 2 mine (face, 2,500 feet southwest of shaft, 3-foot bed, 2-foot 11-inch cut).	201	11181	A	1	5.04	33.05	52.38	9.53	5.10	......	......	......	......	3.5	......	......
				2	......	34.80	55.16	10.04	5.37	......	......	......	......	......	......	......
				3	......	38.68	61.32	......	5.97	......	......	......	......	......	......	......
Same (face, 2,500 feet northeast of shaft, 3-foot bed, 2-foot 11-inch cut).	201	11182	A	1	4.72	33.35	52.85	9.08	5.19	......	......	......	......	3.3	......	......
				2	......	35.00	55.47	9.53	5.45	......	......	......	......	......	......	......
				3	......	38.69	61.31	......	6.02	......	......	......	......	......	......	......
Same (face, 2,500 feet northwest of shaft, 3-foot bed, 2-foot 11-inch cut).	201	11183	A	1	5.00	33.97	52.37	8.66	4.64	......	......	......	......	3.5	......	......
				2	......	35.76	55.12	9.12	4.88	......	......	......	......	......	......	......
				3	......	39.35	60.65	......	5.37	......	......	......	......	......	......	......
Same (composite of samples 11181, 11182, and 11183).	201	11184	A	1	4.85	33.53	52.52	9.10	4.95	5.08	71.20	1.24	8.43	3.4	7,190	12,942
				2	......	35.24	55.20	9.56	5.20	4.77	74.83	1.30	4.34	......	7,557	13,603
				3	......	38.96	61.04	......	5.75	5.27	82.74	1.44	4.80	......	8,356	15,041
Yale, ½ mile northwest of; sec. 26, T. 30 N., R. 25 E.; Western No. 13 mine (face, 2,600 feet north of shaft, 3-foot bed, 2-foot 11½-inch cut).	202	11209	A	1	5.25	31.26	49.22	14.27	5.15	......	......	......	......	3.7	......	......
				2	......	32.99	51.95	15.06	5.44	......	......	......	......	......	......	......
				3	......	38.84	61.16	......	6.40	......	......	......	......	......	......	......
Same (face, 2,800 feet southeast of shaft, 3-foot bed, 2-foot 11½-inch cut).	202	11210	A	1	5.64	32.87	50.98	10.51	4.25	......	......	......	......	4.1	......	......
				2	......	34.84	54.02	11.14	4.50	......	......	......	......	......	......	......
				3	......	39.21	60.79	......	5.06	......	......	......	......	......	......	......
Same (face, 2,800 feet southwest of shaft, 3-foot bed, 2-foot 11½-inch cut).	202	11211	A	1	5.83	31.37	51.29	11.51	4.99	......	......	......	......	4.3	......	......
				2	......	33.31	54.47	12.22	5.30	......	......	......	......	......	......	......
				3	......	37.95	62.05	......	6.04	......	......	......	......	......	......	......
Same (composite of samples 11209, 11210, and 11211).	202	11212	A	1	5.55	32.54	49.74	12.17	4.88	5.08	67.66	1.14	9.07	4.0	6,847	12,325
				2	......	34.45	52.66	12.89	5.17	4.72	71.64	1.21	4.37	......	7,250	13,050
				3	......	39.55	60.45	......	5.94	5.42	82.24	1.39	5.01	......	8,323	14,981
LEAVENWORTH COUNTY.																
Lansing; sec. 19, T. 9 S., R. 23 E.; Penitentiary mine, Bevier bed (in mine, 1,100 feet south and 1,400 feet east of shaft, 1-foot 10-inch bed and cut).	202	12841	B	1	11.04	34.53	41.01	13.42	5.07	......	......	......	......	8.3	......	......
				2	......	38.82	46.09	15.09	5.70	......	......	......	......	......	......	......
				3	......	45.72	54.28	......	6.71	......	......	......	......	......	......	......

Same (4,400 feet east and 500 feet north of shaft, 1-foot 10-inch bed and cut).	202	12842	B	1	10.76	36.63	41.32	11.29	3.34	......	......	......	......	8.0	......	......
				2	......	41.05	46.30	12.65	3.74	......	......	......	......	......	......	......
				3	......	46.99	53.01	......	4.28	......	......	......	......	......	......	......
Same (2,300 feet east and 2,600 feet north of shaft, 1-foot 10-inch bed and cut).	202	12843	B	1	11.99	35.69	39.56	12.76	3.50	......	......	......	......	9.6	......	......
				2	......	40.55	44.95	14.50	3.98	......	......	......	......	......	......	......
				3	......	47.43	52.57	......	4.66	......	......	......	......	......	......	......
Same (composite of samples 12841, 12842, and 12843).	202	12844	B	1	11.10	35.51	40.69	12.70	3.99	5.30	60.72	1.13	16.16	8.6	6,147	11,065
				2	......	39.95	45.76	14.29	4.49	4.58	68.30	1.27	7.07	......	6,915	12,447
				3	......	46.61	53.39	......	5.24	5.34	79.69	1.48	8.25	......	8,068	14,522
Leavenworth, south edge of; sec. 36, T. 8 S., R. 22 E.; No. 1 mine, Bevier bed (3,200 feet north and 560 feet west of shaft, 1-foot 9-inch bed).	202	12849	B	1	12.60	34.17	40.37	12.86	4.50	......	......	......	......	9.7	......	......
				2	......	39.10	46.19	14.71	5.15	......	......	......	......	......	......	......
				3	......	45.84	54.16	......	6.04	......	......	......	......	......	......	......
Same (160 feet west and 3,100 feet south of shaft, 1-foot 9-inch bed).	202	12850	B	1	11.83	34.47	39.13	14.57	4.40	......	......	......	......	8.9	......	......
				2	......	39.10	44.37	16.53	4.99	......	......	......	......	......	......	......
				3	......	46.84	53.16	......	5.98	......	......	......	......	......	......	......
Same (110 feet east and 3,310 feet north of shaft, 1-foot 10-inch bed).	202	12851	B	1	11.43	36.05	38.79	13.73	4.17	......	......	......	......	8.8	......	......
				2	......	40.70	43.80	15.50	4.71	......	......	......	......	......	......	......
				3	......	48.17	51.83	......	5.57	......	......	......	......	......	......	......
Same (composite of samples 12849, 12850, and 12851).	202	12852	B	1	11.96	35.21	39.13	13.70	4.41	5.31	58.94	1.09	16.55	9.1	5,956	10,721
				2	......	39.99	44.45	15.56	5.01	4.52	66.94	1.24	6.73	......	6,765	12,177
				3	......	47.36	52.64	......	5.93	5.35	79.28	1.47	7.97	......	8,012	14,421
North edge of; sec. 25, T. 8 S., R. 22 E.; No. 3 mine; Leavenworth bed (in mine, 3,400 feet southeast of shaft, 1-foot 10-inch bed and cut).	203	12845	B	1	10.61	37.07	36.16	16.16	5.19	......	......	......	......	7.6	......	......
				2	......	41.47	40.45	18.08	5.81	......	......	......	......	......	......	......
				3	......	50.62	49.38	......	7.09	......	......	......	......	......	......	......
Same (in mine, 3,800 feet southeast of shaft, 1-foot 10-inch bed and cut).	203	12846	B	1	12.64	34.77	33.22	19.37	4.66	......	......	......	......	9.6	......	......
				2	......	39.80	38.03	22.17	5.33	......	......	......	......	......	......	......
				3	......	51.14	48.86	......	6.85	......	......	......	......	......	......	......
Same (in mine, 3,350 feet southeast of shaft, 1-foot 10-inch bed and cut).	203	12847	B	1	13.03	34.15	40.17	12.65	4.56	......	......	......	......	9.8	......	......
				2	......	39.27	46.19	14.54	5.24	......	......	......	......	......	......	......
				3	......	45.95	54.05	......	6.13	......	......	......	......	......	......	......
Same (composite of samples 12845, 12846, and 12847).	203	12848	B	1	12.06	35.42	36.45	16.07	4.77	5.16	56.54	1.07	16.39	9.0	5,675	10,215
				2	......	40.28	41.45	18.27	5.42	4.34	64.29	1.22	6.46	......	6,453	11,615
				3	......	49.28	50.72	......	6.63	5.31	78.66	1.49	7.91	......	7,895	14,211

KENTUCKY.

LETCHER COUNTY.

Flatgap, Va., 1½ miles south of; local mine, Lower Standiford bed (in entry, about 15 feet from mine mouth, 3-foot 2-inch bed).	203	15172	B	1	4.07	35.21	55.69	5.03	1.74	......	......	......	......	2.8	7,725	13,905
				2	......	36.70	58.06	5.24	1.81	......	......	......	......	......	8,053	14,495
				3	......	38.73	61.27	......	1.91	......	......	......	......	......	8,498	15,296
3 miles southwest of; J. H. Mullin mine, Lower Bolling bed (in main entry, 40 feet from mouth, 4-foot 7-inch bed, 4-foot 1-inch cut).	203	15173	B	1	3.97	31.77	53.07	11.19	.97	......	......	......	......	2.6	7,140	12,852
				2	......	33.08	55.27	11.65	1.01	......	......	......	......	......	7,435	13,383
				3	......	37.44	62.56	......	1.14	......	......	......	......	......	8,416	15,149
Jenkins; Consolidation No. 204 mine, Upper Elkhorn bed (face of main heading, 1,700 feet from mine mouth, 7-foot 6½-inch bed, 7-foot 3½-inch cut).	204	14904	B	1	3.91	33.84	54.76	7.49	.51	......	......	......	......	1.9	7,459	13,426
				2	......	35.22	56.99	7.79	.53	......	......	......	......	......	7,763	13,973
				3	......	38.20	61.80	......	.57	......	......	......	......	......	8,419	15,154
Same (face of east heading, 1,700 feet from mine mouth, 8-foot 5-inch bed, 8-foot cut).	204	14905	B	1	3.64	35.81	57.96	2.59	.51	5.39	80.06	1.52	9.93	1.7	7,912	14,242
				2	......	37.16	60.15	2.69	.53	5.18	83.09	1.58	6.93	......	8,211	14,790
				3	......	38.19	61.81	......	.54	5.32	85.38	1.62	7.14	......	8,438	15,188

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
KENTUCKY—Continued.																
LETCHER COUNTY—continued.																
Jenkins, 5 miles northeast of; local mine, Lower Elkhorn bed (face of entry, 50 feet from mine mouth, 3-foot ½-inch bed, 2-foot ½-inch cut).	204	14970	B	1	4.83	33.71	57.73	3.73	0.82					3.1	7,690	13,842
				2		35.42	60.66	3.92	.86						8,081	14,546
				3		36.87	63.13		.90						8,411	15,140
Same (position and bed same as sample 14970, 1-foot cut).	204	14971	B	1	5.15	28.79	48.17	17.89	.68					3.6	6,363	11,453
				2		30.35	50.79	18.86	.72						6,709	12,076
				3		37.40	62.60		.89						8,268	14,882
MUHLENBERG COUNTY.																
Central City, ½ mile south of; Central mine, No. 9 bed (face of main south entry, 5-foot 6-inch bed and cut).	205	13257	A	1	8.11	38.24	47.01	6.64	2.40					5.0		
				2		41.62	51.15	7.23	2.61							
				3		44.86	55.14		2.81							
Same (face of room 20, off main south blind entry, 5-foot 6-inch bed and cut).	205	13258	A	1	10.33	35.85	44.65	9.17	2.95					7.4		
				2		39.98	49.79	10.23	3.29							
				3		44.54	55.46		3.67							
Same (room 3, south entry 6, off main west entry, 5-foot 6-inch bed and cut).	205	13259	A	1	7.98	37.78	47.29	6.95	2.50					5.1		
				2		41.06	51.39	7.55	2.72							
				3		44.41	55.59		2.94							
Same (composite of samples 13257, 13258, and and 13259).	205	13260	A	1	8.73	37.76	45.93	7.58	2.65	5.52	67.65	1.42	15.18	5.8	6,782	12,208
				2		41.37	50.33	8.30	2.90	4.98	74.12	1.56	8.14		7,430	13,374
				3		45.11	54.89		3.16	5.43	80.83	1.70	8.88		8,102	14,584
Cleaton; Bevier mine, No. 9 bed (face of room 15, off north air course 8, 4,100 feet from shaft, 5-foot 3-inch bed and cut).	205	15339	A	1	7.98	37.79	45.73	8.50	2.75					5.1	6,689	12,040
				2		41.07	49.69	9.24	2.99						7,269	13,084
				3		45.25	54.75		3.29						8,009	14,416
Same (face of room 21, off north entry 7, 3,800 feet from shaft, 5-foot 6-inch bed and cut).	205	15340	A	1	8.33	37.97	44.53	9.17	3.39					5.4	6,579	11,842
				2		41.42	48.58	10.00	3.70						7,177	12,919
				3		46.02	53.98		4.11						7,974	14,353
Same (face of south entry 8, 4,500 feet from shaft, 5-foot 3-inch bed and cut).	205	15341	A	1	8.21	36.04	45.13	10.62	3.77					5.5	6,467	11,641
				2		39.26	49.17	11.57	4.11						7,045	12,681
				3		44.40	55.60		4.65						7,966	14,339

Graham, 2 miles north of; Skibo mine, No. 9 bed (face of west entry 2, off north entry 1, 4,000 feet north of pit mouth, 4-foot 10-inch bed and cut).	206	16674	A	1	8.81	36.15	46.43	8.61	3.55	5.31	65.34	1.47	15.72	5.4	6,612	11,902
				2	......	39.64	50.92	9.44	3.89	4.76	71.65	1.61	8.65	......	7,251	13,052
				3	......	43.77	56.23	......	4.30	5.26	79.12	1.78	9.54	......	8,007	14,413
Same (face of room 2, off east entry 7, 3,600 feet northeast of pit mouth, 4-foot 9-inch bed).	206	16675	A	1	8.21	36.39	45.34	10.06	3.33	5.36	65.22	1.55	14.48	5.1	6,543	11,777
				2	......	39.64	49.40	10.96	3.63	4.85	71.05	1.69	7.82	......	7,128	12,830
				3	......	44.52	55.48	......	4.08	5.45	79.80	1.90	8.77	......	8,005	14,409
OHIO COUNTY.																
McHenry, near; McHenry mine, No. 9 bed (face of room 4, off south entry 3, 9,000 feet from mine mouth, 4-foot 6-inch bed).	207	15102	A	1	9.89	35.94	43.36	10.81	3.64	5.37	62.27	1.33	16.58	7.0	6,329	11,392
				2	......	39.89	48.11	12.00	4.04	4.74	69.11	1.48	8.63	......	7,024	12,643
				3	......	45.33	54.67	......	4.59	5.39	78.54	1.68	9.80	......	7,982	14,368
Same (room 1, off north entry 4, 8,500 feet from mine mouth, 4-foot 6-inch bed).	207	15103	A	1	9.97	36.54	44.93	8.56	2.62	5.49	65.01	1.40	16.92	7.0	6,540	11,772
				2	......	40.58	49.91	9.51	2.91	4.86	72.21	1.55	8.96	......	7,264	13,075
				3	......	44.84	55.16	......	3.22	5.37	79.80	1.71	9.90	......	8,027	14,449
PIKE COUNTY.																
Hellier, 1 mile southeast of; Edgewater mine, Upper Elkhorn bed (sample taken at mine from three loaded railroad cars).	207	12004	B	1	1.94	35.48	59.88	2.70	.57	5.38	81.37	1.42	8.56	.5	8,163	14,693
				2	......	36.18	61.07	2.75	.58	5.26	82.98	1.45	6.98	......	8,325	14,985
				3	......	37.20	62.80	......	.60	5.41	85.33	1.49	7.17	......	8,561	15,410
MISSOURI.																
ADAIR COUNTY.																
Kirksville, ½ mile west of; sec. 8, T. 63 N., R. 15 W.; Star mine, Tebo (?) bed (in mine, 30 feet south of shaft, 1-foot 5½-inch bed).	207	14799	B	1	15.98	38.15	37.18	8.69	4.12	5.90	59.09	.94	21.26	11.0	5,999	10,798
				2	......	45.41	44.25	10.34	4.90	4.90	70.33	1.12	8.41	......	7,140	12,852
				3	......	50.65	49.35	......	5.46	5.46	78.44	1.25	9.39	......	7,963	14,333
Same (in mine, 12 feet northeast of shaft, 1-foot 8-inch bed).	207	14800	B	1	13.94	37.80	38.11	10.15	5.95	5.71	58.76	1.01	18.42	9.4	6,028	10,850
				2	......	43.92	44.29	11.79	6.91	4.83	68.28	1.17	7.02	......	7,005	12,609
				3	......	49.79	50.21	......	7.83	5.48	77.41	1.33	7.95	......	7,942	14,296
AUDRAIN COUNTY.																
Martinsburg, ½ mile west of; sec. 14, T. 50 N., R. 7 W.; No. 1 mine; Mulky bed (long-wall face, 600 feet south of shaft), 2-foot 6-inch bed and cut.)	208	11483	A	1	11.73	37.18	38.00	13.09	5.07	......	......	......	......	10.5	......	......
				2	......	42.12	43.05	14.83	5.74	......	......	......	......	......	......	......
				3	......	49.45	50.55	......	6.74	......	......	......	......	......	......	......
Same (long-wall face, 600 feet southwest of shaft, 2-foot 6-inch cut).	208	11484	A	1	8.71	40.73	39.92	10.64	5.27	......	......	......	......	7.4	......	......
				2	......	44.62	43.72	11.66	5.77	......	......	......	......	......	......	......
				3	......	50.51	49.49	......	6.53	......	......	......	......	......	......	......
Same (face, 800 feet southwest of shaft, 2-foot 6-inch cut).	208	11485	A	1	9.75	39.12	39.37	11.76	4.88	......	......	......	......	8.5	......	......
				2	......	43.34	43.63	13.03	5.41	......	......	......	......	......	......	......
				3	......	49.83	50.17	......	6.22	......	......	......	......	......	......	......
Same (composite of samples 11483, 11484, and 11485).	208	11478	A	1	10.06	38.65	39.67	11.62	5.16	5.43	60.84	.83	16.12	8.8	6,192	11,146
				2	......	42.97	44.11	12.92	5.74	4.79	67.65	.92	7.98	......	6,885	12,393
				3	......	49.35	50.65	......	6.59	5.50	77.69	1.06	9.16	......	7,907	14,233

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
MISSOURI—Continued.																
BARTON COUNTY.																
Mindenmines, 2 miles northwest of; sec. 30, T. 32 N., R. 33 W.; No. 17 mine; Wier-Pittsburg Lower bed (face, 1,000 feet northeast of shaft, 2-foot 10-inch bed and cut).	208	11185	A	1	6.26	30.13	51.51	12.10	5.45	......	......	......	......	4.7	......	......
				2	......	32.14	54.95	12.91	5.81	......	......	......	......	......	......	......
				3	......	36.90	63.10	......	6.67	......	......	......	......	......	......	......
Same (face, 1,000 feet southeast of shaft, 2-foot 10-inch bed and cut).	208	11186	A	1	6.07	32.66	52.22	9.05	4.14	......	......	......	......	4.6	......	......
				2	......	34.77	55.60	9.63	4.41	......	......	......	......	......	......	......
				3	......	38.48	61.52	......	4.88	......	......	......	......	......	......	......
Same (face, 1,500 feet northwest of shaft, 2-foot 10-inch bed and cut).	208	11187	A	1	5.13	31.97	49.74	13.16	5.01	......	......	......	......	3.7	......	......
				2	......	33.70	52.43	13.87	5.28	......	......	......	......	......	......	......
				3	......	39.13	60.87	......	6.13	......	......	......	......	......	......	......
Same (composite of samples 11185, 11186, and 11187).	208	11188	A	1	5.87	30.98	51.67	11.48	5.00	5.07	67.64	1.09	9.72	4.3	6,855	12.339
				2	......	32.91	54.89	12.20	5.31	4.70	71.85	1.16	4.78	......	7,282	13.108
				3	......	37.48	62.52	......	6.05	5.35	81.84	1.32	5.44	......	8,294	14.929
1 mile northeast of; sec. 28, T. 32 N., R. 33 W.; Pullen No. 1 mine, Wier-Pittsburg Lower bed (south face, 600 feet from shaft, 2-foot 6-inch bed and cut).	209	11177	A	1	5.39	32.26	53.89	8.46	3.55	......	......	......	......	4.2	......	......
				2	......	34.10	56.96	8.94	3.75	......	......	......	......	......	......	......
				3	......	37.45	62.55	......	4.12	......	......	......	......	......	......	......
Same (west face, 400 feet from shaft, 2-foot 7-inch bed and cut).	209	11178	A	1	5.42	32.65	52.13	9.80	3.62	......	......	......	......	4.2	......	......
				2	......	34.52	55.12	10.36	3.83	......	......	......	......	......	......	......
				3	......	38.51	61.49	......	4.27	......	......	......	......	......	......	......
Same (northeast face, 600 feet from shaft, 2-foot 6-inch bed and cut).	209	11179	A	1	5.54	32.34	53.40	8.72	3.39	......	......	......	......	4.2	......	......
				2	......	34.24	56.53	9.23	3.59	......	......	......	......	......	......	......
				3	......	37.72	62.28	......	3.96	......	......	......	......	......	......	......
Same (composite of samples 11177, 11178, and 11179).	209	11180	A	1	5.43	32.06	53.56	8.95	3.62	5.38	71.58	1.37	9.10	4.2	7,218	12,992
				2	......	33.90	56.64	9.46	3.83	5.05	75.69	1.45	4.52	......	7,632	13,738
				3	......	37.44	62.56	......	4.23	5.58	83.60	1.60	4.99	......	8,430	15,174
BATES COUNTY.																
Amsterdam, southern part of; sec. 20, T. 41 N., R. 33 W.; No. 1 mine, Mulberry bed (face, 400 feet east of shaft, 2-foot 6-inch bed and cut).	209	11306	A	1	9.82	31.23	42.57	16.38	4.09	......	......	......	......	8.2	......	......
				2	......	34.63	47.21	18.16	4.54	......	......	......	......	......	......	......
				3	......	42.31	57.69	......	5.55	......	......	......	......	......	......	......

Same (face, 400 feet south of shaft, 2-foot 6-inch bed and cut).	209	11307	A	1	8.74	33.13	44.64	13.49	3.64	......	......	......	......	7.2	......	......
				2	......	36.30	48.92	14.78	3.99	......	......	......	......	......	......	......
				3	......	42.59	57.41	......	4.68	......	......	......	......	......	......	......
Same (face, 400 feet southeast of shaft, 2-foot 6-inch bed and cut).	209	11308	A	1	9.13	32.45	44.21	14.21	3.98	......	......	......	......	7.3	......	......
				2	......	35.71	48.65	15.64	4.38	......	......	......	......	......	......	......
				3	......	42.33	57.67	......	5.19	......	......	......	......	......	......	......
Same (composite of samples 11306, 11307, and 11308).	209	11309	A	1	9.19	32.11	44.17	14.53	3.96	5.11	61.48	1.01	13.91	7.6	6,226	11,207
				2	......	35.36	48.64	16.00	4.36	4.50	67.70	1.11	6.83	......	6,856	12,341
				3	......	42.10	57.90	......	5.19	5.36	80.60	1.32	7.53	......	8,162	14,692
Hume, 4 miles east of, 10 miles west of Rich Hill; sec. 4, T. 38 N., R. 33 W.; No. 1 mine, Mulberry bed (face, 900 feet northwest of shaft, 2-foot 6-inch bed and cut).	210	11226	A	1	12.43	30.02	45.84	11.71	1.98	......	......	......	......	9.8	......	......
				2	......	34.28	52.35	13.37	2.26	......	......	......	......	......	......	......
				3	......	39.57	60.43.	......	2.61	......	......	......	......	......	......	......
Same (face 1,100 feet west of shaft)...............	210	11227	A	1	12.84	30.95	46.86	9.35	1.74	......	......	......	......	10.4	......	......
				2	......	35.51	53.76	10.73	2.00	......	......	......	......	......	......	......
				3	......	39.78	60.22	......	2.24	......	......	......	......	......	......	......
Same (face, 700 feet southwest of shaft)...........	210	11228	A	1	11.39	30.95	46.55	11.11	2.52	......	......	......	......	9.5	......	......
				2	......	34.93	52.53	12.54	2.84	......	......	......	......	......	......	......
				3	......	39.94	60.06	......	3.25	......	......	......	......	......	......	......
Same (composite of samples 11226, 11227, and 11228).	210	11229	A	1	12.27	30.36	46.68	10.69	2.08	5.50	63.26	1.20	17.27	9.9	6,275	11,295
				2	......	34.61	53.20	12.19	2.37	4.72	72.11	1.37	7.24	......	7,153	12,875
				3	......	39.41	60.59	......	2.70	5.38	82.12	1.56	8.24	......	8,146	14,663
New Home; sec. 30, T. 39 N., R. 32 W.; New Home No. 1 mine; Lower Rich Hill bed (face, 200 feet northeast of shaft, 5-foot bed and cut).	210	11222	A	1	9.32	33.97	43.90	12.81	3.84	......	......	......	......	7.7	......	......
				2	......	37.46	48.41	14.13	4.23	......	......	......	......	......	......	......
				3	......	43.63	56.37	......	4.93	......	......	......	......	......	......	......
Same (face, 150 feet southeast of shaft, 5-foot bed and cut).	210	11223	A	1	10.05	32.53	43.22	14.20	4.47	......	......	......	......	8.2	......	......
				2	......	36.16	48.05	15.79	4.97	......	......	......	......	......	......	......
				3	......	42.94	57.06	......	5.90	......	......	......	......	......	......	......
Same (face, 200 feet south of shaft, 5-foot bed and cut).	210	11224	A	1	10.63	31.25	41.85	16.27	6.77	......	......	......	......	8.9	......	......
				2	......	34.97	46.83	18.20	7.57	......	......	......	......	......	......	......
				3	......	42.75	57.25	......	9.25	......	......	......	......	......	......	......
Same (composite of samples 11222, 11223, and 11224).	210	11225	A	1	10.01	33.09	42.45	14.45	5.06	5.27	60.29	1.01	13.93	8.3	6,192	11,146
				2	......	36.77	47.17	16.06	5.61	4.62	66.99	1.12	5.60	......	6,881	12,386
				3	......	43.80	56.20	......	6.68	5.50	79.81	1.33	6.68	......	8,197	14,755
Rich Hill, 2 miles northwest of; sec. 36, T. 39 N., R. 32 W.; Fleming Pit No. 1, Lower Rich Hill bed (face on north side of pit, 5-foot bed, 4-foot 10-inch cut).	210	11197	A	1	7.79	35.37	45.61	11.23	4.13	......	......	......	......	6.1	......	......
				2	......	38.36	49.46	12.18	4.48	......	......	......	......	......	......	......
				3	......	43.68	56.32	......	5.10	......	......	......	......	......	......	......
Same (face on east side of pit, 5-foot bed, 4-foot 10-inch cut).	210	11198	A	1	7.97	35.40	43.60	13.03	5.07	......	......	......	......	6.4	......	......
				2	......	38.47	47.37	14.16	5.51	......	......	......	......	......	......	......
				3	......	44.82	55.18	......	6.42	......	......	......	......	......	......	......
Same (face on south side of pit, 5-foot bed, 4-foot 10-inch cut).	210	11199	A	1	7.55	35.05	43.75	13.65	5.91	......	......	......	......	6.1	......	......
				2	......	37.91	47.32	14.77	6.39	......	......	......	......	......	......	......
				3	......	44.48	55.52	......	7.50	......	......	......	......	......	......	......
Same (composite of samples 11197, 11198, and 11199).	210	11200	A	1	7.79	35.51	44.00	12.70	5.16	5.40	63.17	1.00	12.57	6.2	6,524	11,742
				2	......	38.51	47.72	13.77	5.60	4.91	68.51	1.08	6.13	......	7,075	12,735
				3	......	44.66	55.34	......	6.49	5.69	79.45	1.25	7.12	......	8,205	14,769

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
MISSOURI—Continued.																
BATES COUNTY—continued.																
Rich Hill, 2 miles south of; sec. 17, T. 38 N., R. 31 W.; Ritchie Pit No. 1, Lower Rich Hill bed (face on north side pit, 4-foot 11-inch bed, 4-foot 8-inch cut).	211	11193	A	1	9.72	33.99	42.49	13.80	5.20	......	......	......	......	8.4	......	......
				2	......	37.65	47.06	15.29	5.76	......	......	......	......	......	......	......
				3	......	44.45	55.55	......	6.80	......	......	......	......	......	......	......
Same (face on south side pit, 4-foot 11-inch bed, 4-foot 8-inch cut).	211	11194	A	1	8.54	34.88	40.51	16.07	4.32	......	......	......	......	7.8	......	......
				2	......	38.14	44.29	17.57	4.72	......	......	......	......	......	......	......
				3	......	46.27	53.73	......	5.73	......	......	......	......	......	......	......
Same (composite of samples 11193 and 11194)	211	11195	A	1	9.20	34.34	41.28	15.18	4.88	5.33	60.22	1.11	13.28	7.9	6,195	11,151
				2	......	37.82	45.46	16.72	5.37	4.75	66.32	1.22	5.62	......	6,823	12,281
				3	......	45.41	54.59	......	6.45	5.70	79.64	1.46	6.75	......	8,193	14,747
2 miles south of; sec. 17, T. 38 N., R. 31 W.; Ritchie Pit No. 2, Lower Rich Hill bed (face on south side pit, 2-foot cut).	211	[a]11196	A	1	35.97	22.54	26.85	14.64	.81	5.93	33.23	.70	44.69	30.1	2,920	5,256
				2	......	35.20	41.94	22.86	1.27	3.01	51.90	1.09	19.87	......	4,560	8,208
				3	......	45.63	54.37	......	1.65	3.90	67.28	1.41	25.76	......	5,911	10,640
BOONE COUNTY.																
Columbia, 3½ miles northeast of; sec. 4, T. 48 N., R. 12 W.; No. 1 mine (face, 110 feet north of shaft, 3-foot bed and cut).	211	11476	A	1	11.16	34.95	43.32	10.57	3.71	5.23	62.92	.95	16.62	9.8	6,293	11,327
				2	......	39.34	48.76	11.90	4.18	4.49	70.82	1.07	7.54	......	7,083	12,749
				3	......	44.65	55.35	......	4.74	5.10	80.39	1.21	8.56	......	8,040	14,472
Same (face, 50 feet west of shaft, 3-foot bed and cut)	211	11477	A	1	9.41	34.91	45.34	10.34	4.08	5.42	63.47	.97	15.72	8.0	6,466	11,639
				2	......	38.54	50.05	11.41	4.50	4.82	70.06	1.07	8.14	......	7,138	12,848
				3	......	43.50	56.50	......	5.08	5.44	79.08	1.21	9.19	......	8,057	14,503
3½ miles northeast of; sec. 4, T. 48 N., R. 12 W.; No. 2 mine (face, 50 feet west of shaft, 3-foot bed and cut).	211	11475	A	1	10.48	35.31	44.16	10.05	4.49	5.49	62.81	.95	16.21	9.0	6,365	11,457
				2	......	39.44	49.33	11.23	5.02	4.84	70.17	1.06	7.68	......	7,110	12,798
				3	......	44.43	55.57	......	5.66	5.45	79.05	1.19	8.65	......	8,009	14,416
4½ miles north of; sec. 11, T. 49 N., R. 12 W.; Prather No. 1 mine (face, 600 feet southwest of shaft, 3-foot 6-inch cut).	212	11479	A	1	12.29	31.43	40.33	15.95	6.36	......	......	......	......	11.0	......	......
				2	......	35.83	45.99	18.18	7.25	......	......	......	......	......	......	......
				3	......	43.79	56.21	......	8.86	......	......	......	......	......	......	......
Same (face, 600 feet west of shaft, 3-foot 6-inch cut)	212	11480	A	1	9.82	34.81	42.85	12.52	4.72	......	......	......	......	8.4	......	......
				2	......	38.60	47.52	13.88	5.23	......	......	......	......	......	......	......
				3	......	44.82	55.18	......	6.07	......	......	......	......	......	......	......

Same (face, 800 feet east of shaft, 3-foot 6-inch cut)	211	11481	A	1	9.82	35.13	43.94	11.11	3.75	....	....	....	....	8.4	....	....
				2	....	38.96	48.72	12.32	4.16	....	....	....	....	....	....	....
				3	....	44.43	55.57	....	4.74	....	....	....	....	....	....	....
Same (composite of samples 11479, 11480, and 11481).	211	11482	A	1	10.76	33.63	42.40	13.21	5.06	5.27	60.15	.92	15.39	9.3	6,023	10,841
				2	....	37.69	47.51	14.80	5.67	4.56	67.40	1.03	6.54	....	6,749	12,148
				3	....	44.24	55.76	....	6.65	5.35	79.11	1.21	7.68	....	7,921	14,258
CALLAWAY COUNTY.																
Fulton, 2 miles southwest of; sec. 18, T. 47 N., R. 9 W.; No. 1 mine, Bevier bed (face, 1,000 feet east of shaft, 2-foot 6-inch bed and cut).	212	11501	A	1	10.95	33.45	45.24	10.36	3.69	....	....	....	....	9.5	....	....
				2	....	37.56	50.81	11.63	4.14	....	....	....	....	....	....	....
				3	....	42.50	57.50	....	4.68	....	....	....	....	....	....	....
Same (face, 600 feet northeast of shaft, 2-foot 6-inch bed and cut).	212	11502	A	1	10.23	32.18	46.44	11.15	4.23	....	....	....	....	8.6	....	....
				2	....	35.85	51.73	12.42	4.71	....	....	....	....	....	....	....
				3	....	40.93	59.07	....	5.38	....	....	....	....	....	....	....
Same (face, 600 feet southwest of shaft, 2-foot 6-inch bed and cut).	212	11503	A	1	11.15	33.42	45.51	9.92	3.91	....	....	....	....	9.8	....	....
				2	....	37.61	51.23	11.16	4.40	....	....	....	....	....	....	....
				3	....	42.33	57.67	....	4.95	....	....	....	....	....	....	....
Same (composite of samples 11501, 11502, and 11503).	212	11504	A	1	10.77	32.82	46.04	10.37	4.00	5.37	63.72	.90	15.64	9.3	6,389	11,500
				2	....	36.78	51.60	11.62	4.48	4.67	71.41	1.01	6.81	....	7,160	12,888
				3	....	41.62	58.38	....	5.07	5.28	80.80	1.14	7.71	....	8,102	14,584
HARRISON COUNTY.																
Cainsville; sec. 13, T. 65 N., R. 26 W.; Cainsville mine, Cainsville bed; main west air course, 600 feet west and 50 feet north of shaft, 3-foot 10½-inch bed.	212	12679	B	1	12.99	39.71	35.38	11.92	5.24	....	....	....	....	7.8	5,864	10,555
				2	....	45.64	40.66	13.70	6.02	....	....	....	....	....	6,739	12,130
				3	....	52.88	47.12	....	6.98	....	....	....	....	....	7,808	14,054
Same (south entry 2, off main east entry, 150 feet east and 350 feet south of shaft, 3-foot 9½-inch bed).	213	12680	B	1	11.70	38.79	38.80	10.71	5.13	....	....	....	....	8.8	6,124	11,023
				2	....	43.93	43.94	12.13	5.81	....	....	....	....	....	6,935	12,483
				3	....	49.99	50.01	....	6.61	....	....	....	....	....	7,892	14,206
Same (south entry 2, off main west entry, 400 feet southwest of shaft, 3-foot 9½-inch bed).	213	12681	B	1	13.12	38.88	37.32	10.68	4.78	....	....	....	....	10.1	6,046	10,883
				2	....	44.75	42.96	12.29	5.50	....	....	....	....	....	6,959	12,526
				3	....	51.02	48.98	....	6.27	....	....	....	....	....	7,934	14,281
Melbourne, ½ mile east of; sec. 25, T. 62 N., R. 26 W.; Trenton No. 1 mine, Lexington bed (face, 90 feet south of shaft, 3-foot 3-inch bed, 2-foot 10-inch cut).	213	11374	A	1	15.16	37.32	40.54	6.98	2.86	....	....	....	....	11.6	....	....
				2	....	43.99	47.78	8.23	3.37	....	....	....	....	....	....	....
				3	....	47.94	52.06	....	3.67	....	....	....	....	....	....	....
Same (face, 85 feet northeast of shaft, 3-foot 3-inch bed, 2-foot 10-inch cut).	213	11375	A	1	15.05	37.35	40.19	7.41	3.33	....	....	....	....	12.1	....	....
				2	....	43.97	47.31	8.72	3.92	....	....	....	....	....	....	....
				3	....	48.17	51.83	....	4.29	....	....	....	....	....	....	....
Same (face, 30 feet north of shaft, 3-foot 3-inch bed, 2-foot 10-inch cut).	213	11376	A	1	15.77	35.83	42.86	5.54	2.75	....	....	....	....	12.4	....	....
				2	....	42.54	50.88	6.58	3.26	....	....	....	....	....	....	....
				3	....	45.53	54.47	....	3.49	....	....	....	....	....	....	....
Same (composite of samples 11374, 11375, and 11376)	213	11377	A	1	15.34	37.14	40.74	6.78	3.03	5.89	61.18	1.10	22.02	12.0	6,166	11,099
				2	....	43.87	48.12	8.01	3.58	4.95	72.27	1.30	9.89	....	7,283	13,109
				3	....	47.69	52.31	....	3.89	5.38	78.56	1.41	10.76	....	7,917	14,251

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
MISSOURI—Continued.																
HENRY COUNTY.																
Calhoun, 2½ miles west of; sec. 25, T. 34 N., R. 26 W.; Parks No. 1 mine, Tebo bed (face, 700 feet southeast of mine mouth, 2-foot 8-inch bed, 2-foot 7-inch cut).	213	11314	A	1	10.58	36.04	41.31	12.07	3.73	......	......	......	......	8.6	......	......
				2	......	40.30	46.20	13.50	4.17	......	......	......	......	......	......	......
				3	......	46.59	53.41	......	4.82	......	......	......	......	......	......	......
Same (face, 700 feet southwest of mine mouth, 2-foot 8-inch bed, 2-foot 7-inch cut).	213	11315	A	1	10.55	38.91	39.99	10.55	3.51	......	......	......	......	7.9	......	......
				2	......	43.50	44.71	11.79	3.92	......	......	......	......	......	......	......
				3	......	49.32	50.68	......	4.44	......	......	......	......	......	......	......
Same (face, 800 feet west of mine mouth, 2-foot 8-inch bed, 2-foot 7-inch cut).	213	11316	A	1	10.90	36.95	40.29	11.86	3.50	......	......	......	......	9.5	......	......
				2	......	41.47	45.22	13.31	3.93	......	......	......	......	......	......	......
				3	......	47.84	52.16	......	4.53	......	......	......	......	......	......	......
Same (composite of samples 11314, 11315, and 11316)	213	11317	A	1	10.60	37.24	40.65	11.51	3.58	5.41	61.80	0.90	16.80	8.7	6,249	11,248
				2	......	41.66	45.46	12.88	4.00	4.73	69.13	1.01	8.25	......	6,990	12,582
				3	......	47.82	52.18	......	4.59	5.43	79.35	1.16	9.47	......	8,023	14,441
Clinton, 3½ miles northwest of; sec. 28, T. 42 N., R. 26 W.; Lane No. 1 mine, Tebo bed (face, 210 feet northwest of entry mouth, 30-inch bed and cut).	214	11248	A	1	8.54	38.02	40.87	12.57	4.22	......	......	......	......	6.4	......	......
				2	......	41.57	44.69	13.74	4.61	......	......	......	......	......	......	......
				3	......	48.19	51.81	......	5.34	......	......	......	......	......	......	......
Same (face, 300 feet northeast of entry mouth, 2-foot 6-inch bed and cut).	214	11249	A	1	8.38	36.88	40.54	14.20	3.71	......	......	......	......	5.3	......	......
				2	......	40.25	44.25	15.50	4.05	......	......	......	......	......	......	......
				3	......	47.63	52.37	......	4.79	......	......	......	......	......	......	......
Same (face, 230 feet north of entry mouth, 2-foot 6-inch bed and cut).	214	11250	A	1	10.73	35.35	41.20	12.72	3.74	......	......	......	......	7.6	......	......
				2	......	39.60	46.15	14.25	4.19	......	......	......	......	......	......	......
				3	......	46.18	53.82	......	4.89	......	......	......	......	......	......	......
Same (composite of samples 11248, 11249 and 11250)	214	11251	A	1	9.13	37.43	40.43	13.01	3.81	5.37	62.50	.90	14.41	6.4	6,365	11,457
				2	......	41.19	44.49	14.32	4.19	4.80	68.78	.99	6.92	......	7,005	12,609
				3	......	48.07	51.93	......	4.89	5.60	80.27	1.16	8.08	......	8,176	14,717
1 mile north of; sec. 34, T. 42 N., R. 26 W.; Neil No. 1 mine (face, 200 feet southwest of shaft, 2-foot 6-inch bed, 2-foot 4-inch cut).	214	11302	A	1	8.15	36.57	38.85	16.43	4.76	......	......	......	......	6.8	......	......
				2	......	39.81	42.30	17.89	5.18	......	......	......	......	......	......	......
				3	......	48.48	51.52	......	6.31	......	......	......	......	......	......	......
Same (face, 100 feet northwest of shaft, 2-foot 6-inch bed, 2-foot 4-inch cut).	214	11303	A	1	10.93	35.07	39.58	14.42	3.87	......	......	......	......	9.3	......	......
				2	......	39.37	44.44	16.19	4.34	......	......	......	......	......	......	......
				3	......	46.98	53.02	......	5.18	......	......	......	......	......	......	......
Same (face, 75 feet east of shaft, 2-foot 6-inch bed, 2-foot 4-inch cut).	214	11304	A	1	18.78	32.09	37.63	11.50	2.34	......	......	......	......	15.9	......	......
				2	......	39.51	46.33	14.16	2.86	......	......	......	......	......	......	......
				3	......	46.03	53.97	......	3.36	......	......	......	......	......	......	......

Same (composite of samples 11302, 11303, and 11304).	214	11305	A	1	12.57	34.68	38.77	13.98	3.65	5.49	58.88	.96	17.54	10.7	5,951	10,712
				2	......	39.67	44.34	15.99	4.17	4.68	66.78	1.10	7.28	......	6,807	12,253
				3	......	47.22	52.78	......	4.96	5.57	79.49	1.31	8.67	......	8,102	14,584
2 miles east of; sec. 7, T. 41 N., R. 26 W.; Pharis No. 1 mine, Jordan bed (face, 100 feet west of shaft, 22-inch bed, 20-inch cut).	215	11370	A	1	12.29	33.26	41.59	12.86	2.43	......	......	......	......	10.1	......	......
				2	......	37.92	47.42	14.66	2.77	......	......	......	......	......	......	......
				3	......	44.43	55.57	......	3.25	......	......	......	......	......	......	......
Same (face, 100 feet south of shaft, 22-inch bed, 20-inch cut).	215	11371	A	1	13.07	34.34	44.28	8.31	2.06	......	......	......	......	10.4	......	......
				2	......	39.50	50.94	9.56	2.37	......	......	......	......	......	......	......
				3	......	43.68	56.32	......	2.62	......	......	......	......	......	......	......
Same (face, 50 feet east of shaft, 22-inch bed, 20-inch cut).	215	11372	A	1	12.82	33.32	43.06	10.80	3.00	......	......	......	......	10.4	......	......
				2	......	38.22	49.39	12.39	3.44	......	......	......	......	......	......	......
				3	......	43.62	56.38	......	3.93	......	......	......	......	......	......	......
Same (composite of samples 11370, 11371, and 11372).	215	11373	A	1	12.76	33.05	43.64	10.55	2.57	5.53	61.87	1.12	18.36	10.3	6,205	11,169
				2	......	37.89	50.02	12.09	2.95	4.71	70.92	1.28	8.05	......	7,113	12,803
				3	......	43.10	56.90	......	3.36	5.36	80.67	1.46	9.15	......	8,091	14,564
3 miles southeast of; sec. 25, T. 41 N., R. 26 W.; Sheldon No. 1 mine, Jordan bed (face, 80 feet southwest of shaft, 2-foot 2-inch bed and cut).	215	11262	A	1	9.56	34.72	43.11	12.61	5.67	......	......	......	......	7.9	......	......
				2	......	38.39	47.67	13.94	6.27	......	......	......	......	......	......	......
				3	......	44.61	55.39	......	7.29	......	......	......	......	......	......	......
Same (face, 110 feet south of shaft, 2-foot 2-inch bed and cut).	215	11263	A	1	10.64	33.64	43.77	11.95	5.37	......	......	......	......	8.9	......	......
				2	......	37.65	48.98	13.37	6.01	......	......	......	......	......	......	......
				3	......	43.46	56.54	......	6.94	......	......	......	......	......	......	......
Same (face, 110 feet southeast of shaft, 2-foot 2-inch bed and cut).	215	11264	A	1	9.72	36.62	43.42	10.24	5.13	......	......	......	......	8.1	......	......
				2	......	40.56	48.10	11.34	5.68	......	......	......	......	......	......	......
				3	......	45.75	54.25	......	6.41	......	......	......	......	......	......	......
Same (composite of samples 11262, 11263, and 11264).	215	11265	A	1	9.90	35.03	43.31	11.76	5.38	5.39	61.99	1.06	14.42	8.3	6,356	11,441
				2	......	38.88	48.07	13.05	5.97	4.76	68.80	1.18	6.24	......	7,055	12,699
				3	......	44.72	55.28	......	6.87	5.47	79.13	1.36	7.17	......	8,114	14,605
Deepwater, in eastern part of; sec. 14, T. 40 N., R. 26 W.; Dickey No. 1 pit, Jordan bed (face, on north side of pit, 2-foot 2-inch bed, 2-foot cut).	215	11310	A	1	9.95	33.76	46.55	9.74	3.49	......	......	......	......	8.3	......	......
				2	......	37.49	51.69	10.82	3.88	......	......	......	......	......	......	......
				3	......	42.04	57.96	......	4.35	......	......	......	......	......	......	......
Same (face, on east side of pit, 2-foot 2-inch bed, 2-foot cut).	215	11311	A	1	9.82	33.89	46.17	10.12	3.46	......	......	......	......	8.4	......	......
				2	......	37.58	51.20	11.22	3.84	......	......	......	......	......	......	......
				3	......	42.33	57.67	......	4.33	......	......	......	......	......	......	......
Same (face, on west side of pit, 2-foot 2-inch bed, 2-foot cut).	215	11312	A	1	9.55	34.10	45.83	10.52	3.90	......	......	......	......	7.7	......	......
				2	......	37.70	50.67	11.63	4.31	......	......	......	......	......	......	......
				3	......	42.66	57.34	......	4.88	......	......	......	......	......	......	......
Same (composite of samples 11310, 11311, and 11312).	216	11313	A	1	9.70	33.78	46.26	10.26	3.63	5.38	65.19	1.08	14.51	8.1	6,566	11,819
				2	......	37.41	51.23	11.36	4.02	4.76	72.19	1.14	6.53	......	7,271	13,088
				3	......	42.21	57.79	......	4.54	5.37	81.44	1.29	7.36	......	8,203	14,765
¾ mile east of; sec. 13, T. 40 N., R. 26 W.; Hurst No. 1 mine, Jordan bed (face, 100 feet northeast of shaft, 3-foot 4-inch bed and cut).	216	11252	A	1	9.31	36.38	41.57	12.74	5.07	......	......	......	......	7.3	......	......
				2	......	40.12	45.83	14.05	5.59	......	......	......	......	......	......	......
				3	......	46.68	53.32	......	6.50	......	......	......	......	......	......	......
Same (face, 350 feet west of shaft, 3-foot 4-inch bed and cut).	216	11253	A	1	11.55	33.95	41.80	12.70	4.01	......	......	......	......	10.0	......	......
				2	......	38.38	47.26	14.36	4.53	......	......	......	......	......	......	......
				3	......	44.82	55.18	......	5.29	......	......	......	......	......	......	......
Same (face, 450 feet northeast of shaft, 3-foot 4-inch bed and cut).	216	11254	A	1	9.52	34.52	41.89	14.07	3.91	......	......	......	......	7.9	......	......
				2	......	38.15	46.30	15.55	4.32	......	......	......	......	......	......	......
				3	......	45.17	54.83	......	5.12	......	......	......	......	......	......	......
Same (composite of samples 11252, 11253, and 11254).	216	11255	A	1	10.10	34.83	41.76	13.31	4.32	5.18	60.94	.99	15.26	8.4	6,199	11,158
				2	......	38.74	46.46	14.80	4.81	4.52	67.78	1.10	6.99	......	6,895	12,411
				3	......	45.47	54.53	......	5.65	5.31	79.55	1.29	8.20	......	8,093	14,567

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
MISSOURI—Continued.																
HENRY COUNTY—continued.																
Lewis, ¾ mile north of; sec. 9, T. 42 N., R. 25 W.; Pigg No. 1 mine, Tebo bed (face, 1,300 feet west of shaft).	216	11267	A	1	11.91	33.31	39.98	14.80	4.86	......	......	......	......	10.2	......	......
				2	......	37.81	45.39	16.80	5.52	......	......	......	......	......	......	......
				3	......	45.44	54.56	......	6.63	......	......	......	......	......	......	......
Same (face, 1,300 feet southwest of shaft, 2-foot 6-inch bed and cut).	216	11268	A	1	11.08	36.28	39.96	12.68	3.82	......	......	......	......	9.4	......	......
				2	......	40.80	44.94	14.26	4.30	......	......	......	......	......	......	......
				3	......	47.59	52.41	......	5.02	......	......	......	......	......	......	......
Same (face, 1,400 feet southwest of shaft, 2-foot 6-inch bed and cut).	216	11269	A	1	11.04	38.03	40.56	10.37	3.39	......	......	......	......	9.3	......	......
				2	......	42.75	45.59	11.66	3.81	......	......	......	......	......	......	......
				3	......	48.39	51.61	......	4.31	......	......	......	......	......	......	......
Same (composite of samples 11267, 11268, and 11269, 2-foot 6-inch bed and cut).	216	11270	A	1	11.44	34.67	41.13	12.76	4.25	5.53	59.88	0.95	16.63	9.6	6,117	11,011
				2	......	39.15	46.44	14.41	4.80	4.81	67.62	1.07	7.29	......	6,907	12,433
				3	......	45.74	54.26	......	5.61	5.62	79.01	1.25	8.51	......	8,070	14,526
LINN COUNTY.																
Brookfield, 1¾ miles east of; sec. 20, T. 57 N., R. 19 W.; Crandall No. 1 mine, Tebo bed (face, 750 feet southeast of shaft, 2-foot 5-inch bed, 2-foot 2-inch cut).	217	11421	A	1	14.78	36.64	38.23	10.35	4.62	......	......	......	......	12.7	......	......
				2	......	42.99	44.87	12.14	5.42	......	......	......	......	......	......	......
				3	......	48.93	51.07	......	6.17	......	......	......	......	......	......	......
Same (face, 750 feet southwest of shaft, 2-foot 5-inch bed, 2-foot 2-inch cut).	217	11422	A	1	14.46	36.99	41.19	7.36	4.28	......	......	......	......	12.2	......	......
				2	......	43.24	48.16	8.60	5.00	......	......	......	......	......	......	......
				3	......	47.31	52.69	......	5.47	......	......	......	......	......	......	......
Same (face, 800 feet east of shaft, 2-foot 5-inch bed, 2-foot 2-inch cut).	217	11423	A	1	15.81	36.18	41.45	6.56	3.78	......	......	......	......	13.7	......	......
				2	......	42.97	49.24	7.79	4.49	......	......	......	......	......	......	......
				3	......	46.60	53.40	......	4.87	......	......	......	......	......	......	......
Same (composite of samples 11421, 11422, and 11423).	217	11424	A	1	15.09	36.05	40.71	8.15	4.21	5.81	60.67	.99	20.17	12.9	6,100	10,980
				2	......	42.46	47.94	9.60	4.96	4.86	71.45	1.17	7.96	......	7,184	12,931
				3	......	46.97	53.03	......	5.49	5.38	79.04	1.29	8.80	......	7,947	14,305

3 miles southeast of; sec. 16, T. 57 N., R. 19 W.; Walker No. 1 mine, Tebo bed (face, 250 feet southwest of shaft, 2-foot 6-inch bed, 2-foot 2-inch cut).	217	11413	A	1	14.14	36.74	40.36	8.76	4.36	....	....	....	....	11.8	....	....
				2	....	42.79	47.01	10.20	5.08	....	....	....	....	....	....	....
				3	....	47.65	52.35	....	5.66	....	....	....	....	....	....	....
Same (face, 200 feet northeast of shaft, 2-foot 6-inch bed, 2-foot 2-inch cut).	217	11414	A	1	15.10	36.67	42.51	5.72	3.82	....	....	....	....	12.7	....	....
				2	....	43.19	50.07	6.74	4.50	....	....	....	....	....	....	....
				3	....	46.31	53.69	....	4.83	....	....	....	....	....	....	....
Same (face, 200 feet southeast of shaft, 2-foot 6-inch bed, 2-foot 2-inch cut).	217	11415	A	1	15.48	35.93	40.54	8.05	4.89	....	....	....	....	13.3	....	....
				2	....	42.51	47.97	9.52	5.79	....	....	....	....	....	....	....
				3	....	46.98	53.02	....	6.40	....	....	....	....	....	....	....
Same (composite of samples 11413, 11414, and 11415).	217	11416	A	1	14.92	36.39	41.30	7.39	4.14	5.94	61.01	1.05	20.47	12.6	6,179	11,122
				2	....	42.77	48.54	8.69	4.87	5.03	71.71	1.23	8.47	....	7,263	13,073
				3	....	46.84	53.16	....	5.33	5.51	78.54	1.35	9.27	....	7,954	14,317
Marceline, ½ mile west of; sec. 20, T. 57 N., R. 18 W.; Landreth No. 1 mine, Tebo bed (face, 600 feet east of shaft, 2-foot 4-inch bed and cut).	218	11417	A	1	13.02	38.38	41.52	7.08	4.11	....	....	....	....	10.9	....	....
				2	....	44.13	47.73	8.14	4.73	....	....	....	....	....	....	....
				3	....	48.04	51.96	....	5.15	....	....	....	....	....	....	....
Same (face, 600 feet north of shaft, 2-foot 4-inch bed and cut).	218	11418	A	1	12.09	38.53	40.96	8.42	4.51	....	....	....	....	10.0	....	....
				2	....	43.83	46.59	9.58	5.13	....	....	....	....	....	....	....
				3	....	48.47	51.53	....	5.67	....	....	....	....	....	....	....
Same (face, 600 feet south of shaft, 2-foot 4-inch bed and cut).	218	11419	A	1	12.65	39.57	41.02	6.76	4.11	....	....	....	....	10.4	....	....
				2	....	45.30	46.96	7.74	4.71	....	....	....	....	....	....	....
				3	....	49.10	50.90	....	5.11	....	....	....	....	....	....	....
Same (composite of samples 11417, 11418, and 11419).	218	11420	A	1	12.51	39.29	40.76	7.44	4.27	5.85	63.02	1.06	18.34	10.4	6,380	11,484
				2	....	44.91	46.59	8.50	4.88	5.10	72.03	1.23	8.26	....	7,292	13,126
				3	....	49.08	50.92	....	5.33	5.57	78.72	1.34	9.04	....	7,969	14.344
PUTNAM COUNTY.																
Mendota, 1 mile southeast of; sec. 1, T. 66 N., R. 18 W.; Mendota No. 2 mine, Lexington bed (face, 2,500 feet southwest of shaft, 3-foot bed, 2-foot 10-inch cut).	218	11398	A	1	18.56	31.90	40.07	9.47	4.25	....	....	....	....	15.1	....	....
				2	....	39.17	49.20	11.63	5.22	....	....	....	....	....	....	....
				3	....	44.32	55.68	....	5.91	....	....	....	....	....	....	....
Same (face 3,200 feet south of shaft, 3-foot bed, 2-foot 10-inch cut).	218	11399	A	1	17.53	34.14	39.80	8.53	3.74	....	....	....	....	14.3	....	....
				2	....	41.40	48.26	10.34	4.54	....	....	....	....	....	....	....
				3	....	46.17	53.83	....	5.06	....	....	....	....	....	....	....
Same (face, 3,100 feet southeast of shaft, 3-foot bed, 2-foot 10-inch cut).	218	11400	A	1	19.11	31.49	37.92	11.48	4.33	....	....	....	....	16.1	....	....
				2	....	38.93	46.88	14.19	5.38	....	....	....	....	....	....	....
				3	....	45.37	54.63	....	6.27	....	....	....	....	....	....	....
Same (composite of samples 11398, 11399, and 11400).	218	11401	A	1	18.45	32.62	39.03	9.90	4.09	5.76	55.71	.91	23.63	15.2	5,578	10,040
				2	....	40.00	47.86	12.14	5.02	4.55	68.31	1.12	8.86	....	6,840	12,312
				3	....	45.53	54.47	....	5.71	5.18	77.75	1.27	10.09	....	7,785	14,013
Unionville, 2½ miles northeast of; NW. ¼ sec. 30, T. 66 N., R. 19 W.; Anderson mine, Lexington bed (in mine, about 175 feet southwest of shaft; 2-foot 8½-inch bed, 2-foot 6-inch cut).	219	*14880	B	1	16.51	34.39	37.33	11.77	3.39	....	....	....	....	12.8	5,600	10,080
				2	....	41.19	44.71	14.10	4.06	....	....	....	....	....	6,707	12,073
				3	....	47.95	52.05	....	4.73	....	....	....	....	....	7,808	14,054
Same (about 150 feet east of shaft; 2-foot 10-inch bed, 2-foot 6½-inch cut).	219	*14881	B	1	16.52	34.76	37.39	11.33	2.88	....	....	....	....	12.8	5,563	10,013
				2	....	41.64	44.79	13.57	3.45	....	....	....	....	....	6,664	11,995
				3	....	48.18	51.82	....	3.99	....	....	....	....	....	7,710	13,878

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
MISSOURI—Continued.																
RANDOLPH COUNTY.																
Huntsville, southern part of; sec. 36, T. 54 N., R. 15 W.; Carson No. 1 mine, Bevier bed (face, 250 feet southeast of shaft, 4-foot bed and cut).	219	11452	A	1	13.35	35.06	41.32	10.27	5.56	....	....	....	....	11.5	....	....
				2	....	40.46	47.69	11.85	6.42	....	....	....	....	....	....	....
				3	....	45.90	54.10	....	7.28	....	....	....	....	....	....	....
Same (face, 300 feet north of shaft, 4-foot bed and cut)	219	11453	A	1	13.36	34.79	42.02	9.83	5.20	....	....	....	....	11.5	....	....
				2	....	40.15	48.50	11.35	6.00	....	....	....	....	....	....	....
				3	....	45.29	54.71	....	6.77	....	....	....	....	....	....	....
Same (face, 200 feet southwest of shaft, 4-foot bed and cut).	219	11454	A	1	13.31	34.71	41.67	10.31	5.12	....	....	....	....	11 6	....	....
				2	....	40.04	48.07	11.89	5.91	....	....	....	....	....	....	....
				3	....	45.44	54.56	....	6.71	....	....	....	....	....	....	....
Same (composite of samples 11452, 11453, and 11454).	219	11455	A	1	13.23	34.99	41.53	10.25	5.19	5.58	60.25	0.92	17.81	11.5	6,101	10,982
				2	....	40.33	47.86	11.81	5.98	4.74	69.44	1.06	6.97	....	7,031	12,656
				3	....	45.73	54.27	....	6.78	5.37	78.74	1.20	7.91	....	7,972	14,350
¾ mile south of: sec. 36, T. 54 N., R. 15 W.; Northern No. 2 mine, Bevier bed (face, 4,000 feet south of shaft, 4-foot bed and cut).	219	11448	A	1	12.46	36.94	42.80	7.80	2.99	....	....	....	....	10.6	....	....
				24	....	42.20	48.89	8.91	3.42	....	....	....	....	....	....	....
				3	....	46.33	53.67	....	3.75	....	....	....	....	....	....	....
Same (face, 3,800 feet northeast of shaft, 4-foot bed and cut).	219	11449	A	1	13.03	35.75	41.82	9.40	4.34	....	....	....	....	11.2	....	....
				2	....	41.11	48.08	10.81	4.99	....	....	....	....	....	....	....
				3	....	46.09	53.91	....	5.59	....	....	....	....	....	....	....
Same (face, 3,500 feet southeast of shaft, 4-foot bed and cut).	219	11450	A	1	10.53	36.50	43.90	9.07	4.46	....	....	....	....	8.4	....	....
				2	....	40.80	49.06	10.14	4.98	....	....	....	....	....	....	....
				3	....	45.40	54.60	....	5.54	....	....	....	....	....	....	....
Same (composite of samples 11448, 11449, and 11450).	220	11451	A	1	12.12	36.32	42.88	8.68	3.92	5.58	63.32	1.02	17.48	10.1	6,363	11,453
				2	....	41.33	48.79	9.88	4.46	4.81	72.05	1.16	7.64	....	7,240	13,032
				3	....	45.86	54.14	....	4.96	5.34	79.95	1.29	8.47	....	8,034	14,461
Renick, ½ mile west of; sec. 31, T. 53 N., R. 13 W.; Orris No. 1 mine; Mulky bed (face, 300 feet southwest of shaft, 18-inch bed and cut).	220	11497	A	1	9.82	38.15	43.08	8.95	4.68	....	....	....	....	7.9	....	....
				2	....	42.30	47.78	9.92	5.20	....	....	....	....	....	....	....
				3	....	46.96	53.04	....	5.77	....	....	....	....	....	....	....
Same (face, 300 feet northwest of shaft, 18-inch bed and cut).	220	11498	A	1	7.38	41.09	43.68	7.85	4.04	....	....	....	....	5.1	....	....
				2	....	44.36	47.16	8.48	4.36	....	....	....	....	....	....	....
				3	....	48.47	51.53	....	4.76	....	....	....	....	....	....	....

Same (face, 300 feet west of shaft, 18-inch bed and cut).	220	11499	A	1	8.83	39.76	42.95	8.46	5.35	....	....	....	....	6.7	....	....
				2	....	43.61	47.11	9.28	5.87	....	....	....	....	....	....	....
				3	....	48.07	51.93	....	6.47	....	....	....	....	....	....	....
Same (composite of samples 11497, 11498, and 11499, 18-inch bed and cut).	220	11500	A	1	8.71	39.87	42.88	8.54	4.64	5.61	66.08	.94	14.19	6.6	6,706	12,071
				2	....	43.67	46.98	9.35	5.08	5.08	72.38	1.03	7.08	....	7,346	13,223
				3	....	48.17	51.83	....	5.60	5.60	79.84	1.14	7.82	....	8,108	14,585
RAY COUNTY.																
Vibbard, ½ mile south of; sec. 27, T. 53 N., R. 29 W.; Vibbard No. 1 mine, Lexington bed (face, 250 feet south of shaft, 2-foot 4-inch bed, 2-foot cut).	220	11365	A	1	11.89	38.00	42.68	7.43	2.95	....	....	....	....	7.2	....	....
				2	....	43.13	48.44	8.43	3.35	....	....	....	....	....	....	....
				3	....	47.10	52.90	....	3.66	....	....	....	....	....	....	....
Same (face, 250 feet east of shaft, 2-foot 4-inch bed, 2-foot cut).	220	11366	A	1	13.58	36.55	43.77	6.10	2.23	....	....	....	....	8.4	....	....
				2	....	42.29	50.65	7.06	2.58	....	....	....	....	....	....	....
				3	....	45.50	54.50	....	2.78	....	....	....	....	....	....	....
Same (face, 200 feet west of shaft, 2-foot 4-inch bed, 2-foot cut).	220	11367	A	1	13.83	36.47	42.60	7.10	3.00	....	....	....	....	9.2	....	....
				2	....	42.32	49.44	8.24	3.48	....	....	....	....	....	....	....
				3	....	46.12	53.88	....	3.79	....	....	....	....	....	....	....
Same (composite of samples 11365, 11366, and 11367).	220	11368	A	1	13.08	36.94	43.15	6.83	2.79	5.73	64.22	.99	19.44	8.3	6,417	11,551
				2	....	42.50	49.64	7.86	3.21	4.92	73.89	1.14	8.98	....	7,383	13,289
				3	....	46.13	53.87	....	3.48	5.34	80.19	1.24	9.75	....	8,013	14,423
VERNON COUNTY.																
Moundville, northwest corner of; sec. 5, T. 34 N., R. 32 W.; Brown No. 1 mine (face, 40 feet northwest of shaft, 2-foot 2-inch cut).	221	11201	A	1	5.68	33.67	50.05	10.60	5.00	....	....	....	....	4.2	....	....
				2	....	35.70	53.06	11.24	5.30	....	....	....	....	....	....	....
				3	....	40.22	59.78	....	5.97	....	....	....	....	....	....	....
Same (face, 100 feet northeast of shaft, 2-foot 2-inch cut).	221	11202	A	1	5.98	33.10	52.48	8.44	3.86	....	....	....	....	4.6	....	....
				2	....	35.21	55.81	8.98	4.11	....	....	....	....	....	....	....
				3	....	38.69	61.31	....	4.52	....	....	....	....	....	....	....
Same (face, 125 feet north of shaft, 2-foot 2-inch cut).	221	11203	A	1	7.93	30.05	50.99	11.03	5.89	....	....	....	....	6.6	....	....
				2	....	32.64	55.38	11.98	6.40	....	....	....	....	....	....	....
				3	....	37.08	62.92	....	7.27	....	....	....	....	....	....	....
Same (composite of samples 11201, 11202, and 11203).	221	11204	A	1	6.50	32.61	50.83	10.06	4.95	5.21	67.97	1.09	10.72	5.1	6,921	12,458
				2	....	34.88	54.36	10.76	5.29	4.80	72.69	1.17	5.29	....	7,402	13,324
				3	....	39.09	60.91	....	5.93	5.38	81.46	1.31	5.92	....	8,295	14,931
Panama, 2 miles south of; sec. 4, T. 37 N., R. 31 W.; Jones No. 1 mine, Rich Hill bed (face, 450 feet east of shaft, 5-foot cut).	221	11205	A	1	6.22	35.36	42.89	15.53	5.45	....	....	....	....	5.0	....	....
				2	....	37.70	45.74	16.56	5.81	....	....	....	....	....	....	....
				3	....	45.18	54.82	....	6.96	....	....	....	....	....	....	....
Same (face, 1,000 feet southeast of shaft, 5-foot cut)	221	11206	A	1	6.49	34.40	43.64	15.47	5.88	....	....	....	....	4.7	....	....
				2	....	36.79	46.67	16.54	6.29	....	....	....	....	....	....	....
				3	....	44.08	55.92	....	7.54	....	....	....	....	....	....	....
Same (face, 100 feet east of shaft, 5-foot cut)......	221	11207	A	1	5.21	36.67	45.51	12.61	5.92	....	....	....	....	3.5	....	....
				2	....	38.69	48.01	13.30	6.25	....	....	....	....	....	....	....
				3	....	44.63	55.37	....	7.21	....	....	....	....	....	....	....
Same (composite of samples 11205, 11206, and 11207).	221	11208	A	1	6.02	35.94	43.37	14.67	5.84	5.27	63.55	1.08	9.64	4.4	6,579	11,842
				2	....	38.24	46.15	15.61	6.21	4.89	67.62	1.10	4.57	....	7,001	12,602
				3	....	45.31	54.69	....	7.36	5.79	80.13	1.30	5.42	....	8,296	14,933

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
MONTANA.																
BLAINE COUNTY.																
Cleveland, 4 miles northeast of, about 25 miles southeast of Chinook; SW. ¼ sec. 35, T. 30 N., R. 20 E; Cook mine (face of entry, 500 feet from mine mouth; 3 foot 6½ inch bed, 3-foot 1-inch cut.)	222	*14782	B	1	18.77	31.48	40.40	9.35	0.73	......	......	......	......	7.1	5,075	9,135
				2	....	38.76	49.73	11.51	.90	......	......	......	......	....	6,248	11,246
				3	....	43.80	56.20	....	1.02	......	......	......	......	....	7,061	12,710
CARBON COUNTY.																
Bear Creek, 1¼ miles west of; sec. 6, T. 8 S., R. 21 E; No. 2 mine, No. 2 bed (pillar between rooms 9 and 10, off north entry 2; 6-foot 2¼-inch bed and cut).	222	*15151	A	1	9.94	35.05	43.82	11.19	1.75	......	......	......	......	3.1	5,897	10,615
				2	....	38.92	48.65	12.43	1.94	......	......	......	......	....	6,548	11,786
				3	....	44.44	55.56	....	2.22	......	......	......	......	....	7,477	13,459
Same (position, bed, and cut same as sample 15151).	222	*15145	A	1	10.07	34.78	44.10	11.05	1.83	......	......	......	......	3.2	5,935	10,683
				2	....	38.68	49.03	12.29	2.03	......	......	......	......	....	6,600	11,880
				3	....	44.10	55.90	....	2.31	......	......	......	......	....	7,526	13,545
Same (face of room 10, west entry 1; 125 feet in from entry; 6-foot 4-inch bed and cut).	222	*15146	A	1	10.39	34.12	47.56	7.93	1.31	......	......	......	......	3.4	6,096	10,973
				2	....	38.07	53.08	8.85	1.46	......	......	......	......	....	6,808	12,245
				3	....	41.77	58.23	....	1.60	......	......	......	......	....	7,464	13,435
Same (position, bed, and cut same as sample 15146).	222	*15152	A	1	10.40	34.66	46.68	8.26	1.28	......	......	......	......	3.2	6,095	10,971
				2	....	38.68	52.10	9.22	1.43	......	......	......	......	....	6,803	12,245
				3	....	42.61	57.39	....	1.58	......	......	......	......	....	7,494	13,489
Same (face of room 9 off east entry 9; 45 feet in from entry; 5-foot 6½-inch bed and cut).	222	*15147	A	1	10.66	35.18	44.79	9.37	1.66	......	......	......	......	3.6	5,958	10,724
				2	....	39.38	50.13	10.49	1.86	......	......	......	......	....	6,669	12,004
				3	....	44.00	56.00	....	2.08	......	......	......	......	....	7,451	13,412
Same (position, bed, and cut same as sample 15147).	222	*15153	A	1	10.67	34.75	45.13	9.45	1.65	......	......	......	......	3.7	5,950	10,710
				2	....	38.90	50.52	10.58	1.85	......	......	......	......	....	6,660	11,988
				3	....	43.50	56.50	....	2.07	......	......	......	......	....	7,448	13,406
Same (face of last crosscut, just started, in room 22, off east entry 4; 6-foot 3½-inch bed and cut).	222	*15148	A	1	11.07	34.11	46.67	8.15	1.47	......	......	......	......	3.9	6,028	10,850
				2	....	38.36	52.48	9.16	1.65	......	......	......	......	....	6,778	12,200
				3	....	42.23	57.77	....	1.82	......	......	......	......	....	7,461	13,430
Same (position, bed, and cut same as sample 15148).	222	*15154	A	1	11.11	33.65	46.89	8.35	1.44	......	......	......	......	3.9	6,010	10,818
				2	....	37,86	52.75	9.39	1.62	......	......	......	......	....	6,761	12,170
				3	....	41.78	58.22	....	1.79	......	......	......	......	....	7,461	13,430

Same (face of room 11, off rock tunnel entry, 35 feet in from entry, 5-foot 9-inch bed and cut).	222	*15149	A	1	10.80	35.06	44.02	10.12	1.82	......	......	......	......	3.8	5,885	10,593
				2	......	39.31	49.34	11.35	2.04	......	......	......	......	......	6,598	11,876
				3	......	44.34	55.66	......	2.30	......	......	......	......	......	7,443	13,397
Same (position, bed, and cut same as sample 15149).	222	*15155	A	1	10.75	35.19	43.60	10.46	1.80	......	......	......	......	3.9	5,861	10,550
				2	......	39.43	48.85	11.72	2.02	......	......	......	......	......	6,567	11,821
				3	......	44.67	55.33	......	2.29	......	......	......	......	......	7,439	13,390
Same (composite of samples 15146, 15147, 15148, and 15149).	223	*15150	A	1	10.66	35.29	45.14	8.91	1.58	5.55	61.22	1.58	21.16	3.7	5,986	10,775
				2	......	39.50	50.53	9.97	1.77	4.89	68.52	1.77	13.08	......	6,700	12,060
				3	......	43.87	56.13	......	1.97	5.43	76.11	1.97	14.52	......	7,442	13,396
Same (composite of samples 15152, 15153, 15154, and 15155).	223	*15156	A	1	10.65	34.10	46.11	9.14	1.53	5.50	60.87	1.58	21.38	3.7	5,997	10,795
				2	......	38.16	51.61	10.23	1.71	4.83	68.13	1.77	13.33	......	6,712	12,082
				3	......	42.51	57.49	......	1.90	5.38	75.90	1.97	14.85	......	7,477	13,459
Secs. 5 and 6, T. 8 S., R. 21 E; North Side mine, No. 3 bed (last pillar on second west main entry, 500 feet from drift mouth, 5-foot 6-inch bed and cut).	224	*15143	A	1	10.23	34.96	45.96	8.86	1.55	......	......	......	......	3.5	6,120	11,016
				2	......	38.95	51.18	9.87	1.73	......	......	......	......	......	6,818	12,272
				3	......	43.22	56.78	......	1.92	......	......	......	......	......	7,565	13,617
Same (position, bed, and cut same as sample 15143).	224	*15144	A	1	10.12	34.54	46.63	8.71	1.67	......	......	......	......	3.2	6,130	11,034
				2	......	38.43	51.88	9.69	1.86	......	......	......	......	......	6,820	12,276
				3	......	42.55	57.45	......	2.06	......	......	......	......	......	7,552	13,594
1 mile west of; sec. 6, T. 8 S., R. 21 E.; South Side mine, No. 3 bed (face of room 2 off north entry 1, 50 feet in from entry, 7-foot 11-inch bed, 6-foot 11-inch cut).	223	*15127	A	1	10.30	34.31	44.03	11.36	2.19	......	......	......	......	2.9	5,859	10,546
				2	......	38.25	49.09	12.66	2.44	......	......	......	......	......	6,532	11,758
				3	......	43.80	56.20	......	2.79	......	......	......	......	......	7,479	13,462
Same (position, bed, and cut same as sample 15127).	223	*15133	A	1	10.22	33.98	44.12	11.68	2.09	......	......	......	......	2.7	5,854	10,537
				2	......	37.85	49.14	13.01	2.33	......	......	......	......	......	6,520	11,736
				3	......	43.51	56.49	......	2.68	......	......	......	......	......	7,495	13,491
Same (in crosscut from main entry to back entry, 350 feet from mouth of main entry and 75 feet from face of back entry, 7-foot 8-inch bed, 7-foot 4-inch cut).	223	*15128	A	1	9.99	34.65	44.45	10.91	2.53	......	......	......	......	2.9	5,898	10,616
				2	......	38.50	49.38	12.12	2.81	......	......	......	......	......	6,553	11,795
				3	......	43.81	56.19	......	3.20	......	......	......	......	......	7,457	13,423
Same (position, bed, and cut same as sample 15128).	223	*15132	A	1	10.10	34.65	43.96	11.29	2.41	......	......	......	......	2.9	5,870	10,566
				2	......	38.54	48.90	12.56	2.68	......	......	......	......	......	6,529	11,752
				3	......	44.07	55.93	......	3.06	......	......	......	......	......	7,467	13,441
Same (face of south back entry 1, 150 feet from main slope, 7-foot 2-inch bed, 6-foot 7-inch cut).	223	*15129	A	1	9.80	33.62	45.02	11.56	1.95	......	......	......	......	3.2	5,910	10,638
				2	......	37.27	49.91	12.82	2.16	......	......	......	......	......	6,552	11,794
				3	......	42.75	57.25	......	2.48	......	......	......	......	......	7,516	13,529
Same (position, bed, and cut same as sample 15129).	223	*15131	A	1	9.60	33.47	45.37	11.56	2.00	......	......	......	......	3.1	5,900	10,620
				2	......	37.02	50.19	12.79	2.21	......	......	......	......	......	6,527	11,749
				3	......	42.45	57.55	......	2.53	......	......	......	......	......	7,485	13,473
Same (composite of samples 15127, 15128, and 15129).	224	*15130	A	1	9.95	33.93	44.81	11.31	2.26	5.28	59.13	1.41	20.61	3.0	5,885	10,593
				2	......	37.68	49.76	12.56	2.51	4.63	65.66	1.57	13.07	......	6,535	11,763
				3	......	43.09	56.91	......	2.87	5.29	75.09	1.80	14.95	......	7,473	13,451
Same (composite of samples 15131, 15132, and 15133).	224	*15134	A	1	9.87	34.00	44.61	11.52	2.25	5.32	59.79	1.47	19.65	2.9	5,882	10,588
				2	......	37.72	49.50	12.78	2.50	4.68	66.34	1.63	12.07	......	6,526	11,74
				3	......	43.25	56.75	......	2.87	5.37	76.06	1.87	13.83	......	7,482	13,468

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
MONTANA—Continued.																
CHOUTEAU COUNTY.																
Big Sandy, 6 miles northeast of; NW. ¼ SW. ¼ sec. 18 T. 28 N., R. 14 E., Mackton mine; Mackton bed (face of north entry on second level, 335 feet below mouth of slope and 140 feet from mouth of entry, 7-foot 5-inch bed, 5-foot 6-inch cut).	225	*14613	B	1	13.03	36.29	40.06	10.62	0.53	......	......	......	......	1.8	5,342	9,616
				2	......	41.73	46.06	12.21	.61	......	......	......	......	......	6,142	11,056
				3	......	47.53	52.47	......	.69	......	......	......	......	......	6,996	12,593
Same (face of upper south entry, 250 feet from mouth of entry and 150 feet below mouth of slope; 7-foot 6-inch bed, 6-foot 1-inch cut).	225	*14615	B	1	12.36	36.93	35.78	14.93	.70	......	......	......	......	1.9	5,052	9,094
				2	......	42.14	40.82	17.04	.80	......	......	......	......	......	5,764	10,375
				3	......	50.80	49.20	......	.96	......	......	......	......	......	6,948	12,506
CUSTER COUNTY.																
Terry, 4 miles west of; sec. 24, T. 12 N., R. 50 E.; Cameron mine (8-foot 1-inch bed, 3-foot 5-inch cut).	225	10677	B	1	28.16	......	......	15.91	.27	......	......	......	......	19.9	3,503	6,305
				2	......	......	......	22.15	.38	......	......	......	......	......	4,876	8,777
				3	......	......	......	......	.49	......	......	......	......	......	6,263	11,273
Westmore, 17 miles south of; sec. 34, T. 6 N., R. 57 E.; prospect.	226	*10910	B	1	38.67	27.02	25.10	9.21	.30	......	......	......	......	26.2	3,345	6,021
				2	......	44.06	40.92	15.02	.49	......	......	......	......	......	5,454	9,817
				3	......	51.84	48.16	......	.58	......	......	......	......	......	6,418	11,552
DAWSON COUNTY.																
Glendive, 15 miles northeast of; sec. 31, T. 18 N., R. 57 E.; prospect (right wall, 25 feet from outcrop).	226	*10914	B	1	31.52	28.61	33.54	6.33	.65	......	......	......	......	17.2	3,822	6,880
				2	......	41.78	48.98	9.24	.95	......	......	......	......	......	5,581	10,046
				3	......	46.03	53.97	......	1.05	......	......	......	......	......	6,149	11,068
6½ miles north of; sec. 27, T. 17 N., R. 55 E.; Snyder mine, middle bed of the Fort Union (face, 80 feet from mine mouth).	226	*11045	B	1	32.13	25.55	34.23	8.09	1.36	......	......	......	......	23.4	3,948	7,106
				2	......	37.65	50.43	11.92	2.00	......	......	......	......	......	5,817	10,471
				3	......	42.74	57.26	......	2.27	......	......	......	......	......	6,604	11,887
FERGUS COUNTY.																
Hilger, 13 miles north of; sec. 8, T. 19 N., R. 19 E., Stone mine (face of entry, 140 feet from mine mouth, 6-foot bed, 4-foot 4-inch cut).	227	*14495	B	1	23.11	33.47	30.61	12.81	1.08	......	......	......	......	11.1	4,305	7,749
				2	......	43.53	39.81	16.66	1.40	......	......	......	......	......	5,599	10,078
				3	......	52.23	47.77	......	1.68	......	......	......	......	......	6,718	12,092

FLATHEAD COUNTY.																
Flathead River, North Fork of; sec. 33, T. 34 N., R. 20 W., outcrop (surface exposure, 6-foot 9-inch bed, 3-foot 6-inch cut).	227	*12786	B	1	22.16	32.52	36.44	8.88	2.27	......	......	......	......	10.0	4,890	8,801
				2	......	41.78	46.81	11.41	2.92	......	......	......	......	......	6,282	11,307
				3	......	47.16	52.84	......	3.30	......	......	......	......	......	7,091	12,763
MISSOULA COUNTY.																
Missoula, 2 miles north of; sec. 4, T. 13 N., R. 19 W., Hell Gate mine (run of mine, from bunker at mouth of mine, represents coal from east entry; 6-foot bed).	228	*13541	B	1	24.70	29.33	26.11	19.86	.85	5.56	39.04	0.74	33.95	16.3	3,737	6,727
				2	......	38.95	34.68	26.37	1.12	3.74	51.85	0.98	15.93	......	4,963	8,933
				3	......	52.90	47.10	......	1.53	5.08	70.42	1.33	21.64	......	6,740	12,132
Same (in east entry off slope entry, 600 feet north and 300 feet east of mine mouth, 6-foot 10-inch bed).	228	*13542	B	1	25.21	29.19	26.01	19.59	.76	......	......	......	......	16.7	......	......
				2	......	39.08	34.78	26.19	1.02	......	......	......	......	......	......	......
				3	......	52.88	47.12	......	1.38	......	......	......	......	......	......	......
MUSSELSHELL COUNTY.																
Painted Rock, 1 mile northwest of; sec. 14, T. 5 N., R. 22 E.; Sholz mine (face of north main entry, 110 feet from mouth of entry, 2-foot 11-inch cut).	228	15114	B	1	11.59	31.82	40.35	16.24	1.00	......	......	......	......	4.5	5,563	10,013
				2	......	35.99	45.64	18.37	1.13	......	......	......	......	......	6,292	11,326
				3	......	44.09	55.91	......	1.38	......	......	......	......	......	7,708	13,874
ROSEBUD COUNTY.																
Bighorn, 6½ miles east of; sec. 2, T. 4 N., R. 35 E., prospect (10 feet from outcrop, 3-foot 2½-inch bed).	228	*14755	B	1	19.79	30.67	35.19	14.35	1.50	......	......	......	......	8.5	4,766	8,579
				2	......	38.24	43.87	17.89	1.87	......	......	......	......	......	5,942	10,696
				3	......	46.57	53.43	......	2.28	......	......	......	......	......	7,237	13,027
TETON COUNTY.																
Browning, 22 miles northwest of; SE. ¼ NE. ¼ sec. 35, T. 36 N., R. 12 W., J. M. Stone prospect (face of entry, 50 feet from mouth of entry).	229	12602	B	1	8.00	35.37	44.18	12.45	1.00	5.17	62.92	1.46	17.00	2.3	6,137	11,047
				2	......	38.45	48.02	13.53	1.09	4.65	68.39	1.59	10.75	......	6,671	12,008
				3	......	44.47	55.53	......	1.26	5.38	79.09	1.84	12.43	......	7,715	13,887
Cut Bank, 5 miles northwest of; NB. ¼ sec. 29, T. 34 N., R. 6 W.; Allison prospect (in entry, 22-inch bed).	229	12494	B	1	7.82	30.21	34.24	27.73	1.78	......	......	......	......	2.9	4,837	8,707
				2	......	32.77	37.15	30.08	1.93	......	......	......	......	......	5,247	9,445
				3	......	46.87	53.13	......	2.76	......	......	......	......	......	7,504	13,507
4 miles southwest of; NE. ¼ SW. ¼ sec 26, T. 33 N., R. 6 W.; prospect (in fresh open cut, 11-inch cut).	229	*14109	B	1	16.57	29.21	36.25	17.97	.59	......	......	......	......	6.2	......	......
				2	......	35.01	43.45	21.54	.71	......	......	......	......	......	......	......
				3	......	44.62	55.38	......	.90	......	......	......	......	......	......	......
Valier, 8 miles northwest of, on Birch Creek; SW. ¼ NW. ¼ sec. 32, T. 31 N., R. 5 W.; Blair mine (face of new entry 150 feet in, 2-foot 3-inch cut).	229	12427	B	1	6.56	40.36	39.19	13.89	3.08	5.16	60.71	1.13	16.03	1.3	6,070	10,926
				2	......	43.19	41.94	14.87	3.30	4.74	64.97	1.21	10.91	......	6,496	11,693
				3	......	50.74	49.26	......	3.88	5.57	76.32	1.42	12.81	......	7,631	13,736
VALLEY COUNTY.																
Antelope, 3 miles west of; sec. 21, T. 34 N., R. 55 E.; Richardson mine, "Richardson" bed (in new entry, 200 feet north of mouth of entry, 8-foot 3-inch cut).	230	*14614	B	1	29.14	29.65	33.82	7.39	.31	......	......	......	......	5.4	4,167	7,501
				2	......	41.84	47.73	10.43	.44	......	......	......	......	......	5,880	10,584
				3	......	46.71	53.29	......	.49	......	......	......	......	......	6,564	11,815
Bainville, 8 miles northeast of; sec. 10, T. 28 N., R. 59 E.; Red Bank mine, Red Bank bed (full section taken at face).	230	*10898	B	1	42.81	25.72	26.83	4.64	.24	7.21	36.21	.62	51.08	35.3	3,392	6,105
				2	......	44.97	46.92	8.11	.42	4.28	63.31	1.08	22.80	......	5,931	10,675
				3	......	48.94	51.06	......	.46	4.66	68.90	1.18	24.80	......	6,155	11,618

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
MONTANA—Continued.																
VALLEY COUNTY—continued.																
Culbertson, 3 miles north of; sec. 8, T. 28 N., R. 56 E.; Bruegger mine, E bed (75 feet north of mine mouth).	231	*10724	B	1	32.62	27.43	30.85	9.10	1.28	6.46	40.35	0.70	42.11	23.6	3,729	6,712
				2	....	40.71	45.79	13.50	1.90	4.21	59.88	1.04	19.47	....	5,534	9,961
				3	....	47.06	52.94	....	2.20	4.86	69.23	1.20	22.51	....	6,398	11,516
3¾ miles southeast of; sec. 3, T. 27 N., R. 56 E.; Butterfield mine, DD bed (150 feet from mine mouth, 4½-foot cut).	231	*10727	B	1	37.51	26.82	29.61	6.06	.32	6.75	39.88	.65	46.34	26.0	3,655	6,580
				2	....	42.92	47.38	9.70	.51	4.13	63.82	1.04	20.80	....	5,849	10,529
				3	....	47.53	52.47	....	.56	4.57	70.67	1.15	23.05	....	6,477	11,660
3 miles southeast of; sec. 34, T. 28 N., R. 56 E.; Dempsey mine, E bed (face, about 60 feet northeast of mine mouth).	231	*10725	B	1	38.88	27.21	27.68	6.23	.75	6.89	36.59	.74	48.80	29.2	3,341	6,014
				2	....	44.52	45.29	10.19	1.23	4.20	59.86	1.21	23.31	....	5,466	9,839
				3	....	49.57	50.43	....	1.37	4.68	66.65	1.35	25.95	....	6,086	10,955
3 miles southeast of; sec. 34, T. 28 N., R. 56 E., ½ mile from sample 10725; prospect, F bed (face, 50 feet from opening).	232	*10726	B	1	35.33	31.80	25.14	7.73	1.13	6.28	34.98	.83	49.05	26.3	3,002	5,404
				2	....	49.17	38.88	11.95	1.75	3.63	54.09	1.28	27.30	....	4,642	8,356
				3	....	55.84	44.16	....	1.99	4.12	61.43	1.45	31.01	....	5,272	9,490
Froid, 5½ miles southwest of; sec. 15, T. 29 N., R. 55 E.; Astrope mine (in entry, 115 feet north of mouth of entry).	232	*11005	B	1	40.74	25.20	27.83	6.23	.72	7.11	36.57	.68	48.69	35.0	3,415	6,147
				2	....	42.53	46.96	10.51	1.22	4.35	61.71	1.15	21.06	....	5,763	10,375
				3	....	47.52	52.48	....	1.36	4.86	68.95	1.29	23.54	....	6,440	11,592
3½ miles south of; sec. 11, T. 29 N., R. 55 E.; prospect, F bed (from a well which penetrated bed).	232	*11004	B	1	37.73	25.83	30.93	5.51	.75	7.01	39.66	.71	46.36	31.5	3,722	6,700
				2	....	41.48	49.67	8.85	1.20	4.53	63.69	1.14	20.59	....	5,977	10,759
				3	....	45.51	54.49	....	1.32	4.97	69.87	1.25	22.59	....	6,558	11,804
Medicine Lake, 15 miles east of; sec. 4, T. 31 N., R. 59 E.; Belgon mine (face of pit).	232	*10899	B	1	41.29	25.66	28.22	4.83	.36	7.12	37.18	.65	49.86	33.9	3,478	6,260
				2	....	43.71	48.06	8.23	.61	4.31	63.33	1.11	22.41	....	5,924	10,663
				3	....	47.63	52.37	....	.66	4.70	69.01	1.21	24.42	....	6,455	11,619
22 miles northeast of; sec. 8, T. 34 N., R. 58 E.; Coal Ridge mine, Coal Ridge bed (face of entry, about 400 feet from mine mouth).	233	*11002	B	1	40.41	24.50	27.58	7.51	.39	6.94	35.15	.57	49.44	35.2	3,239	5,830
				2	....	41.11	46.29	12.60	.65	4.11	58.99	.96	22.69	....	5,435	9,783
				3	....	47.04	52.96	....	.74	4.70	67.50	1.10	25.96	....	6,219	11,194
6 miles northeast of; sec. 30, T. 33 N., R. 56 E.; Jones mine, Jones bed (in entry, 250 feet south of entry mouth).	233	*11006	B	1	41.63	26.66	26.14	5.57	.40	6.81	33.48	.66	53.08	35.6	2,932	5,278
				2	....	45.67	44.79	9.54	.69	3.73	57.36	1.13	27.55	....	5,023	9,041
				3	....	50.49	49.51	....	.76	4.12	63.41	1.25	30.46	....	5,553	9,995
2½ miles southeast of; sec. 6, T. 31 N., R. 56 E.; Ted Young mine, Young bed (in mine, 300 feet southeast of mine mouth).	233	*11003	B	1	39.31	24.15	30.80	5.74	.53	6.92	37.80	.65	48.36	33.2	3,488	6,278
				2	....	39.79	50.75	9.46	.87	4.20	62.28	1.07	22.12	....	5,747	10,345
				3	....	43.95	56.05	....	.96	4.64	68.79	1.18	24.43	....	6,348	11,426

Mondak, 7 miles northwest of, 1½ miles from the Great Northern Railway; sec. 18, T. 27 N., R. 59 E.; open pit.	234	*10900	B	1	41.63	27.11	27.18	4.08	.37	6.83	36.04	.77	51.91	33.3	3,185	5,733
				2	....	46.44	46.57	6.99	.63	3.77	61.74	1.32	25.55	....	5,457	9,823
				3	....	49.93	50.07	....	.68	4.05	66.38	1.42	27.47	....	5,867	10,562
Plentywood, 2½ miles northeast of; sec. 10, T. 35 N., R. 55 E.; Pierce mine, "Richardson" bed (in mine, about 250 feet northwest of mine mouth, 8-foot bed, 6-foot 6-inch cut).	234	*14670	B	1	34.29	20.56	28.06	8.09	.66	....	....	....	....	5.9	3,756	6,761
				2	....	44.98	42.71	12.31	1.00	....	....	....	....	....	5,716	10,289
				3	....	51.30	48.70	....	1.14	....	....	....	....	....	6,519	11,734
NEVADA.																
ESMERALDA COUNTY.																
Coaldale, 3½ miles southeast of; sec. 33, T. 2 N., R. 37 E.; Darms mine, C bed (in drift, off slope, 300 feet from mouth of slope and 20 feet from slope, 7-foot bed and cut).	234	13978	B	1	2.08	33.67	33.74	30.51	7.18	....	....	....	....	.6	5,329	9,592
				2	....	34.38	34.46	31.16	7.33	....	....	....	....	....	5,442	9,796
				3	....	49.94	50.06	....	10.65	....	....	....	....	....	7,905	14,229
about 3½ miles southeast of; sec. 29, T. 2 N., R. 37 E.; Nevada mine, C bed (150 feet from mouth of entry, 9-foot bed and cut.)	235	13979	B	1	1.89	26.74	27.42	43.95	.60	....	....	....	....	.7	4,499	8,098
				2	....	27.26	27.94	44.80	.61	....	....	....	....	....	4,586	8,255
				3	....	49.38	50.62	....	1.11	....	....	....	....	....	8,308	14,954
Same (about 100 feet from mouth of entry, 8-foot bed and cut).	235	13980	B	1	1.65	31.31	35.48	31.56	7.42	....	....	....	....	.2	5,331	9,596
				2	....	31.84	36.07	32.09	7.54	....	....	....	....	....	5,421	9,758
				3	....	46.88	53.12	....	11.10	....	....	....	....	....	7,982	14,368
Same (sample cut from block of best coal, 12-inch cut).	235	14409	B	1	1.61	39.18	43.79	15.42	6.64	....	....	....	....	.1	6,766	12,179
				2	....	39.82	44.51	15.67	6.75	....	....	....	....	....	6,877	12,379
				3	....	47.22	52.78	....	8.00	....	....	....	....	....	8,155	14,679
NEW MEXICO.																
BERNALILLO COUNTY.																
Albuquerque, about 25 miles east of; sec. 6, T. 10 N., R. 6 E.; Holmes mine (at face in entry, 300 feet from entry mouth, 2-foot 3-inch cut).	235	12650	B	1	1.61	31.11	36.14	31.14	3.24	....	....	....	....	.5	5,581	10,046
				2	....	31.62	36.73	31.65	3.29	....	....	....	....	....	5,673	10,211
				3	....	46.26	53.74	....	4.81	....	....	....	....	....	8,300	14,940
COLFAX COUNTY.																
Dawson; secs. 1, 2, 11, and 12, T. 29 N., and secs. 35 and 36, T. 28 N., R. 20 E.; Dawson No. 2 mine, Raton bed (face of north entry 5; 6,700 feet northeast of "high line" entrance).	236	12233	A	1	2.95	35.08	49.15	12.82	.92	....	....	....	....	1.4	....	....
				2	....	36.15	50.64	13.21	.95	....	....	....	....	....	....	....
				3	....	41.65	58.35	....	1.09	....	....	....	....	....	....	....
Same (face of east entry 13; 6,500 feet from "high line" entrance).	236	12234	A	1	3.75	36.47	48.12	11.66	.59	....	....	....	....	2.0	....	....
				2	....	37.89	50.00	12.11	.61	....	....	....	....	....	....	....
				3	....	43.11	56.89	....	.69	....	....	....	....	....	....	....
Same (composite of samples 12233 and 12234) ...	236	12235	A	1	3.32	35.68	48.66	12.34	.71	5.31	70.35	1.19	10.10	1.7	7,087	12,757
				2	....	36.90	50.34	12.76	.73	5.11	72.76	1.23	7.41	....	7,330	13,194
				3	....	42.30	57.70	....	.84	5.86	83.40	1.41	8.49	....	8,402	15,124
Gardiner, ½ mile southeast of Blossburg; Gardiner No. 1 mine Raton bed (face of room 6, west entry 1, off main entry, 400 feet from outcrop; 6-foot bed, 5-foot 5-inch cut).	236	14795	B	1	2.34	36.70	45.73	15.23	.64	5.13	67.91	1.30	9.79	.9	6,809	12,256
				2	....	37.58	46.82	15.60	.66	4.99	69.54	1.33	7.88	....	6,972	12,550
				3	....	44.52	55.48	....	.78	5.91	82.39	1.58	9.34	....	8,260	14,868

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
NEW MEXICO—Continued.																
COLFAX COUNTY—continued.																
Koehler; T. 29 N., R. 22 E.; Koehler No. 1 mine, Raton bed (face of room 20, off east entry 7; 4,500 feet southeast of mouth of main entry, 9-foot bed, 6-foot cut).	237	12335	A	1	4.96	36.32	48.98	9.74	0.62	......	......	......	......	3.4	......	......
				2	......	38.22	51.53	10.25	.65	......	......	......	......	......	......	......
				3	......	42.58	57.42	......	.72	......	......	......	......	......	......	......
Same (face of west entry 8; 4,300 feet southeast of mouth of main entry, 8-foot 2-inch bed, 6-foot 2-inch cut).	237	12336	A	1	3.08	36.60	48.35	11.97	.78	......	......	......	......	1.3	......	......
				2	......	37.76	49.89	12.35	.80	......	......	......	......	......	......	......
				3	......	43.08	56.92	......	.91	......	......	......	......	......	......	......
Same (face of room 20, off east entry 7; 4,200 feet southeast of mouth of main entry).	237	12337	A	1	2.87	35.27	47.20	14.66	.48	......	......	......	......	1.0	......	......
				2	......	36.31	48.60	15.09	.49	......	......	......	......	......	......	......
				3	......	42.76	57.24	......	.58	......	......	......	......	......	......	......
Same (composite of samples 12335, 12336, and 12337).	237	12338	A	1	3.64	36.10	48.26	12.00	.70	5.25	69.76	1.31	10.98	1.9	7,013	12,623
				2	......	37.46	50.09	12.45	.73	5.03	72.40	1.36	8.03	......	7,278	13,100
				3	......	42.79	57.21	......	.83	5.75	82.70	1.55	9.17	......	8,313	14,963
Same (end of west entry 9; 4,900 feet southwest of mouth of main entry; 6-foot 3-inch bed).	237	14796	B	1	2.34	35.87	49.94	11.85	.63	5.17	71.27	1.34	9.74	1.0	7,178	12,920
				2	......	36.73	51.14	12.13	.65	5.03	72.98	1.37	7.84	......	7,350	13,230
				3	......	41.80	58.20	......	.74	5.72	83.05	1.56	8.93	......	8,364	15,055
Sugarite, 7 miles northeast of Raton; NE. ¼ NE. ¼ sec. 9, T. 31 N., R. 24 E.; Sugarite No. 1 mine; Sugarite bed (in entry 1,500 feet southwest of mine mouth, 5-foot 6¼-inch bed, 5-foot 1-inch cut).	238	14791	B	1	3.83	38.12	49.12	8.93	.56	5.71	72.37	1.58	10.85	2.1	7,263	13,073
				2	......	39.64	51.07	9.29	.58	5.49	75.25	1.64	7.75	......	7,552	13,594
				3	......	43.70	56.30	......	.64	6.05	82.96	1.81	8.54	......	8,325	14,985
7 miles northeast of; NE. ¼ NW. ¼ sec. 3, T. 31 N., R. 24 E.; Sugarite No. 2 mine; Sugarite bed (in entry 1,200 feet northeast of mine mouth, 5-foot 7-inch bed, 4-foot 9-inch cut).	238	14792	B	1	2.91	39.72	48.67	8.70	.51	5.62	73.52	1.57	10.08	1.0	7,380	13,284
				2	......	40.91	50.13	8.96	.53	5.46	75.73	1.62	7.70	......	7,601	13,682
				3	......	44.94	55.06	......	.58	6.00	83.18	1.78	8.46	......	8,349	15,028
Yankee; Yankee No. 3 mine, Yankee bed (face of room 2, off west entry 10, 5-foot 9½-inch bed, 4-foot 7-inch cut.)	238	13365	A	1	5.21	36.99	48.25	9.55	.56	......	......	......	......	3.4	......	......
				2	......	39.02	50.90	10.08	.59	......	......	......	......	......	......	......
				3	......	43.39	56.61	......	.66	......	......	......	......	......	......	......
Same (face of south entry 5, 6-foot 3½ inch bed, 4-foot 7-inch cut).	238	13366	A	1	6.05	35.23	42.10	16.62	.53	......	......	......	......	4.4	......	......
				2	......	37.50	44.81	17.69	.56	......	......	......	......	......	......	......
				3	......	45.56	54.44	......	.68	......	......	......	......	......	......	......

Same (composite of samples 13365 and 13366)......	239	13367	A	1	5.58	35.78	45.47	13.17	.55	5.31	66.76	1.18	13.03	3.9	6,677	12,019
				2	......	37.89	48.16	13.95	.58	4.97	70.71	1.25	8.54	......	7,072	12,730
				3	......	44.03	55.97	......	.67	5.78	82.17	1.45	9.93	......	8,218	14,792
LINCOLN COUNTY.																
White Oaks, 2 miles southeast of; SW. ¼ sec. 32, T. 6 S., R. 13 E., Wildcat mine (at foot of main slope, 25-inch bed, 21½-inch cut).	239	15054	B	1	2.52	34.63	45.99	16.86	.76	4.97	66.65	1.32	9.44	1.3	6,642	11,956
				2	......	35.53	47.17	17.30	.78	4.81	68.38	1.35	7.38	......	6,814	12,265
				3	......	42.96	57.04	......	.94	5.82	82.69	1.63	8.92	......	8,239	14,830
2½ miles southeast of; NW. ¼ sec. 5, T. 7 S., R. 13 E.; Old Abe mine, Old Abe bed (4-foot ½-inch bed and cut).	239	15053	B	1	2.49	37.86	44.95	14.70	.83	5.15	68.06	1.23	10.03	.9	6,813	12,263
				2	......	38.83	46.10	15.07	.85	4.99	69.80	1.26	8.03	......	6,987	12,577
				3	......	45.72	54.28	......	1.00	5.88	82.18	1.48	9.46	......	8,226	14,807
M'KINLEY COUNTY.																
Blackrock, 10 miles northeast of; sec. 6, T. 11 N., R. 17 W.; Zuni Indian School mine (room 2, west of the main entry, 3-foot 2-inch bed, 2-foot 10-inch cut).	240	15032	B	1	14.69	34.93	41.56	8.82	.79	5.82	60.93	1.12	22.52	11.0	6,005	10,809
				2	......	40.94	48.72	10.34	.93	4.91	71.42	1.31	11.09	......	7,039	12,670
				3	......	45.66	54.34	......	1.04	5.48	79.65	1.46	12.37	......	7,851	14,132
SANTA FE COUNTY.																
Madrid, Anthracite No. 4 mine, White Ash bed (at face, 1,500 feet southeast of mouth of main entry, 2-foot 9-inch bed and cut).	240	14886	B	1	7.55	7.25	75.88	9.32	.76	2.10	78.97	1.00	7.85	2.2	6,723	12,101
				2	......	7.84	82.08	10.08	.82	1.36	85.42	1.08	1.24	......	7,272	13,090
				3	......	8.72	91.28	......	.91	1.51	95.00	1.20	1.38	......	8,087	14,557
Blacksmith mine, Peacock (?) bed (at face, 310 feet south of mine mouth, 5-foot 3-inch bed, 2-foot 4-inch cut).	241	14884	B	1	4.62	26.44	61.34	7.60	1.27	5.22	75.94	1.38	8.59	3.8	7,519	13,534
				2	......	27.72	64.31	7.97	1.33	4.94	79.62	1.45	4.69	......	7,883	14,189
				3	......	30.12	69.88	......	1.45	5.37	86.52	1.58	5.08	......	8,566	15,419
Holen mine, Cook & White bed (at face, 1,300 feet northeast of mine mouth, 3-foot 5-inch bed, 3-foot 3-inch cut).	241	14885	B	1	3.65	35.05	49.47	11.83	.96	5.04	69.89	1.29	10.99	1.6	6,863	12,353
				2	......	36.38	51.34	12.28	1.00	4.81	72.54	1.34	8.03	......	7,123	12,821
				3	......	41.47	58.53	......	1.14	5.48	82.70	1.53	9.15	......	8,120	14,616
Peacock prospect, Peacock bed (at face, 400 feet northeast of entry mouth, 2-foot 4-inch bed and cut).	242	14887	B	1	3.16	41.37	44.46	11.01	1.59	5.28	69.71	1.24	11.17	.9	6,918	12,452
				2	......	42.72	45.91	11.37	1.64	5.09	71.98	1.28	8.64	......	7,144	12,859
				3	......	48.20	51.80	......	1.85	5.74	81.22	1.44	9.75	......	8,061	14,510
NORTH DAKOTA.																
ADAMS COUNTY.																
Haynes, 3½ miles northeast of; sec. 16, T. 129 N., R. 94 W.; Nipper & Monroe mine, Haynes bed (in mine, 634 feet east of mine mouth, 8-foot 3-inch cut).	242	*14542	B	1	32.65	30.57	28.49	8.29	1.53	......	......	......	......	14.5	4,087	7,357
				2	......	45.39	42.30	12.31	2.27	......	......	......	......	......	6,068	10,922
				3	......	51.76	48.24	......	2.59	......	......	......	......	......	6,920	12,456
9 miles northeast of; sec. 36, T. 130 N., R. 94 W.; William Pinkham mine, Haynes bed (225 feet south of entry mouth, 12-foot bed, 8-foot 10-inch cut).	242	*14544	B	1	32.43	30.89	28.11	8.57	1.49	......	......	......	......	15.3	4,071	7,328
				2	......	45.71	41.61	12.68	2.21	......	......	......	......	......	6,025	10,845
				3	......	52.35	47.65	......	2.53	......	......	......	......	......	6,900	12,420
BOWMAN COUNTY.																
Amor, 2 miles southeast of; sec. 34, T. 130 N., R. 103 W.; Durkin prospect, T Cross bed (face of pit, fresh cut, 8-foot 7½-inch bed, 2-foot 7½-inch cut).	242	*14857	B	1	43.63	23.24	26.95	6.18	.65	......	......	......	......	35.8	3,238	5,828
				2	......	41.23	47.81	10.96	1.15	......	......	......	......	......	5,744	10,339
				3	......	46.31	53.69	......	1.29	......	......	......	......	......	6,451	11,612
Scranton; sec. 24, T. 131 N., R. 100 W.; Scranton mine, Harmon (?) bed (face of main east entry off main entry; 500 feet from the main entry and 1,000 feet from portal, 20-foot 3½-inch bed, 6-foot cut).	243	*14485	B	1	34.80	31.09	25.98	8.13	.66	......	......	......	......	22.5	3,842	6,916
				2	......	47.68	39.85	12.47	1.01	......	......	......	......	......	5,892	10,606
				3	......	54.47	45.53	......	1.15	......	......	......	......	......	6,732	12,118

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
NORTH DAKOTA—Continued.																
MORTON COUNTY.																
Leith, 1½ miles west of; sec. 12, T. 133 N., R. 88 W.; Jones mine (in mine, about 75 feet southwest of the shaft, 5-foot 6-inch cut).	244	*14729	B	1	36.18	29.77	25.35	8.70	0.68	6.76	39.45	0.59	43.82	27.1	3,722	6,700
				2	....	46.65	39.72	13.63	1.07	4.29	61.81	0.92	18.28	....	5,832	10,498
				3	....	54.01	45.99	....	1.24	4.97	71.56	1.07	21.16	....	6,752	12,154
WARD COUNTY.																
Plaza, 25 miles west of; sec. 17, T. 152 N., R. 92 W.; prospect No. 3 bed (4-foot 11-inch bed, 3-foot 3-inch cut).	244	*11294	B	1	41.00	24.88	27.54	6.58	.44	....	....	....	....	33.7	3,266	5,879
				2	....	42.17	46.68	11.15	.75	....	....	....	....	....	5,536	9,965
				3	....	47.46	52.54	....	.84	....	....	....	....	....	6,231	11,216
WILLIAMS COUNTY.																
Avoca, ½ mile northeast of, 5 miles northeast of Williston; NW. ¼ sec. 3, T. 154 N., R. 100 W.; Bruegger mine, C bed (50 feet west of end of main entry and 1,225 feet northwest of mine mouth, 11-foot bed, 5-foot 6-inch cut).	244	*12587	B	1	41.53	26.97	27.18	4.32	.30	....	....	....	....	33.2	3,578	6,440
				2	....	46.13	46.48	7.39	.51	....	....	....	....	....	6,119	11,014
				3	....	49.81	50.19	....	.55	....	....	....	....	....	6,607	11,893
Trenton, 6 miles northeast of; sec. 27, T. 154 N., R. 102 W.; Geltz mine (at end of main entry, about 150 feet from mine mouth, 5-foot 6-inch bed, 5-foot 1-inch cut).	245	* 12411	B	1	42.71	25.70	24.77	6.82	1.38	....	....	....	....	34.7	3,368	6,062
				2	....	44.86	43.24	11.90	2.41	....	....	....	....	....	5,879	10,582
				3	....	50.92	49.08	....	2.74	....	....	....	....	....	6,673	12,011
Williston, 3½ miles southeast of; sec. 33, T. 154 N., R. 100 W.; Powell mine, C bed (50 feet north of main entry, about 900 feet northeast of mine mouth, 10-foot bed, 6-foot 6-inch cut).	245	* 12588	B	1	42.91	26.81	24.98	5.30	.71	....	....	....	....	34.7	3,462	6,232
				2	....	46.96	43.76	9.28	1.24	....	....	....	....	....	6,064	10,915
				3	....	51.76	48.24	....	1.37	....	....	....	....	....	6,684	12,031
2 miles north and 1 mile east of; sec. 7, T. 154 N., R. 100 W.; B bed, U. S. Reclamation Service mine (William Spangler's room, 50 feet north of main entry and 1,225 feet east of portal, 10-foot 3-inch bed, 8-foot cut).	245	* 12533	B	1	43.87	24.88	25.43	5.82	.49	....	....	....	....	33.2	3,299	5,938
				2	....	44.33	45.30	10.37	.87	....	....	....	....	....	5,877	10,579
				3	....	49.46	50.54	....	.97	....	....	....	....	....	6,557	11,808

OHIO.

JEFFERSON COUNTY.

Amsterdam, about 1 mile south of; sec. 18, T. 11 N., R. 4 W., Amsterdam mine, Lower Freeport (?), locally known as No. 7 bed (center of face of face entry, 5-foot 2-inch bed, 4-foot 1-inch cut).	246	10533	A	1	3.36	37.56	52.06	7.02	2.40					1.4		
				2		38.87	53.87	7.26	2.48							
				3		41.91	58.09		2.67							
Same (room 4 off entry 4, east section, 1,000 feet northeast of shaft, 5-foot 7½-inch bed, 3-foot 9-inch cut).	246	15388	B	1	3.66	37.43	51.23	7.68	3.07					1.6	7,342	13,216
				2		38.85	53.18	7.97	3.19						7,621	13,718
				3		42.21	57.79		3.47						8,281	14,906
Same (room 15, west entry 9, section 2 north, 4,000 feet northwest of shaft; 5-foot 2½-inch bed, 3-foot 4-inch cut).	246	15389	B	1	3.52	38.31	50.78	7.39	2.99					1.4	7,363	13,253
				2		39.71	52.63	7.66	3.10						7,632	13,738
				3		43.01	56.99		3.36						8,265	14,877
Same (west entry 8, section 3 south, 4,500 feet southwest of shaft; 5-foot 3½-inch bed, 3-foot 5½-inch cut).	246	15390	B	1	3.27	38.87	50.37	7.49	3.23					1.2	7,359	13,246
				2		40.18	52.08	7.74	3.34						7,608	13,694
				3		43.55	56.45		3.63						8,246	14,843
Same (composite of samples 15388, 15389, and 15390).	246	15394	B	1	3.50	37.98	51.08	7.44	3.09	5.43	73.39	1.46	9.19	1.4	7,381	13,286
				2		39.36	52.93	7.71	3.20	5.22	76.05	1.51	6.51		7,649	13,768
				3		42.65	57.35		3.47	5.66	82.40	1.64	6.83		8,288	14,918
Halfway between Bergholz and; sec. 8, T. 12 N., R. 4 W.; Eastern Ohio mine, Lower Freeport (?) bed (room 11, off west butt entry 12, 3,000 feet west of mouth of slope, 5-foot 4-inch bed, 4-foot 8-inch cut).	247	15391	B	1	3.58	37.58	49.17	9.67	3.08					1.5	7,156	12,881
				2		38.97	51.00	10.03	3.19						7,421	13,358
				3		43.32	56.68		3.55						8,248	14,846
Same (face of west entry 13, off south face entry, 4-foot 9-inch cut).	247	15392	B	1	3.47	38.61	49.66	8.26	2.40					1.5	7,346	13,223
				2		40.00	51.44	8.56	2.49						7,610	13,698
				3		43.74	56.26		2.72						8,322	14,980
Same (left butt entry 12, off north face entry, 2,700 feet northeast of mouth of slope, 5-foot 4½-inch bed, 4-foot 11-inch cut).	247	15393	B	1	4.04	38.61	48.93	8.42	2.95					2.0	7,234	13,021
				2		40.24	50.99	8.77	3.07						7,539	13,570
				3		44.11	55.89		3.37						8,263	14,873
Same (composite of samples 15391, 15392, and 15393).	247	15395	B	1	3.69	37.97	49.59	8.75	2.82	5.32	71.76	1.39	9.96	1.7	7,283	13,109
				2		39.42	51.49	9.09	2.93	5.10	74.51	1.44	6.93		7,562	13,612
				3		43.36	56.64		3.22	5.61	81.96	1.58	7.63		8,318	14,972
2 miles east of; sec. 7, T. 12 N., R. 4 W.; Elizabeth mine, Lower Freeport (?) bed (room 22, off west entry 3, 2,600 feet northwest of shaft; 4-foot 5½-inch bed, 3-foot 9-inch cut).	247	15440	B	1	3.40	38.01	52.37	6.22	2.20					1.7	7,518	13,532
				2		39.35	54.21	6.44	2.28						7,783	14,009
				3		42.06	57.94		2.44						8,318	14,972
Same (room 33, off second south face entry, 2,400 feet west of shaft, 5-foot 5½-inch bed, 4-foot 1-inch cut).	247	15441	B	1	3.50	37.93	50.63	7.94	3.14					1.4	7,364	13,255
				2		39.31	52.46	8.23	3.25						7,631	13,736
				3		42.84	57.16		3.54						8,316	14,969
Same (composite of samples 15440 and 15441).....	247	15442	B	1	3.38	38.01	51.64	6.97	2.71	5.40	73.31	1.46	10.15	1.6	7,438	13,388
				2		39.34	53.45	7.21	2.80	5.20	75.88	1.51	7.40		7,698	13,856
				3		42.40	57.60		3.02	5.60	81.78	1.63	7.97		8,296	14,933
Same, Lower Freeport (?) bed (room 6, off west entry 9, 2,600 feet north of shaft, 4-foot 7½-inch bed, 3-foot 10½-inch cut).	247	15443	B	1	3.59	37.03	50.64	8.74	3.70	5.19	71.30	1.39	9.68	1.8	7,224	13,003
				2		38.41	52.52	9.07	3.84	4.97	73.95	1.44	6.73		7,493	13,487
				3		42.24	57.76		4.22	5.47	81.32	1.58	7.41		8,240	14,832

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
OHIO—Continued.																
JEFFERSON COUNTY—continued.																
Brilliant, 1½ miles northwest of; sec. 6, T. 5 N., R. 2 W.; Goucher No. 2 mine, Pittsburgh or No. 8 bed (face of entry 2, north side of mine, 500 feet northwest of opening, 4-foot 9½-inch bed, 4-foot 5½-inch cut).	248	15563	B	1	3.79	38.78	50.03	7.40	3.84	......	......	......	......	2.0	7,267	13,081
				2	......	40.31	52.00	7.69	3.99	......	......	......	......	......	7,553	13,595
				3	......	43.67	56.33	......	4.32	......	......	......	......	......	8,182	14,728
Same (room 1 off right entry 1, 500 feet northwest of opening; 5-foot 3½-inch bed, 4-foot 8-inch cut).	248	15564	B	1	4.28	37.41	50.29	8.02	3.72	......	......	......	......	2.5	7,201	12,962
				2	......	39.08	52.54	8.38	3.89	......	......	......	......	......	7,523	13,541
				3	......	42.66	57.34	......	4.25	......	......	......	......	......	8,211	14,780
Same (composite of samples 15563 and 15564).....	248	15565	B	1	4.11	37.96	50.23	7.70	3.84	5.23	71.69	1.39	10.15	2.3	7,230	13,014
				2	......	39.59	52.38	8.03	4.00	4.97	74.77	1.45	6.78	......	7,540	13,572
				3	......	43.05	56.95	......	4.35	5.40	81.30	1.58	7.37	......	8,129	14,756
Hopedale, 2 miles southeast of; sec. 32, T. 9 N., R. 3 W.; Parlett mine, Pittsburgh or No. 8 bed (room 9, off west entry 2 off south entry, 2,200 feet northeast of the mouth of the slope; 5-foot 11½-bed, 4-foot 6-inch cut).	248	15444	B	1	5.01	36.15	52.83	6.01	1.74	......	......	......	......	2.2	7,327	13,189
				2	......	38.06	55.61	6.33	1.83	......	......	......	......	......	7,713	13,883
				3	......	40.63	59.37	......	1.95	......	......	......	......	......	8,234	14,821
Same (in north entry 2, 180 feet beyond rooms 9 and 10, off west entry, 400 feet northwest of the mouth of the slope; 7-foot 11-inch bed, 4-foot 5-inch cut).	248	15445	B	1	4.32	37.09	49.14	9.45	3.82	......	......	......	......	2.6	7,072	12,730
				2	......	38.76	51.36	9.88	3.99	......	......	......	......	......	7,391	13,304
				3	......	43.01	56.99	......	4.43	......	......	......	......	......	8,201	14,762
Same (composite of samples 15444 and 15445).....	248	15446	B	1	4.66	36.63	50.82	7.89	2.80	5.28	71.72	1.35	10.96	2.4	7,193	12,947
				2	......	38.42	53.30	8.28	2.94	4.99	75.23	1.42	7.14	......	7,545	13,581
				3	......	41.89	58.11	......	3.21	5.44	82.02	1.55	7.78	......	8,226	14,807
Piney Fork, 2 miles southwest of Smithfield; sec. 16, T. 8 N., R. 3 W.; Cabbage Run mine; Pittsburgh bed (room 26, left entry 3, 1,050 feet northeast of mine mouth, 4-foot 8-inch bed, 4-foot 1-inch cut).	249	15491	B	1	5.45	35.73	54.22	4.60	.84	......	......	......	......	3.2	7,401	13,322
				2	......	37.79	57.35	4.86	.89	......	......	......	......	......	7,827	14,089
				3	......	39.72	60.28	......	.94	......	......	......	......	......	8,227	14,809

Two miles southwest of Smithfield; sec. 21, T. 8 N., R. 3 W.; Piney Fork No. 1 mine, Pittsburgh bed (room 14 off left entry off main east entry, 4,000 feet southwest of mine mouth, 4-foot 9½-inch, bed, 4-foot 1½-inch cut).	249	15489	B	1	4.87	36.63	51.25	7.25	2.60	5.36	71.91	1.38	11.50	2.8	7,218	12,992
				2	....	38.51	53.87	7.62	2.73	5.07	75.59	1.45	7.54	....	7,588	13,658
				3	....	41.69	58.31	....	2.96	5.49	81.83	1.57	8.15	....	8,214	14,785
Two miles southwest of Smithfield; sec. 17, T. 8 N., R. 3 W.; Piney Fork No. 2 mine, Pittsburgh bed (room 28, off east entry 21, 5,600 feet northeast of mine mouth, 4-foot 5½-inch bed, 3-foot 10½-inch cut).	249	15490	B	1	5.47	35.77	53.28	5.48	.77	....	....	....	....	3.2	7,324	13,188
				2	....	37.84	56.36	5.80	.81	....	....	....	....	....	7,748	13,946
				3	....	40.17	59.83	....	.86	....	....	....	....	....	8,225	14,806
Smithfield, 2 miles east of; sec. 29, T. 5 N., R. 2 W.; Plumb Run No. 1 mine, Pittsburgh bed (room 7, east entry 7, south face entry, 2.500 feet northeast of mine mouth; 4-foot 9-inch bed, 4-foot 6-inch cut).	250	15486	B	1	4.78	35.93	53.98	5.31	.98	5.35	74.13	1.46	12.77	2.6	7,417	13,351
				2	....	37.73	56.69	5.58	1.03	5.06	77.85	1.53	8.95	....	7,789	14,020
				3	....	39.96	60.04	....	1.09	5.36	82.45	1.62	9.48	....	8,249	14,848
Two miles east of; sec. 30, T. 5 N., R. 2 W.; Plumb Run No. 4 mine, Pittsburgh or No. 8 bed (room 14, off right entry 15, 4,400 feet northeast of mine mouth, 4-foot 4½-inch bed, 4-foot 1-inch cut.)	250	15487	B	1	4.52	36.40	51.10	7.98	3.33	....	....	....	....	2.6	7,157	12,883
				2	....	38.12	53.52	8.36	3.49	....	....	....	....	....	7,496	13,493
				3	....	41.60	58.40	....	3.81	....	....	....	....	....	8,180	14,724
Two miles east of; sec. 29, T. 5 N., R 2 W; Plumb Run No. 5 mine, Pittsburgh or No. 8 bed (room 3, right entry 2, off first right face entry, 1,800 feet southeast of mine mouth, 4-foot 6½-inch bed, 4-foot 3½-inch cut).	250	15488	B	1	4.26	36.61	52.18	6.95	2.57	....	....	....	....	2.3	7,307	13,153
				2	....	38.24	54.50	7.26	2.68	....	....	....	....	....	7,632	13,738
				3	....	41.23	58.77	....	2.80	....	....	....	....	....	8,230	14,814
Steubenville, sec. 29, T. 2 N., R. 1 W.; La Belle mine, Lower Freeport bed (fifth south butt entry, off southeast entry, 4,400 feet southeast of shaft, 3-foot 4½-inch bed, 3-foot 1½-inch cut).	250	15676	B	1	3.80	35.39	53.39	7.42	1.91	....	....	....	....	1.5	7,344	13,219
				2	....	36.79	55.50	7.71	1.99	....	....	....	....	....	7,634	13,741
				3	....	39.86	60.14	....	2.16	....	....	....	....	....	8,271	14,888
Same (in room 9, off left entry, off second south butt entry, 5,200 feet southeast from shaft, 3-foot 2-inch bed, 3-foot ¾-inch cut).	250	15675	B	1	3.82	35.90	53.43	6.85	1.90	5.24	74.20	1.45	10.36	1.6	7,418	13,352
				2	....	37.33	55.55	7.12	1.98	5.01	77.15	1.51	7.23	....	7,712	13,882
				3	....	40.19	59.81	....	2.13	5.39	83.07	1.63	7.78	....	8,304	14,947
NOBLE COUNTY.																
Belle Valley, 1 mile northwest of; Noble mine, No. 7 bed (face of south entry 8 (motor road) 5,700 feet southwest of shaft bottom, 4-foot 3½-inch bed, 4-foot ½-inch cut).	251	17213	A	1	4.88	37.76	48.93	8.43	2.95	....	....	....	....	1.4	7,073	12,731
				2	....	39.70	51.44	8.86	3.10	....	....	....	....	....	7,436	13,385
				3	....	43.56	56.44	....	3.40	....	....	....	....	....	8,159	14,686
Same (face of room 5, off south entry 7, 5,600 feet southwest of shaft bottom, 5-foot 4-inch bed, 5-foot 2-inch cut).	251	17214	A	1	5.13	37.17	49.59	8.11	2.99	....	....	....	....	1.5	7,063	12,696
				2	....	39.18	52.27	8.55	3.15	....	....	....	....	....	7,435	13,383
				3	....	42.85	57.15	....	3.44	....	....	....	....	....	8,130	14,634
Same (composite of samples 17213 and 17214).....	251	17215	A	1	5.15	37.34	49.00	8.51	2.94	5.42	70.51	1.50	11.12	1.5	7,074	12,733
				2	....	39.37	51.66	8.97	3.10	5.12	74.34	1.58	6.89	....	7,459	13,426
				3	....	43.25	56.75	....	3.41	5.62	81.66	1.74	7.57	....	8,194	14,749

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
OKLAHOMA.																
HASKELL COUNTY.																
McCurtain, 1 mile west of; San Bois No. 2 mine; McCurtain (Hartshorne) bed (face of south entry 13; 5,000 feet west of slope mouth, 8-foot ⅞-inch bed, 5-foot 8¾-inch cut).	252	13688	A	1	2.44	20.77	69.84	6.95	1.15	......	......	......	......	1.7	......	......
				2	......	21.29	71.59	7.12	1.18	......	......	......	......	......	......	......
				3	......	22.92	77.08	......	1.27	......	......	......	......	......	......	......
Same (face of room 3 off north entry 13, top side; 5,000 feet west of slope mouth, 7-foot 3⅝-inch bed, 5-foot 7¼-inch cut).	252	13689	A	1	2.57	20.29	71.21	5.93	.83	......	......	......	......	1.8	......	......
				2	......	20.83	73.08	6.09	.85	......	......	......	......	......	......	......
				3	......	22.18	77.82	......	.91	......	......	......	......	......	......	......
Same (face of south entry 11, 50 feet from slope and 5,000 feet west of slope mouth, 6-foot 10-inch bed, 5-foot 6-inch cut).	252	13690	A	1	3.21	21.78	68.84	6.17	.59	......	......	......	......	2.2	......	......
				2	......	22.50	71.13	6.37	.61	......	......	......	......	......	......	......
				3	......	24.03	75.97	......	.65	......	......	......	......	......	......	......
Same (face of south entry 12, top side; 5,500 feet west of slope mouth, 8-foot 11¼-inch bed, 5-foot 8¾-inch cut).	252	13691	A	1	2.81	20.79	70.65	5.75	.56	......	......	......	......	2.0	......	......
				2	......	21.39	72.69	5.92	.58	......	......	......	......	......	......	......
				3	......	22.74	77.26	......	.62	......	......	......	......	......	......	......
Same (neck of room 6, off north entry 12, top side; 5,300 feet west of slope mouth, 6-foot 6-inch bed, 5-foot 4-inch cut).	252	13692	A	1	2.92	21.44	68.80	6.84	.76	......	......	......	......	2.0	......	......
				2	......	22.09	70.86	7.05	.78	......	......	......	......	......	......	......
				3	......	23.76	76.24	......	.84	......	......	......	......	......	......	......
Same (composite of samples 13688, 13689, 13690, 13691, and 13692).	252	13693	A	1	2.70	21.07	69.88	6.35	.77	4.46	81.33	1.67	5.42	1.9	7,832	14,098
				2	......	21.65	71.82	6.53	.79	4.28	83.58	1.72	3.10	......	8,049	14,488
				3	......	23.16	76.84	......	.85	4.58	89.42	1.84	3.31	......	8,612	15.502
PITTSBURG COUNTY.																
Adamson, 1½ miles east of; sec. 9, T. 5 N., R. 17 E.; Eclipse No. 1 mine, Lower Hartshorne bed (face of east entry 3, 600 feet northeast of slope mouth, 4-foot 6-inch bed and cut).	252	16143	A	1	5.18	35.61	55.22	3.99	1.14	......	......	......	......	3.8	......	......
				2	......	37.55	58.24	4.21	1.20	......	......	......	......	......	......	......
				3	......	39.20	60.80	......	1.25	......	......	......	......	......	......	......
Same (face of west entry 3; 600 feet northwest of slope mouth; 4-foot bed and cut).	253	16144	A	1	4.39	36.33	55.35	3.93	1.57	......	......	......	......	3.0	......	......
				2	......	38.00	57.89	4.11	1.64	......	......	......	......	......	......	......
				3	......	39.63	60.37	......	1.71	......	......	......	......	......	......	......
Same (composite of samples 16143 and 16144).....	253	16145	A	1	4.83	35.76	55.55	3.86	1.34	5.57	77.08	1.97	10.18	3.4	7.683	13,829
				2	......	37.58	58.36	4.06	1.41	5.29	81.00	2.07	6.17	......	8,073	14,531
				3	......	39.17	60.83	......	1.47	5.51	84.43	2.16	6.43	......	8,414	15,145

Alderson, 1½ miles southwest of; sec. 15, T. 5 N., R. 15 E.: Alderson No. 5 mine, McAlester bed (face of room 1, off fourth west back entry, off west slope, 2,200 feet west of shaft, 3-foot 7-inch bed and cut).	253	11915	A	1	2.81	36.37	56.35	4.47	.75					1.1		
				2		37.42	57.98	4.60	.77							
				3		39.22	60.78		.81							
Same (face of room 7, off west entry 7, off west slope, 2,800 feet northwest of shaft, 3-foot 5-inch bed and cut).	253	11916	A	1	3.17	35.28	57.39	4.16	.58					1.3		
				2		36.43	59.27	4.30	.60							
				3		38.07	61.93		.63							
Same (face of room 66, off east entry 11, off east slope, 4,000 feet east of shaft, 3-foot 4-inch bed, 2-foot 6-inch cut).	253	11917	A	1	3.25	34.55	57.59	4.61	.59					1.3		
				2		35.71	59.53	4.76	.61							
				3		37.50	62.50		.64							
Same (face of west entry 2, off east slope, about 2,000 feet southeast of shaft, 3-foot 7-inch bed and cut).	253	11918	A	1	2.97	35.78	56.82	4.43	.64					1.2		
				2		36.87	58.56	4.57	.66							
				3		38.64	61.36		.69							
Same (composite of samples 11915, 11916, 11917, and 11918).	253	11919	A	1	3.07	35.26	57.30	4.37	.62	5.45	78.31	1.70	9.55	1.2	7,762	13,972
				2		36.38	59.11	4.51	.64	5.27	80.79	1.75	7.04		8,008	14,414
				3		38.10	61.90		.67	5.52	84.60	1.83	7.38		8,386	15,095
2 miles east of; T. 5 N., R. 15 E.; Alderson No. 38 mine, McAlester bed (room 1, off west entry 7, 2,200 feet down main slope, 2-foot 9-inch bed and cut).	254	11592	A	1	3.19	32.71	57.06	7.04	.50	5.23	75.57	1.62	10.04	1.7	7,486	13,475
				2		33.79	58.94	7.27	.52	5.04	78.06	1.67	7.44		7,733	13,919
				3		36.44	63.56		.56	5.44	84.20	1.80	8.00		8,339	15,010
Same (room 4, off east entry 5, 2,300 feet down main slope, 3-foot 3-inch bed and cut).	254	11594	A	1	3.57	33.38	58.05	5.00	.49	5.34	77.37	1.66	10.14	2.0	7,634	13,741
				2		34.62	60.19	5.19	.51	5.12	80.23	1.72	7.23		7,916	14,249
				3		36.51	63.49		.54	5.40	84.63	1.81	7.62		8,349	15,028
Same (room 26, off east entry 4, 1,900 feet down main slope, 3-foot 6-inch bed and cut).	254	11670	A	1	3.22	34.84	57.21	4.73	.48	5.25	78.18	1.73	9.63	1.8	7,707	13,873
				2		36.00	59.11	4.89	.50	5.05	80.78	1.79	6.99		7,964	14,335
				3		37.85	62.15		.53	5.31	84.93	1.88	7.35		8,373	15,071
Same (face of east entry 6, 3-foot 6-inch bed and cut).	254	15188	A	1	2.84	35.71	55.93	5.52	.61	5.38	77.69	1.89	8.91	1.2	7,654	13,777
				2		36.75	57.57	5.68	.63	5.21	79.96	1.95	6.57		7,877	14,179
				3		38.96	61.04		.67	5.52	84.77	2.07	6.97		8,351	15,032
Same (face of west air course 7, 3-foot 9-inch bed and cut).	254	15189	A	1	3.21	35.65	56.50	4.64	.67	5.28	77.43	1.85	10.13	1.5	7,694	13,849
				2		36.83	58.38	4.79	.69	5.08	80.00	1.91	7.53		7,949	14,308
				3		38.68	61.32		.72	5.34	84.02	2.01	7.91		8,349	15,028
Craig, Bolen-Darnall No. 4 mine, McAlester bed (face of eighth west entry, 3,000 feet northwest of mine mouth, 2-foot 11½-inch bed, 2-foot 10½-inch cut).	254	15104	A	1	3.51	36.23	55.09	5.17	.82					1.7	7,531	13,556
				2		37.55	57.09	5.36	.85						7,805	14,049
				3		39.68	60.32		.90						8,247	14,845
Same (face of eighth east entry, 3,000 feet from mine mouth, 3-foot ½-inch bed and cut).	254	15105	A	1	3.57	36.06	55.15	5.22	.86					1.7	7,536	13,565
				2		37.39	57.20	5.41	.89						7,815	14,067
				3		39.53	60.47		.94						8,262	14,872
McAlester, 2 miles east of, sec. 32, T. 6 N., R. 15 E.; Busby No. 5 mine, McAlester bed (room 11, off west entry 3, off east slope, 1,600 feet from shaft, 4-foot 2-inch bed and cut).	255	11569	A	1	3.28	33.79	57.75	5.18	.54	5.20	77.96	1.69	9.43	1.9	7,576	13,637
				2		34.94	59.70	5.36	.56	5.00	80.60	1.75	6.73		7,833	14,099
				3		36.92	63.08		.59	5.28	85.16	1.85	7.12		8,276	14,897
Same (face of west entry 2, off west slope, 300 feet beyond room [illegible], 4-foot bed and cut).	255	11586	A	1	3.99	31.01	59.51	5.49	.56					2.4		
				2		32.30	61.98	5.72	.58							
				3		34.26	65.74		.62							
Same (face of east entry 3, off east slope, 120 feet beyond room 22, 4-foot 2-inch bed and cut).	255	11587	A	1	3.56	32.94	57.93	5.57	.55					2.1		
				2		34.16	60.06	5.78	.57							
				3		36.25	63.75		.60							

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
OKLAHOMA—Continued.																
PITTSBURG COUNTY—continued.																
McAlester, 2 miles east of, sec. 32, T. 6 N., R. 15 E., Busby No. 5 mine, McAlester bed (face of east entry 4, off west slope, 4-foot bed and cut).	255	11588	A	1	3.29	32.88	58.59	5.24	0.58					1.8		
				2		34.00	60.58	5.42	.60							
				3		35.95	64.05		.63							
Same (face of east entry 5, off east slope, 4-foot 3-inch bed and cut).	255	11589	A	1	3.13	32.52	59.69	4.66	.52					1.8		
				2		33.57	61.62	4.81	.54							
				3		35.27	64.73		.57							
Same (face of east entry 3, off west slope, 200 feet beyond room 8, 4-foot bed and cut).	255	11590	A	1	3.17	32.31	59.41	5.11	.61					1.7		
				2		33.37	61.35	5.28	.63							
				3		35.23	64.77		.67							
Same (composite of samples 11586–11590).........	255	11591	A	1	3.58	32.11	59.04	5.27	.56	5.31	77.11	1.62	10.13	2.0	7,564	13,615
				2		33.31	61.22	5.47	.58	5.09	79.97	1.68	7.21		7,845	14,121
				3		35.24	64.76		.61	5.38	84.60	1.78	7.63		8,299	14,938
OREGON.																
COOS COUNTY.																
West Fork, 36 miles southwest of; NE. ¼ sec. 28, T. 32 S., R. 11 W.; Flanigan prospect, Anderson bed (in entry, 125 feet from mouth, 5-foot 6-inch bed, 5-foot 1½-inch cut).	255	14828	B	1	10.13	31.31	24.91	33.65	1.11					6.2	4,148	7,466
				2		34.84	27.72	37.44	1.24						4,615	8,307
				3		55.69	44.31		1.98						7,377	13,279
37 miles southwest of; SW. ¼ NE. ¼ sec. 29, T. 32 S., R. 11 W.; Hammond prospect, Carter bed (in entry, 100 feet from mouth, 6-foot bed, 4-foot 5-inch cut).	256	14831	B	1	8.05	34.98	37.12	19.85	.58					3.7	5,460	9,828
				2		38.04	40.37	21.59	.63						5,938	10,688
				3		48.52	51.48		.80						7,573	13,631
36 miles southwest of; SE. ¼ NE. ¼ sec. 32, T. 32 S., R. 11 W.; Hillis prospect, Anderson bed (in entry 90 feet from mouth, 6-foot 10-inch bed, 1-foot 3½-inch cut).	256	14827	B	1	9.97	38.45	36.35	15.23	.63					6.7	5,849	10,528
				2		42.71	40.37	16.92	.70						6,496	11,693
				3		51.41	48.59		.84						7,819	14,074
Same (in entry 90 feet from mouth, 6-foot 10-inch bed, 2-foot 9½-inch cut).	256	14830	B	1	6.71	35.51	27.41	30.37	2.22					3.3	4,750	8,550
				2		38.06	29.39	32.55	2.38						5,092	9,166
				3		56.43	43.57		3.53						7,549	13,588

38 miles southwest of; NW. ¼ NE. ¼ sec. 30, T. 32 S., R. 11 W.; Vanderpool prospect, Anderson bed (in entry, 217 feet from mouth, 5-foot 11½-inch bed, 1-foot 10-inch cut).	257	14829	B	1	7.28	37.63	27.56	27.53	2.59	....	....	....	....	3.3	4,917	8,851
				2	....	40.58	29.73	29.69	2.79	....	....	....	....	....	5,308	9,545
				3	....	57.72	42.28	....	3.97	....	....	....	....	....	7,542	13,576
MALHEUR COUNTY.																
Nyssa, 25 miles southwest of; NE. ¼ sec. 2, T. 22 S., R. 44 E. (unsurveyed); Honolulu prospect (at mouth of slope, 2-foot 4-inch bed, 8-inch cut).	257	*12585	B	1	30.31	19.03	18.86	31.80	2.66	....	....	....	....	20.7	2,431	4,376
				2	....	27.31	27.06	45.63	3.82	....	....	....	....	....	3,488	6,278
				3	....	50.23	49.77	....	7.03	....	....	....	....	....	6,415	11,547
PENNSYLVANIA.																
ALLEGHENY COUNTY.																
Bruceton, about 1 mile south of; near Wallace Station; Experimental mine, Pittsburgh bed (rib of first cross cut, 200 feet from mouth of air course, 5-foot 11-inch bed, 5-foot 7⅞-inch cut).	258	11983	A	1	3.11	35.16	55.95	5.78	1.17	....	....	....	....	1.3	....	....
				2	....	36.29	57.74	5.97	1.21	....	....	....	....	....	....	....
				3	....	38.59	61.41	....	1.29	....	....	....	....	....	....	....
Same (rib near face of air course, 390 feet from mouth of air course, 5-foot 7 1/16-inch bed, 5-foot 3-inch cut).	258	11984	A	1	2.68	35.37	55.76	6.19	1.48	....	....	....	....	1.0	....	....
				2	....	36.34	57.30	6.36	1.52	....	....	....	....	....	....	....
				3	....	38.81	61.19	....	1.62	....	....	....	....	....	....	....
Same (face of main entry, 345 feet from mouth of main entry, 5-foot 3-inch bed, 5-foot ¾-inch cut).	258	11985	A	1	2.65	35.49	55.18	6.68	1.49	....	....	....	....	1.0	....	....
				2	....	36.46	56.68	6.86	1.53	....	....	....	....	....	....	....
				3	....	39.15	60.85	....	1.64	....	....	....	....	....	....	....
Same (composite of samples 11983, 11984, and 11985).	258	11986	A	1	2.73	36.03	54.98	6.26	1.39	5.26	76.82	1.46	8.81	1.1	7,675	13,815
				2	....	37.04	56.52	6.44	1.43	5.10	78.98	1.50	6.55	....	7,891	14,204
				3	....	39.59	60.41	....	1.53	5.45	84.41	1.60	7.01	....	8,434	15,181
Same (in air course, 75 feet from outcrop at mouth of air course, 5-foot 11½-inch bed, 5-foot 10½-inch cut).	258	12251	A	1	2.87	35.21	57.56	4.36	1.15	5.37	78.49	1.56	9.07	0.9	7,800	14,040
				2	....	36.25	59.26	4.49	1.18	5.20	80.81	1.61	6.71	....	8,030	14,454
				3	....	37.95	62.05	....	1.24	5.44	84.61	1.69	7.02	....	8,407	15,133
Same (left side of air course, 150 feet from mouth of air course, 6-foot bed, 5-foot 10½-inch cut).	258	12262	A	1	3.34	35.16	56.91	4.59	1.42	5.38	77.51	1.51	9.59	1.4	7,764	13,975
				2	....	36.38	58.87	4.75	1.47	5.18	80.19	1.56	6.85	....	8,033	14,456
				3	....	38.20	61.80	....	1.54	5.44	84.19	1.64	7.19	....	8,434	15,181
Same (left side of air course, 430 feet from outcrop at mouth of air course, 5-foot 5-inch bed, 5-foot 2½-inch cut).	258	12264	A	1	2.87	35.12	55.84	6.17	1.52	5.34	77.04	1.53	8.40	0.9	7,644	13,759
				2	....	36.16	57.49	6.35	1.56	5.17	79.31	1.58	6.03	....	7,869	14,164
				3	....	38.61	61.39	....	1.67	5.52	84.69	1.69	6.43	....	8,403	15,125
Same (530 feet inby outcrop, right side of main entry, 5-foot 2¾-inch bed, 4-foot 9½-inch cut).	258	12569	A	1	3.32	36.82	54.19	5.67	1.03	5.32	76.94	1.53	9.51	1.3	7,635	13,743
				2	....	38.08	56.06	5.86	1.07	5.12	79.58	1.58	6.79	....	7,897	14,215
				3	....	40.45	59.55	....	1.14	5.44	84.53	1.68	7.21	....	8,388	15,098
Same (580 feet inby outcrop; right side of main entry, 5-foot 2-inch bed, 4-foot 10-inch cut).	258	12572	A	1	2.93	36.19	55.14	5.74	1.15	5.46	76.83	1.51	9.31	0.8	7,645	13,761
				2	....	37.28	56.81	5.91	1.18	5.28	79.15	1.56	6.92	....	7,876	14,177
				3	....	39.62	60.38	....	1.25	5.61	84.12	1.66	7.36	....	8,371	15,068
Same (630 feet inby outcrop; right side of main entry, 5-foot 2¾-inch bed, 5-foot ¾-inch cut).	258	12577	A	1	3.22	34.64	53.68	8.46	1.07	5.20	74.24	1.46	9.57	1.3	7,377	13,279
				2	....	35.79	55.47	8.74	1.11	5.00	76.71	1.51	6.93	....	7,623	13,721
				3	....	39.22	60.78	....	1.22	5.48	84.06	1.65	7.59	....	8,353	15,035
Same (680 feet inby outcrop; right side of main entry, 5-foot 4-inch bed and cut).	258	12557	A	1	3.30	36.09	53.46	7.15	1.17	5.36	74.58	1.53	10.21	1.3	7,509	13,516
				2	....	37.32	55.29	7.39	1.21	5.16	77.12	1.58	7.54	....	7,765	13,977
				3	....	40.30	59.70	....	1.31	5.57	83.27	1.71	8.14	....	8,385	15,093
Same (in crosscut, off main entry, at station E-1250, 3 feet from main entry, 5-foot 5½-inch bed, 5-foot 3-inch cut).	258	14797	A	1	3.34	35.49	54.54	6.63	1.24	....	....	....	....	1.9	....	....
				2	....	36.72	56.42	6.86	1.28	....	....	....	....	....	....	....
				3	....	39.43	60.57	....	1.37	....	....	....	....	....	....	....
Same (in main entry at station E-1270, 5 foot 7½ inch bed, 5-foot 2¾-inch cut).	258	14798	A	1	3.27	35.84	54.97	5.92	1.33	....	....	....	....	1.8	....	....
				2	....	37.05	56.83	6.12	1.37	....	....	....	....	....	....	....
				3	....	39.47	60.53	....	1.46	....	....	....	....	....	....	....

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.					Ultimate.				Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
ALLEGHENY COUNTY—continued.																
Creighton; Hite mine, Upper Freeport (E) and Lower Freeport (D) beds (face of room 17, butt entry 1, 8,000 feet west of mine mouth, 5-foot 10-inch bed, 4-foot 11-inch cut).	260	11130	A	1	2.69	34.80	55.99	6.52	0.96	......	......	......	......	1.2	......	......
				2	......	35.76	57.54	6.70	.99	......	......	......	......	......	......	......
				3	......	38.33	61.67	......	1.06	......	......	......	......	......	......	......
Same (face of room 18, butt entry 3, 9,000 feet from mine mouth, 5-foot 11-inch bed, 5-foot 3-inch cut).	260	11132	A	1	2.66	34.35	56.42	6.57	.99	......	......	......	......	1.2	......	......
				2	......	35.29	57.96	6.75	1.02	......	......	......	......	......	......	......
				3	......	37.84	62.16	......	1.09	......	......	......	......	......	......	......
Same (grab sample of cannel (roof) coal, at intersection of line entry and main entry 1, 1-foot 3-inch cut).	260	11133	A	1	1.12	26.17	36.45	36.26	1.58	......	......	......	......	0.4	......	......
				2	......	26.47	36.86	36.67	1.60	......	......	......	......	......	......	......
				3	......	41.80	58.20	......	2.53	......	......	......	......	......	......	......
Oak Station, short distance south of Pittsburgh; Oak mine, Pittsburgh bed (room 10, left entry 13, 5,300 feet north of mine mouth).	266	10845	A	1	3.48	35.15	55.45	5.92	1.18	5.42	75.73	1.45	10.30	1.7	7,611	13,700
				2	......	36.42	57.45	6.13	1.22	5.21	78.46	1.50	7.48	......	7,885	14,193
				3	......	38.80	61.20	......	1.30	5.55	83.58	1.60	7.97	......	8,400	15,120
BEDFORD COUNTY.																
Hopewell, 2 miles east of; Cambria No. 3 mine, Upper Freeport (locally known as Kelley) bed (face of heading 2, 7,300 feet northeast of shaft, 3-foot 11½-inch bed and cut).	261	15057	A	1	......	......	......	......	......	......	......	......	......	......	......	......
				2	......	17.08	70.50	12.42	2.26	......	......	......	......	......	7,594	13,669
				3	......	19.50	80.50	......	2.58	......	......	......	......	......	8,671	15,608
Same (face of room 7, off heading 2, 7,000 feet northeast of shaft; 3-foot 10⅝-inch bed, 3-foot 9⅝-inch cut).	261	15055	A	1	1.55	15.86	68.84	13.75	2.04	......	......	......	......	1.2	7,299	13,138
				2	......	16.11	69.92	13.97	2.07	......	......	......	......	......	7,414	13,345
				3	......	18.73	81.27	......	2.41	......	......	......	......	......	8,618	15,512
Same (600 feet to the right from top of plane 3; 5,100 feet northeast of shaft; 4-foot 8¼-inch bed, 4-foot 3¼-inch cut).	261	15056	A	1	1.25	16.58	71.67	10.50	1.00	......	......	......	......	1.0	7,646	13,763
				2	......	16.79	72.58	10.63	1.01	......	......	......	......	......	7,743	13,937
				3	......	18.79	81.21	......	1.13	......	......	......	......	......	8,664	15,595
Same (face of heading 3; 7,400 feet northeast of shaft; 3-foot 11½-inch bed and cut).	261	15058	A	1	1.74	15.88	69.60	12.78	2.42	......	......	......	......	1.4	7,355	13,239
				2	......	16.16	70.83	13.01	2.46	......	......	......	......	......	7,485	13,473
				3	......	18.58	81.42	......	2.83	......	......	......	......	......	8,605	15,489

Same (face of room, 200 feet from face of heading 4, 7,000 feet northeast of shaft, 3-foot 7½-inch bed and cut).	261	15059	A	1	1.61	16.78	69.70	11.91	2.20						1.2	7,496	13,493
				2		17.06	70.83	12.11	2.24							7,619	13,714
				3		19.41	80.59		2.55							8,669	15,604
Same (composite of samples 15055, 15056, 15058, and 15059).	261	15060	A	1	1.58	16.32	69.98	12.12	1.94	4.09	77.01	1.44	3.40		1.2	7,449	13,408
				2		16.58	71.10	12.32	1.97	3.97	78.25	1.46	2.03			7,569	13,624
				3		18.91	81.09		2.25	4.53	89.24	1.67	2.31			8,632	15,538
CAMBRIA COUNTY.																	
Barnesboro, 1 mile east of; Allport mine, Lower Freeport or D bed (face of left entry 2, off heading 9, off main heading, 6,250 feet southeast of mine mouth, 4-foot 4½-inch bed, 4-foot 1½-inch cut).	262	W19740	A	1	3.5	20.9	68.2	7.4	1.80							7,739	13,930
				2		21.7	70.6	7.7	1.85							8,017	14,430
				3		23.5	76.5		2.00							8,683	15,630
Same (rib in room 7, off left entry 2, off right entry 9, off main heading, 6,250 feet southeast of mine mouth, 4-foot 10¼-inch bed, 4-foot 9½-inch cut).	262	W19741	A	1	2.9	20.7	68.8	7.6	1.75							7,806	14,050
				2		21.3	70.9	7.8	1.80							8,039	14,470
				3		23.1	76.9		1.95							8,722	15,700
Same (face of right entry 9, off main heading, 6,950 feet southwest of mine mouth, 4-foot 8-inch bed, 4-foot 6½-inch cut).	262	W19742	A	1	2.3	21.9	69.2	6.6	1.40							7,933	14,280
				2		22.4	70.8	6.8	1.45							8,117	14,610
				3		24.0	76.0		1.56							8,706	15,670
Same (composite of samples W19740, W19741, and W19742).	262	12221	A	1	1.75	21.70	69.13	7.42	1.81	4.69	80.33	1.38	4.37			7,904	14,227
				2		22.09	70.36	7.55	1.84	4.58	81.76	1.40	2.87			8,045	14,481
				3		23.89	76.11		1.99	4.95	88.44	1.51	3.11			8,702	15,664
1 mile east of; Allport mine, Upper Freeport or E bed (face of right entry 3, off main heading, 3,700 feet southeast of mine mouth, 3-foot 8¾-inch bed, 3-foot 6-inch cut).	262	W19738	A	1	2.3	22.2	67.6	7.9	1.05							7,822	14,080
				2		22.7	69.2	8.1	1.10							8,011	14,420
				3		24.7	75.3		1.20							8,717	15,690
Same (face of main slant, off main entry, 3,700 feet southeast of mine mouth, 3-foot 7-inch bed, 3-foot 5-inch cut).	262	W19739	A	1	2.4	22.7	67.4	7.5	.80							7,883	14,190
				2		23.3	69.0	7.7	.80							8,072	14,530
				3		25.2	74.8		.87							8,744	15,740
Same (600 feet in on second panel from main slant, off main entry, 2,800 feet south of mine mouth, 3-foot 7½-inch bed and cut).	262	W19743	A	1	2.5	23.2	68.4	5.9	.65							7,989	14,380
				2		23.8	70.1	6.1	.70							8,189	14,740
				3		25.3	74.7		.75							8,717	15,690
Same (composite of samples W19738, W19739, and W19743).	262	12231	A	1	1.45	23.40	67.92	7.23	.83	4.78	81.06	1.35	4.75			7,934	14,281
				2		23.74	68.92	7.34	.84	4.69	82.25	1.37	3.51			8,051	14,492
				3		25.62	74.38		.91	5.06	88.76	1.48	3.79			8,689	15,640
1¾ miles east of; Cymbria mine, Miller, B, or Lower Kittanning bed (face of right entry 2, about 1,000 feet southwest of shaft, 2-foot 10½-inch bed, 2-foot 9¾-inch cut).	263	W20394	A	1	2.2	20.4	71.5	5.9	.85							8,022	14,440
				2		20.9	73.1	6.0	.85							8,206	14,770
				3		22.2	77.8		.90							8,728	15,710
Same (face of room 18, off left entry 2, 1,200 feet northeast of shaft, 2-foot 7¼-inch bed and cut).	263	W20395	A	1	2.1	21.7	70.8	5.4	.85							8,094	14,570
				2		22.2	72.3	5.5	.85							8,272	14,890
				3		23.5	76.5		.90							8,756	15,760
Same (face of main heading, 1,900 feet from shaft, 2-foot 4⅝-inch bed, 2-foot 4¼-inch cut).	263	W20444	A	1	2.3	21.4	67.4	8.9	3.00							7,728	13,910
				2		21.9	69.0	9.1	3.10							7,911	14,240
				3		24.1	75.9		3.41							8,706	15,670
Same (face of left entry 3, 1,400 feet northeast of shaft, 2-foot 5-inch bed and cut).	263	W20445	A	1	3.1	21.3	67.1	8.5	3.15							7,694	13,850
				2		22.0	69.2	8.8	3.30							7,939	14,290
				3		24.1	75.9		3.62							8,700	15,660
Same (face of right entry 3, 1,200 feet southeast of shaft, 2-foot 11-inch bed and cut).	263	W20446	A	1	3.4	22.1	68.3	6.2	1.20							7,944	14,300
				2		22.9	70.7	6.4	1.25							8,228	14,810
				3		24.5	75.5		1.34							8,794	15,830
Same (composite of samples W20394, W20395, W20444, W20445, and W20446).	263	12372	A	1	.82	22.86	69.15	7.17	2.02	4.66	81.56	1.23	3.36			8,024	14,443
				2		23.05	69.72	7.23	2.04	4.61	82.24	1.24	2.64			8,091	14,564
				3		24.85	75.15		2.20	4.97	88.65	1.34	2.84			8,721	15,698

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Barnesboro, 2 miles east of; Cymbria No. 1 mine, Lower Freeport or D bed (face of first crosscut, off left entry 2, 5,500 feet from mine mouth, 4-foot 5-inch bed, 4-foot 1-inch cut).	264	12162	A	1	3.42	22.74	66.91	6.93	1.77	......	......	......	......	2.8	7,801	14,042
				2	......	23.54	69.28	7.18	1.83	......	......	......	......	......	8,077	14,539
				3	......	25.36	74.64	......	1.97	......	......	......	......	......	8,702	15,664
Same (face of eighth right, off left entry 2, 4,000 feet from mine mouth, 3-foot 11½-inch bed, 3-foot 6½-inch cut).	264	12163	A	1	2.57	23.10	67.49	6.84	1.22	......	......	......	......	2.0	7,891	14,204
				2	......	23.71	69.27	7.02	1.25	......	......	......	......	......	8,099	14,578
				3	......	25.50	74.50	......	1.34	......	......	......	......	......	8,710	15,678
Same (face of first left, off left entry 2, 4,000 feet from mine mouth, 4-foot 7½-inch bed, 4-foot 3¾-inch cut).	264	12164	A	1	2.95	23.41	67.74	5.90	1.34	......	......	......	......	2.4	7,940	14,292
				2	......	24.12	69.80	6.08	1.38	......	......	......	......	......	8,181	14,726
				3	......	25.68	74.32	......	1.47	......	......	......	......	......	8,710	15,678
Same (face of third right, off main entry 2, 3,200 feet northeast of mine mouth, 4-foot 1¾-inch bed, 3-foot 10½-inch cut).	264	12170	A	1	2.76	22.69	68.08	6.47	1.57	......	......	......	......	2.2	7,882	14,188
				2	......	23.33	70.02	6.65	1.61	......	......	......	......	......	8,106	14,591
				3	......	24.99	75.01	......	1.72	......	......	......	......	......	8,683	15,629
Same (face of second left, off main heading, 2,900 feet northeast of mine mouth, 4-foot 3½-inch bed, 4-foot ¾-inch cut).	264	12171	A	1	2.98	23.46	67.84	5.72	1.41	......	......	......	......	2.4	7,963	14,333
				2	......	24.18	69.92	5.90	1.45	......	......	......	......	......	8,207	14,773
				3	......	25.70	74.30	......	1.54	......	......	......	......	......	8,722	15,700
Same (composite of samples 12162, 12163, 12164, 12170, and 12171).	264	12165	A	1	3.08	22.50	67.93	6.49	1.59	5.01	80.39	1.26	5.26	2.4	7,875	14,175
				2	......	23.22	70.08	6.70	1.64	4.82	82.95	1.30	2.59	......	8,125	14,625
				3	......	24.89	75.11	......	1.76	5.17	88.91	1.39	2.77	......	8,708	15,674
2 miles east of; Cymbria No. 2½ mine, Upper Freeport or E bed (face of left entry 2, 2,000 feet from mine mouth, 3-foot 7¾-inch bed, 3-foot 4¾-inch cut).	265	12158	A	1	2.76	24.03	63.89	9.32	1.52	......	......	......	......	2.2	7,647	13,765
				2	......	24.71	65.71	9.58	1.56	......	......	......	......	......	7,864	14,155
				3	......	27.33	72.67	......	1.73	......	......	......	......	......	8,697	15,655
Same (face of room 12, off main heading, 2,200 feet from mine mouth, 3-foot 7½-inch bed, 3-foot 5-inch cut).	265	12159	A	1	3.24	23.16	65.86	7.74	1.38	......	......	......	......	2.6	7,773	13,991
				2	......	23.94	68.06	8.00	1.43	......	......	......	......	......	8,033	14,459
				3	......	26.02	73.98	......	1.55	......	......	......	......	......	8,732	15,718
Same (face of third left main heading, 3-foot 7¾-inch bed, 3-foot 4¾-inch cut).	265	12160	A	1	3.88	22.74	65.76	7.62	1.37	......	......	......	......	3.2	7,720	13,896
				2	......	23.66	68.41	7.93	1.43	......	......	......	......	......	8,032	14,458
				3	......	25.70	74.30	......	1.55	......	......	......	......	......	8,724	15,708

Same (face of room 6, off main heading, 1,500 feet southwest of mine mouth, 3-foot 7¾-inch bed, 3-foot 4½-inch cut).	265	12172	A	1	2.88	23.22	65.74	8.16	1.67	....	....	....	....	2.3	7,745	13,941
				2	....	23.91	67.69	8.40	1.72	....	....	....	....	....	7,975	14,355
				3	....	26.10	73.90	....	1.88	....	....	....	....	....	8,706	15,671
Same (face of room 8, off left heading 2, 1,200 feet southwest of mine mouth, 3-foot 9½-inch bed, 3-foot 8-inch cut).	265	12173	A	1	2.97	22.98	65.43	8.62	1.98	....	....	....	....	2.4	7,688	13,838
				2	....	23.68	67.44	8.88	2.04	....	....	....	....	....	7,923	14,261
				3	....	25.99	74.01	....	2.24	....	....	....	....	....	8,695	15,651
Same (composite of samples 12158, 12159, 12160, 12172, and 12173).	265	12161	A	1	2.98	22.85	65.78	8.39	1.63	5.02	77.87	1.32	5.77	2.5	7,726	13,907
				2	....	23.55	67.80	8.65	1.68	4.84	80.26	1.36	3.21	....	7,963	14,333
				3	....	25.78	74.22	....	1.84	5.30	87.86	1.49	3.51	....	8,717	15,691
¾ mile west of; Empire A mine, Lower Freeport or D bed (room 1, off left entry 5, off new parallel heading, 14,000 feet northwest of drift mouth, 4-foot 4¾-inch bed, 4-foot ½-inch cut).	265	W19643	A	1	2.6	23.3	66.9	7.2	1.25	....	....	....	....	....	7,817	14,070
				2	....	23.9	68.7	7.4	1.30	....	....	....	....	....	8,022	14,440
				3	....	25.8	74.2	....	1.40	....	....	....	....	....	8,667	15,600
Same (face of room 4, off old first left heading, off main entry, 2,000 feet northwest of drift mouth, 4-foot 6¾-inch bed, 4-foot 3-inch cut).	265	W19644	A	1	1.9	23.8	67.6	6.7	1.05	....	....	....	....	....	7,956	14,320
				2	....	24.3	68.9	6.8	1.10	....	....	....	....	....	8,111	14,600
				3	....	26.1	73.9	....	1.18	....	....	....	....	....	8,700	15,660
Same (corner of back heading, off right entry 3, off old parallel heading, 3,000 feet northeast of drift mouth, 4-foot 4¼-inch bed, 4-foot 2¼-inch cut).	265	W19645	A	1	2.3	23.1	67.5	7.1	1.15	....	....	....	....	....	7,828	14,090
				2	....	23.7	69.0	7.3	1.20	....	....	....	....	....	8,017	14,430
				3	....	25.6	74.4	....	1.29	....	....	....	....	....	8,650	15,570
Same (40 feet from face of parallel heading, off main entry, 9,000 feet northeast of drift mouth, 4-foot 3⅞-inch bed, 4-foot ½-inch cut).	265	W19646	A	1	3.3	24.5	65.0	7.2	1.85	....	....	....	....	....	7,778	14,000
				2	....	25.3	67.3	7.4	1.90	....	....	....	....	....	8,044	14,480
				3	....	27.3	72.7	....	2.05	....	....	....	....	....	8,689	15,640
Same (face of room 24, off left entry 14, off main entry, 11,000 feet northwest of drift mouth, 3-foot 9½-inch bed, 3-foot 6⅝-inch cut).	265	W19647	A	1	2.1	22.7	66.5	8.7	1.80	....	....	....	....	....	7,728	13,910
				2	....	23.2	67.9	8.9	1.85	....	....	....	....	....	7,894	14,210
				3	....	25.5	74.5	....	2.03	....	....	....	....	....	8,667	15,600
Same (face of room 17, off left entry 3, off new parallel heading, 14,000 feet northwest of drift mouth, 4-foot 10½-inch bed, 4-foot 6½-inch cut).	265	W19648	A	1	2.4	23.5	67.2	6.9	1.50	....	....	....	....	....	7,883	14,190
				2	....	24.1	68.8	7.1	1.55	....	....	....	....	....	8,078	14,540
				3	....	25.9	74.1	....	1.67	....	....	....	....	....	8,694	15,650
Same (composite of W19643, W19644, W19645, W19646, W19647, and W19648).	265	12212	A	1	1.54	23.86	67.08	7.52	1.53	4.73	80.47	1.38	4.37	....	7,886	14,195
				2	....	24.23	68.13	7.64	1.55	4.63	81.73	1.40	3.05	....	8,009	14,416
				3	....	26.23	73.77	....	1.68	5.01	88.49	1.52	3.30	....	8,671	15,608
Beaverdale; Pennsylvania No. 15 mine, Kittanning or B bed (face of first slant, off right heading 5, off first dip, off main entry, 3-foot 3-inch bed and cut).	266	12174	A	1	3.25	15.85	73.42	7.48	2.10	....	....	....	....	2.8	7,807	14,053
				2	....	16.38	75.89	7.73	2.17	....	....	....	....	....	8,069	14,524
				3	....	17.75	82.25	....	2.35	....	....	....	....	....	8,745	15,741
Same (face of sump for main dip, off left heading 4, off first dip, off main entry, 3-foot 4½-inch bed and cut).	266	12175	A	1	2.72	16.70	75.38	5.20	.55	....	....	....	....	2.2	8,067	14,521
				2	....	17.17	77.48	5.35	.57	....	....	....	....	....	8,293	14,927
				3	....	18.14	81.86	....	.60	....	....	....	....	....	8,762	15,772
Same (face of first slant, off right heading 4, off first dip off main entry, 3-foot 4-inch bed and cut).	266	12176	A	1	3.90	15.09	73.03	7.98	2.28	....	....	....	....	3.3	7,720	13,896
				2	....	15.70	76.00	8.30	2.37	....	....	....	....	....	8,033	14,459
				3	....	17.12	82.88	....	2.58	....	....	....	....	....	8,760	15,768
Same (face of first slant, off right heading 3, off first dip off main entry, 3-foot 1-inch bed and cut).	266	12177	A	1	3.53	15.68	73.95	6.84	1.94	....	....	....	....	3.0	7,847	14,125
				2	....	16.25	76.66	7.09	2.01	....	....	....	....	....	8,134	14,641
				3	....	17.49	82.51	....	2.16	....	....	....	....	....	8,755	15,759
Same (composite of samples 12174, 12175, 12176, and 12177).	266	12178	A	1	3.44	16.18	73.46	6.92	1.83	4.64	80.61	1.20	4.80	2.8	7,841	14,114
				2	....	16.76	76.07	7.17	1.90	4.41	83.48	1.24	1.80	....	8,120	14,616
				3	....	18.05	81.95	....	2.05	4.75	89.92	1.34	1.94	....	8,747	15,745
Same (pillar, room 23, off right heading 3, off first dip, off main entry).	266	12179	A	1	2.89	14.68	74.83	7.60	2.36	4.59	80.18	1.25	4.02	2.4	7,817	14,071
				2	....	15.12	77.05	7.83	2.43	4.40	82.57	1.29	1.48	....	8,050	14,490
				3	....	16.41	83.59	....	2.64	4.77	89.59	1.40	1.60	....	8,734	15,721

Table of chemical analyses—Continued.

Locality, bed, etc	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Bens Creek, 1 mile east of; Bens Creek No. 1 mine, Miller, B, or Lower Kittanning bed (barrier pillar on No. 2 side track, 100 feet from room 21 and 3,200 feet southwest of old main heading, 3-foot 6⅝-inch bed and cut).	267	W19697	A	1	2.1	17.0	74.7	6.2	1.10	......	......	......	......	......	8,017	14,430
				2	......	17.4	76.3	6.3	1.15	......	......	......	......	......	8,189	14,740
				3	......	18.6	81.4	......	1.23	......	......	......	......	......	8,739	15,730
Same (pillar on No. 1 side track, between manway and room 1, 1,200 feet southwest cf shaft, 3-foot 1½-inch bed and cut).	267	W19700	A	1	3.4	16.3	74.7	5.6	.55	......	......	......	......	......	7,956	14,320
				2	......	16.9	77.3	5.8	.60	......	......	......	......	......	8,233	14,820
				3	......	17.9	82.1	......	.64	......	......	......	......	......	8,739	15,730
Same (main pillar of old main heading, 900 feet southwest of shaft, 3-foot 9-inch bed and cut).	267	W19702	A	1	3.7	17.7	72.7	5.9	.65	......	......	......	......	......	7,933	14,280
				2	......	18.3	75.6	6.1	.70	......	......	......	......	......	8,233	14,820
				3	......	19.5	80.5	......	.75	......	......	......	......	......	8,772	15,790
Same (pillar of heading 1, between rooms 27 and 28, 2,700 feet southwest of shaft, 3-foot 7¾-inch bed and cut).	267	W19737	A	1	3.3	16.3	74.7	5.7	1.05	......	......	......	......	......	8,000	14,400
				2	......	16.9	77.2	5.9	1.10	......	......	......	......	......	8,272	14,890
				3	......	18.0	82.0	......	1.17	......	......	......	......	......	8,789	15,820
Same (composite of samples W19697, W19700, W19702, and W19737).	267	12211	A	1	1.83	16.82	75.34	6.01	.84	4.56	83.43	1.28	3.88	......	8,080	14,544
				2	......	17.13	76.75	6.12	.86	4.44	84.98	1.30	2.30	......	8,230	14,814
				3	......	18.25	81.75	......	.92	4.73	90.52	1.38	2.45	......	8,767	15,781
Same (face of heading 6, off first dip, 700 feet southeast of shaft, 3-foot 4-inch bed and cut).	267	W19736	A	1	2.6	16.6	74.8	6.0	.75	......	......	......	......	......	8,028	14,450
				2	......	17.0	76.8	6.2	.80	......	......	......	......	......	8,239	14,830
				3	......	18.1	81.9	......	.85	......	......	......	......	......	8,783	15,810
1 mile south of; Pennsylvania No. 2 mine, Miller, B, or Lower Kittanning bed (face of heading 19, 4,100 feet northeast of drift mouth, 3-foot 9½-inch bed, 3-foot 7¾-inch cut).	268	W19695	A	1	2.8	16.2	74.0	7.0	1.50	......	......	......	......	......	7,961	14,330
				2	......	16.7	76.1	7.2	1.55	......	......	......	......	......	8,189	14,740
				3	......	18.0	82.0	......	1.67	......	......	......	......	......	8,822	15,880
Same (pillar between rooms 14 and 15, fourth main heading, 2,900 feet northwest of drift mouth, 3-foot 3⅜-inch bed, 3-foot 3⅜-inch cut).	268	W19696	A	1	2.4	16.6	74.3	6.7	1.10	......	......	......	......	......	7,978	14,360
				2	......	17.0	76.1	6.9	1.15	......	......	......	......	......	8,178	14,720
				3	......	18.3	81.7	......	1.24	......	......	......	......	......	8,783	15,810
Same (pillar between rooms 2 and 3, off butt entry 17, 3,900 feet north of drift mouth, 3-foot 7¾-inch bed and cut).	268	W19698	A	1	2.8	16.4	75.6	5.2	.90	......	......	......	......	......	8,083	14,550
				2	......	16.9	77.8	5.3	.95	......	......	......	......	......	8,317	14,970
				3	......	17.8	82.2	......	1.00	......	......	......	......	......	8,783	15,810

Same (pillar between rooms 1 and 2, off butt entry 16, 3,900 feet north of drift mouth, 3-foot 5-inch bed and cut).	268	W19699	A	1	2.7	16.2	75.8	5.3	.75						8,078	14,540
				2		16.7	77.9	5.4	.80						8,300	14,940
				3		17.7	82.3		.85						8,778	15,800
Same (pillar between rooms 21 and 22, off heading 5, 3,200 feet north of drift mouth, 3-foot 6¼-inch bed and cut).	268	W19701	A	1	2.3	16.8	74.7	6.2	1.30						8,083	14,550
				2		17.2	76.5	6.3	1.35						8,272	14,890
				3		18.4	81.6		1.44						8,828	15,890
Same (composite of samples W19695, W19696, W19698, W19699, and W19701).	268	12222	A	1	1.57	16.13	76.31	5.99	1.09	4.56	83.76	1.32	3.28		8,106	14,591
				2		16.39	77.52	6.09	1.11	4.46	85.09	1.34	1.91		8,235	14,823
				3		17.45	82.55		1.18	4.75	90.60	1.43	2.04		8,769	15,784
Carrolltown Road, ½ mile west of; Logan No. 5 mine, Miller bed (in first main heading, 4,300 feet south of mine mouth, 2-foot 6½-inch bed and cut).	269	W20364	A	1	2.2	19.8	71.7	6.3	1.00						7,972	14,350
				2		20.2	73.4	6.4	1.05						8,150	14,670
				3		21.6	78.4		1.12						8,706	15,670
Same (face of right entry 2; 250 feet from second main heading and 3,750 feet south of mine mouth, 2-foot 5⅞-inch bed and cut).	269	W20365	A	1	2.2	22.4	68.6	6.8	1.20						7,922	14,260
				2		22.9	70.1	7.0	1.25						8,100	14,580
				3		24.6	75.4		1.34						8,706	15,670
Same (face of room 8, off air course of right entry 1, off second main entry, 4,300 feet south of mine mouth, 2-foot 6⅛-inch bed and cut).	269	W20366	A	1	1.6	22.7	68.6	7.1	1.15						7,939	14,290
				2		23.1	69.7	7.2	1.20						8,072	14,530
				3		24.9	75.1		1.29						8,700	15,660
Same (face of second main heading, 3,600 feet southeast of mine mouth, 2-foot 4½-inch bed and cut).	269	W20367	A	1	2.2	23.0	67.8	7.0	1.35						7,989	14,380
				2		23.5	69.4	7.1	1.40						8,167	14,700
				3		25.3	74.7		1.51						8,789	15,820
Same (face of fourth left heading, off second main heading, 3,200 feet southeast of mine mouth, 2-foot 6¼-inch bed and cut).	269	W20368	A	1	2.0	22.4	70.0	5.6	1.30						8,078	14,540
				2		22.9	71.4	5.7	1.35						8,244	14,840
				3		24.3	75.7		1.43						8,744	15,740
Same (composite of samples W20364-W20368)	269	12370	A	1	.93	23.10	69.29	6.68	1.30	4.81	81.64	1.26	4.31		8,047	14,485
				2		23.32	69.94	6.74	1.31	4.75	82.41	1.27	3.52		8,123	14,621
				3		25.01	74.99		1.40	5.09	88.37	1.36	3.78		8,710	15,678
Colver; Colver mine, B bed (face of main entry A, 2,400 feet southwest of bore hole, 3-foot 11-inch bed, 3-foot 7¾-inch cut).	269	13634	A	1	[a] 2.24	22.04	69.08	6.64	1.76					1.1	8,001	14,402
				2		22.54	70.67	6.79	1.80						8,184	14,731
				3		24.18	75.82		1.93						8,780	15,804
Same (face of entry A-2, 1,175 feet southeast of bore hole 25, 4-foot 1½-inch bed, 3-foot 10-inch cut).	269	13635	A	1	[a] 1.96	20.77	71.28	5.99	1.01					.7	8,065	14,517
				2		21.19	72.70	6.11	1.03						8,226	14,807
				3		22.57	77.43		1.10						8,762	15,772
Same (face of entry A-3, 1,400 feet south of bore hole 25, 4-foot 2-inch bed, 3-foot 10½-inch cut).	269	13636	A	1	[a] 2.02	21.85	68.85	7.28	1.42					.8	7,937	14,287
				2		22.30	70.27	7.43	1.45						8,101	14,582
				3		24.09	75.91		1.57						8,752	15,754
Same (face of main heading, 1,200 feet southeast of bore hole 25, 4-foot 3-inch bed, 3-foot 10-inch cut).	269	13637	A	1	[a] 2.77	20.58	71.74	4.91	.92					1.1	8,104	14,587
				2		21.17	73.78	5.05	.95						8,335	15,003
				3		22.30	77.70		1.00						8,778	15,800
Same (face of entry B-1, 725 feet southeast of bore hole 25, 3-foot 10-inch bed, 3-foot 6-inch cut).	269	13638	A	1	[a] 2.16	20.66	72.04	5.14	.97					.9	8,126	14,627
				2		21.12	73.63	5.25	.99						8,306	14,951
				3		22.29	77.71		1.04						8,766	15,779
Same (face of entry A-1, 950 feet southeast of bore hole 25, 4-foot 1-inch bed, 3-foot 7¾-inch cut).	269	13639	A	1	[a] 1.89	20.92	71.82	5.37	.95					.7	8,120	14,616
				2		21.32	73.21	5.47	.97						8,277	14,899
				3		22.55	77.45		1.03						8,756	15,761
Same (composite of samples 13634-13639)	270	13640	A	1	2.00	21.17	70.90	5.93	1.23	4.64	81.57	1.57	5.06	.9	8,062	14,512
				2		21.60	72.35	6.05	1.26	4.51	83.23	1.60	3.35		8,226	14,807
				3		22.99	77.01		1.34	4.80	88.59	1.70	3.57		8,756	15,761

[a] Sample air dried a few hours only, does not represent the total moisture as received.

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Colver; Colver mine, B bed (position, bed, and cut same as sample 13639).	269	13641	A	1	[a] 1.93	20.93	71.91	5.23	1.02	....	....	....	....	1.5	....	....
				2	....	21.34	73.33	5.33	1.04	....	....	....	....	....	....	....
				3	....	22.54	77.46	....	1.10	....	....	....	....	....	....	....
Same (position, bed, and cut same as sample 13638).	269	13642	A	1	[a] 3.27	20.04	71.50	5.19	1.05	....	....	....	....	2.8	....	....
				2	....	20.72	73.91	5.37	1.09	....	....	....	....	....	....	....
				3	....	21.89	78.11	....	1.15	....	....	....	....	....	....	....
Same (position, bed, and cut same as sample 13637).	269	13643	A	1	[a] 4.66	19.73	70.91	4.70	.88	....	....	....	....	4.2	....	....
				2	....	20.69	74.38	4.93	.92	....	....	....	....	....	....	....
				3	....	21.76	78.24	....	.97	....	....	....	....	....	....	....
Dunlo, one mile south of; Henriette mine, Miller or B bed (rib of room 2, off right heading 1, off slope 2,700 feet north of shaft No. 2, 3-foot 9½-inch bed, 3-foot 5½-inch cut).	270	14238	A	1	2.34	18.70	74.24	4.72	.60	....	....	....	....	1.3	....	....
				2	....	19.15	76.02	4.83	.61	....	....	....	....	....	....	....
				3	....	20.12	79.88	....	.64	....	....	....	....	....	....	....
Same (junction of sixth slant and main heading, on curve, 3,050 feet north of shaft No. 2, 3-foot 10-inch bed, 3-foot 7-inch cut).	270	14239	A	1	2.56	18.57	73.86	5.01	.75	....	....	....	....	1.4	....	....
				2	....	19.06	75.80	5.14	.77	....	....	....	....	....	....	....
				3	....	20.09	79.91	....	.81	....	....	....	....	....	....	....
Same (rib of pillar of left entry 6, near room 13, north of rock fault, 4,350 feet northwest of shaft No. 2, 3-foot 6½-inch bed, 3-foot 4½-inch cut).	270	14240	A	1	3.01	17.45	74.94	4.60	.60	....	....	....	....	1.9	....	....
				2	....	17.99	77.27	4.74	.62	....	....	....	....	....	....	....
				3	....	18.89	81.11	....	.65	....	....	....	....	....	....	....
Same (pillar of left heading 6, 450 feet north of back heading off slope 2, 4,150 feet northwest of shaft No. 2, 3-foot 7½-inch bed, 3-foot 4-inch cut).	270	14241	A	1	2.77	17.41	75.19	4.63	.68	....	....	....	....	1.8	....	....
				2	....	17.91	77.33	4.76	.70	....	....	....	....	....	....	....
				3	....	18.81	81.19	....	.74	....	....	....	....	....	....	....
Same (rib between rooms 1 and 2, first level above main entry, about 1,500 feet northwest of shaft No. 2, 3-foot 8-inch cut).	270	14242	A	1	2.82	18.42	74.25	4.51	.59	....	....	....	....	1.8	....	....
				2	....	18.95	76.41	4.64	.61	....	....	....	....	....	....	....
				3	....	19.87	80.13	....	.64	....	....	....	....	....	....	....
Same (composite of samples 14238-14242).........	271	14243	A	1	2.65	18.08	74.64	4.63	.62	4.55	83.80	1.35	5.05	1.6	8,089	14,560
				2	....	18.57	76.67	4.76	.64	4.38	86.08	1.39	2.75	....	8,309	14,956
				3	....	19.50	80.50	....	.67	4.60	90.38	1.46	2.89	....	8,724	15,703
Same (position and bed same as sample 14240, 2 inches of top coal or bone).	270	14244	A	1	2.84	17.06	75.19	4.91	.53	....	....	....	....	1.7	....	....
				2	....	17.56	77.39	5.05	.55	....	....	....	....	....	....	....
				3	....	18.49	81.51	....	.58	....	....	....	....	....	....	....

Same (position and bed same as sample 14238, 4 inches of top coal or bone).	270	14245	A	1	2.39	19.11	74.89	3.61	.55	....	....	....	....	1.5	....	....
				2	....	19.58	76.72	3.70	.56	....	....	....	....	....	....	....
				3	....	20.33	79.67	....	.58	....	....	....	....	....	....	....
Same (position and bed same as sample 14239, 3 inches of top coal or bone).	270	14246	A	1	1.80	18.60	69.71	9.89	2.07	....	....	....	....	.8	....	....
				2	....	18.94	70.99	10.07	2.11	....	....	....	....	....	....	....
				3	....	21.06	78.94	....	2.35	....	....	....	....	....	....	....
El Mora, near; Peerless No. 1 mine, Lower Kittanning or B bed (heading stump on left entry 3, off main entry, about 1,700 feet northeast of drift mouth, 3-foot 3¾-inch bed and cut).	271	12134	A	1	3.18	20.91	69.73	6.18	1.97	....	....	....	....	2.6	7,939	14,290
				2	....	21.60	72.02	6.38	2.03	....	....	....	....	....	8,199	14,758
				3	....	23.07	76.93	....	2.17	....	....	....	....	....	8,757	15,763
Same (face of room 1, right heading 8, off main entry, 2,300 feet east of drift mouth, 3-foot ½-inch bed and cut).	271	12135	A	1	3.55	21.42	68.06	6.97	1.31	....	....	....	....	2.9	7,840	14,112
				2	....	22.21	70.56	7.23	1.36	....	....	....	....	....	8,129	14,632
				3	....	23.94	76.06	....	1.47	....	....	....	....	....	8,762	15,772
Same (face of left heading 7, off main entry, about 2,800 feet northeast of drift mouth, 2-foot 11½-inch bed, 2-foot 11-inch cut).	271	12136	A	1	3.28	21.07	69.19	6.46	1.43	....	....	....	....	2.5	7,953	14,315
				2	....	21.78	71.54	6.68	1.48	....	....	....	....	....	8,223	14,801
				3	....	23.34	76.66	....	1.59	....	....	....	....	....	8,812	15,862
Same (rib of room 1, right heading 3, off main heading, 1,600 feet southeast of drift mouth, 2-foot 11¾-inch bed and cut).	271	12137	A	1	3.59	20.43	68.79	7.19	1.95	....	....	....	....	2.7	7,875	14,175
				2	....	21.19	71.35	7.46	2.02	....	....	....	....	....	8,168	14,702
				3	....	22.90	77.10	....	2.18	....	....	....	....	....	8,826	15,887
Same (425 feet in heading 1, off main entry, 2-foot 11½-inch bed and cut).	271	12138	A	1	3.92	20.85	69.19	6.04	.90	....	....	....	....	3.3	7,809	14,056
				2	....	21.70	72.01	6.29	.94	....	....	....	....	....	8,128	14,630
				3	....	23.16	76.84	....	1.00	....	....	....	....	....	8,673	15,611
Same (composite of samples 12134–12138).........	272	12139	A	1	3.47	20.73	69.39	6.41	1.55	4.88	80.02	1.36	5.78	2.8	7,874	14,173
				2	....	21.47	71.89	6.64	1.61	4.65	82.89	1.41	2.80	....	8,157	14,683
				3	....	23.00	77.00	....	1.72	4.98	88.78	1.51	3.01	....	8,737	15,727
Figart, 1½ miles northeast of; Frick No. 2 mine, Middle Kittanning or C bed (face of main heading 2, 400 feet from mouth of main entry, 3-foot 2½-inch bed, 2-foot 2¾-inch cut).	272	W20433	A	1	2.2	24.6	64.2	9.0	1.15	....	....	....	....	....	7,694	13,850
				2	....	25.1	65.7	9.2	1.20	....	....	....	....	....	7,867	14,160
				3	....	27.6	72.4	....	1.32	....	....	....	....	....	8,667	15,600
Same (face of entry 1, off second main heading, 400 feet from mouth of main entry, 3-foot 1-inch bed, 2-foot 3½-inch cut).	272	W20434	A	1	2.4	23.2	64.5	9.9	1.35	....	....	....	....	....	7,617	13,710
				2	....	23.8	66.1	10.1	1.40	....	....	....	....	....	7,800	14,040
				3	....	26.5	73.5	....	1.56	....	....	....	....	....	8,672	15,610
Same (face of room 5, off main left heading, 850 feet from mouth of main entry, 3-foot 2½-inch bed, 2-foot ¾-inch cut).	272	W20474	A	1	2.6	24.1	66.6	6.7	1.00	....	....	....	....	....	7,861	14,150
				2	....	24.8	68.3	6.9	1.05	....	....	....	....	....	8,072	14,530
				3	....	26.6	73.4	....	1.13	....	....	....	....	....	8,667	15,600
Same (face of first left heading, 500 feet from mouth of main entry, 3-foot 1-inch bed, 2-foot 4-inch cut).	272	W20475	A	1	2.9	23.4	65.2	8.5	.95	....	....	....	....	....	7,694	13,850
				2	....	24.1	67.1	8.8	1.00	....	....	....	....	....	7,928	14,270
				3	....	26.4	73.6	....	1.10	....	....	....	....	....	8,694	15,650
Same (face of straight main heading, 1,200 feet from mouth of entry, 2-foot 11½-inch bed, 1-foot 11½-inch cut).	272	W20476	A	1	2.6	23.9	67.3	6.2	.85	....	....	....	....	....	7,917	14,250
				2	....	24.5	69.1	6.4	.85	....	....	....	....	....	8,133	14,640
				3	....	26.2	73.8	....	.91	....	....	....	....	....	8,689	15,640
Same (composite of samples W20433, W20434, W20474, W20475, and W20476).	272	12366	A	1	1.11	25.68	64.99	8.22	1.18	4.73	79.71	1.26	4.90	....	7,818	14.072
				2	....	25.97	65.72	8.31	1.19	4.66	80.60	1.27	3.97	....	7,906	14,231
				3	....	28.32	71.68	....	1.30	5.08	87.90	1.39	4.33	....	8,622	15,520

[a] Sample air dried the usual length of time, represents true moisture as received.

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Hastings, 2½ miles northeast of; Miller Run mine, Miller or B bed (face of main air course, 2,000 feet from mine mouth, 4-foot 1-inch bed, 3-foot 3-inch cut).	273	W20440	A	1	3.0	22.4	66.3	8.3	1.40	......	......	......	......	......	7,694	13,850
				2	......	23.1	68.3	8.6	1.45	......	......	......	......	......	7,933	14,280
				3	......	25.3	74.7	......	1.59	......	......	......	......	......	8,683	15,630
Same (face of room 4, off right entry 5; 200 feet from main entry), 4-foot 1¼-inch bed, 3-foot 3¼-inch cut).	273	W20441	A	1	2.1	21.9	66.5	9.5	1.75	......	......	......	......	......	7,644	13,760
				2	......	22.4	67.9	9.7	1.80	......	......	......	......	......	7,811	14,060
				3	......	24.8	75.2	......	1.99	......	......	......	......	......	8,650	15,570
Same (face of right entry 1; 200 feet off left entry 2, 4-foot 1-inch bed, 2-foot 8½-inch cut).	273	W20442	A	1	2.3	22.4	67.0	8.3	.85	......	......	......	......	......	7,806	14,050
				2	......	22.9	68.6	8.5	.90	......	......	......	......	......	7,989	14,380
				3	......	25.0	75.0	......	.98	......	......	......	......	......	8,733	15,720
Same (face of left entry 2; 500 feet off main entry, 4-foot 1⅜-inch bed, 3-foot 4⅜-bed cut).	273	W20443	A	1	2.1	23.5	64.7	9.7	1.25	......	......	......	......	......	7,644	13,760
				2	......	24.0	66.1	9.9	1.30	......	......	......	......	......	7,811	14,060
				3	......	26.6	73.4	......	1.44	......	......	......	......	......	8,667	15,600
Same (composite of sample W20440-W20443).....	273	12371	A	1	.81	23.92	66.15	9.12	1.36	4.67	79.15	1.26	4.44	......	7,800	14,040
				2	......	24.12	66.69	9.19	1.37	4.62	79.80	1.27	3.75	......	7,864	14.155
				3	......	26.56	73.44	......	1.51	5.09	87.88	1.40	4.12	......	8,660	15,588
Same (pillar of room 5, left entry 1; 300 feet off main entry, 3-foot 11½-inch bed, 3-foot 3½-inch cut).	273	W20447	A	1	2.4	21.0	69.9	6.7	.85	......	......	......	......	......	7,906	14,230
				2	......	21.5	71.6	6.9	.90	......	......	......	......	......	8,094	14,570
				3	......	23.1	76.9	......	.97	......	......	......	......	......	8,694	15,650
1 mile south of; Pennsylvania No. 11 mine, Lower Freeport or D bed (face of right entry 5, off main heading, 5-foot 3⅝-inch bed, 4-foot 10⅞-inch cut).	274	12150	A	1	2.78	23.49	67.61	6.12	1.16	......	......	......	......	2.2	7,945	14,301
				2	......	24.16	69.54	6.30	1.19	......	......	......	......	......	8,172	14,710
				3	......	25.78	74.22	......	1.27	......	......	......	......	......	8,721	15,698
Same (face of old main heading, 4-foot 8¼-inch bed, 4-foot 4¾-inch cut).	274	12151	A	1	2.81	23.54	66.65	7.00	1.25	......	......	......	......	2.2	7,856	14,141
				2	......	24.22	68.58	7.20	1.29	......	......	......	......	......	8,063	14,549
				3	......	26.10	73.90	......	1.39	......	......	......	......	......	8,710	15,678
Same (face of second right cross heading, 4-foot ¼-inch bed, 3-foot 9-inch cut).	274	12166	A	1	2.88	23.92	66.34	6.86	1.33	......	......	......	......	2.2	7,845	14,121
				2	......	24.63	68.31	7.06	1.37	......	......	......	......	......	8,078	14,540
				3	......	26.50	73.50	......	1.47	......	......	......	......	......	8,692	15,646
Same (face of left heading 3, 3-foot 8-inch bed, 3-foot 4½-inch cut).	274	12167	A	1	3.26	23.50	64.78	8.46	1.73	......	......	......	......	2.7	7,644	13,759
				2	......	24.29	66.96	8.75	1.79	......	......	......	......	......	7,902	14,224
				3	......	26.62	73.38	......	1.96	......	......	......	......	......	8,660	15,588

Same (composite of samples 12150, 12151, 12166 and 12167).	274	12168	A	1	2.89	23.67	66.34	7.10	1.37	5.02	79.49	1.30	5.72	2.3	7.837	14,107
				2	....	24.38	68.31	7.31	1.41	4.84	81.86	1.34	3.24	....	8.071	14,528
				3	....	26.30	73.70	....	1.52	5.22	88.32	1.45	3.49	....	8,708	15,674
Same (pillar in room 16, off left entry 3, 4-foot 4½-inch bed, 4-feet 4½-inch cut).	274	12169	A	1	3.42	23.16	67.27	6.15	1.62	5.10	79.55	1.28	6.30	2.8	7,839	14,110
				2	....	23.98	69.65	6.37	1.68	4.89	82.37	1.33	3.36	....	8,117	14,611
				3	....	25.61	74.39	....	1.79	5.22	87.97	1.42	3.60	....	8,669	15,604
1 mile south of; Pennsylvania No. 12 mine, Lower Freeport or D bed (face of original old main heading, 5,800 feet from mine mouth, 3-foot 1½-inch bed, 3-foot cut).	274	12152	A	1	2.28	25.33	64.46	7.93	2.14	....	....	....	....	1.6	7,777	13,999
				2	....	25.92	65.97	8.11	2.19	....	....	....	....	....	7,958	14.324
				3	....	28.21	71.79	....	2.38	....	....	....	....	....	8,661	15,590
Same (face of entry 5, off main heading, 4-foot ⅝-inch bed, 3-foot 10⅝-inch cut).	274	12153	A	1	3.00	22.49	67.52	6.99	2.02	....	....	....	....	2.3	7,828	14,090
				2	....	23.18	69.61	7.21	2.08	....	....	....	....	....	8,070	14,526
				3	....	24.98	75.02	....	2.24	....	....	....	....	....	8,697	15,655
Same (face of right heading 4, 1½ miles from mine mouth, 3-foot 11½-inch bed, 3-foot 8½-inch cut).	274	12154	A	1	2.55	24.09	66.39	6.97	1.56	....	....	....	....	1.9	7,861	14,150
				2	....	24.72	68.13	7.15	1.60	....	....	....	....	....	8,067	14,521
				3	....	26.62	73.38	....	1.72	....	....	....	....	....	8,688	15,638
Same (face of room 1, off entry 1, off right heading 3, 4-foot 4½-inch bed, 4-foot 2½-inch cut).	274	12155	A	1	2.63	23.31	66.07	7.99	1.58	....	....	....	....	2.2	7.750	13,950
				2	....	23.94	67.85	8.21	1.62	....	....	....	....	....	7,959	14,326
				3	....	26.08	73.92	....	1.76	....	....	....	....	....	8,671	15,608
Same (face of right entry 3, off main heading, 4-foot 3-inch bed, 3-foot 9-inch cut).	274	12156	A	1	2.78	22.50	67.47	7.25	1.74	....	....	....	....	2.1	7,821	14,078
				2	....	23.14	69.40	7.46	1.79	....	....	....	....	....	8,045	14,481
				3	....	25.01	74.99	....	1.93	....	....	....	....	....	8,693	15,647
Same (composite of samples 12152–12156).........	274	12157	A	1	2.73	23.61	66.18	7.48	1.84	4.88	78.80	1.31	5.69	2.0	7,797	14,035
				2	....	24.27	68.04	7.69	1.89	4.71	81.01	1.35	3.35	....	8,016	14,429
				3	....	26.29	73.71	....	2.05	5.10	87.76	1.46	3.63	....	8,684	15,631
Johnstown, 3 miles south of; Valley No. 1 mine, Upper Kittanning or C bed (face of room 11 in Andy's heading, 1,000 feet west of tipple No. 2, 4-foot bed, 3-foot 11¾-inch cut).	275	W19563	A	1	2.0	14.0	73.6	10.4	2.05	....	....	....	....	....	7,667	13,800
				2	....	14.3	75.1	10.6	2.10	....	....	....	....	....	7,817	14,070
				3	....	16.0	84.0	....	2.35	....	....	....	....	....	8,744	15,740
Same (face of room 21, left heading 7, 3,400 feet southeast of mine mouth, 4-foot 2-inch bed and cut).	275	W19564	A	1	1.9	14.0	74.8	9.3	2.25	....	....	....	....	....	7,733	13,920
				2	....	14.3	76.2	9.5	2.30	....	....	....	....	....	7.883	14,190
				3	....	15.8	84.2	....	2.54	....	....	....	....	....	8.711	15,680
Same (face of room 4, right heading 8, 3,600 feet southeast of mine mouth, 4-foot 1¾-inch cut.	275	W19566	A	1	1.8	14.5	76.2	7.5	.70	....	....	....	....	....	7,950	14,310
				2	....	14.8	77.6	7.6	.70	....	....	....	....	....	8,100	14,580
				3	....	16.0	84.0	....	.76	....	....	....	....	....	8,767	15,780
Same (rib of main air course, 150 feet beyond right heading 12, 4-foot 4½-inch bed, 4-foot 3⅞-inch cut).	275	W19567	A	1	2.4	14.4	75.3	7.9	1.00	....	....	....	....	....	7,861	14,150
				2	....	14.7	77.2	8.1	1.05	....	....	....	....	....	8,050	14,490
				3	....	16.0	84.0	....	1.14	....	....	....	....	....	8,761	15,770
Same (composite of samples W19563, W19564, W19566, and W19567).	275	12213	A	1	1.32	14.63	75.24	8.81	1.57	4.26	81.19	1.39	2.78	....	7,804	14,047
				2	....	14.83	76.24	8.93	1.59	4.17	82.28	1.41	1.62	....	7,909	14,236
				3	....	16.28	83.72	....	1.75	4.58	90.35	1.55	1.77	....	8,685	15,633
Same (chain pillar, 4,200 feet northeast of mine mouth, 3-foot 9⅜-inch bed and cut).	275	W19565	A	1	2.3	14.3	74.6	8.8	2.45	....	....	....	....	....	7,739	13,930
				2	....	14.6	76.4	9.0	2.50	....	....	....	....	....	7,922	14,260
				3	....	16.0	84.0	....	2.75	....	....	....	....	....	8,706	15,670

Table of chemical analyses—Continued.

Locality, bed. etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Nanty Glo, 1 mile north of; Lincoln No. 1 mine; Lower Kittanning, Miller, or B bed (face of new main heading, 3-foot 4-inch bed and cut).	276	12180	A	1	3.03	19.35	71.18	6.44	1.84	......	......	......	......	2.4	7,936	14,285
				2	......	19.95	73.41	6.64	1.90	......	......	......	......	......	8,184	14,731
				3	......	21.37	78.63	......	2.04	......	......	......	......	......	8,766	15,779
Same (face of right entry 2, 3-foot 5-inch bed and cut).	276	12181	A	1	3.28	18.01	72.23	6.48	1.70	......	......	......	......	2.6	7,902	14,224
				2	......	18.62	74.68	6.70	1.76	......	......	......	......	......	8,170	14,706
				3	......	19.96	80.04	......	1.89	......	......	......	......	......	8,757	15,763
Same (face of right entry 4, 3-foot 4-inch bed and cut).	276	12182	A	1	2.84	19.60	71.48	6.08	1.68	......	......	......	......	2.4	7,980	14,364
				2	......	20.17	73.57	6.26	1.73	......	......	......	......	......	8,213	14,783
				3	......	21.52	78.48	......	1.85	......	......	......	......	......	8,762	15,772
Same (face of left entry 7, 3-foot 6-inch bed and cut).	276	12183	A	1	2.93	19.69	71.41	5.97	1.58	......	......	......	......	2.5	7,971	14,348
				2	......	20,28	73.57	6.15	1.63	......	......	......	......	......	8,212	14,782
				3	......	21.61	78.39	......	1.74	......	......	......	......	......	8,750	15,750
Same (face of left entry 9, 3-foot 5-inch bed and cut).	276	12184	A	1	2.74	19.07	71.20	6.99	2.21	......	......	......	......	2.3	7,881	14,186
				2	......	19.61	73.21	7.18	2.27	......	......	......	......	......	8,103	14,585
				3	......	21.13	78.87	......	2.45	......	......	......	......	......	8,730	15,714
Same (composite of samples 12180, 12181, 12182, 12183, and 12184).	276	12185	A	1	2.84	19.78	70.89	6.49	1.85	4.87	80.83	1.32	4.64	2.4	7,936	14,285
				2	......	20.36	72.96	6.68	1.90	4.68	83.19	1.36	2.19	......	8,168	14,702
				3	......	21.82	78.18	......	2.04	5.02	89.15	1.46	2.33	......	8,753	15,755
One-half mile southeast of; Springfield No. 1 mine, Lower Kittanning or B bed (face of left entry 4, 2,700 feet from mine mouth, 3-foot 4-inch bed and cut).	276	12128	A	1	3.31	17.71	71.87	7.11	2.00	......	......	......	......	2.8	7,841	14,114
				2	......	18.32	74.33	7.35	2.07	......	......	......	......	......	8,109	14,596
				3	......	19.77	80.23	......	2.23	......	......	......	......	......	8,753	15,755
Same (face of main entry, 3,500 feet from mine mouth, 3-foot 8-inch bed and cut).	276	12129	A	1	4.31	16.57	72.57	6.55	2.31	......	......	......	......	3.7	7,795	14,031
				2	......	17.32	75.83	6.85	2.41	......	......	......	......	......	8,146	14,663
				3	......	18.59	81.41	......	2.59	......	......	......	......	......	8,745	15,741
Same (face of third crosscut off dip heading, 2,500 feet from mine mouth, 3-foot 7-inch bed and cut).	276	12130	A	1	3.09	17.11	73.28	6.52	2.35	......	......	......	......	2.5	7,901	14,222
				2	......	17.66	75.61	6.73	2.43	......	......	......	......	......	8,153	14,675
				3	......	18.94	81.06	......	2.61	......	......	......	......	......	8,741	15,734
Same (face of left entry 2, 3,000 feet from mine mouth, 3-foot 8-inch bed and cut).	276	12131	A	1	2.98	20.78	69.93	6.31	1.88	......	......	......	......	2.4	7,925	14,265
				2	......	21.42	72.08	6.50	1.94	......	......	......	......	......	8,168	14,702
				3	......	22.91	77.09	......	2.07	......	......	......	......	......	8,736	15,725

Same (composite of samples 12128, 12129, 12130, 12131).	277	12132	A	1	3.48	17.97	71.78	6.82	2.22	4.74	80.12	1.20	4.90	2.9	7,857	14,143
				2	......	18.62	74.31	7.07	2.30	4.51	83.01	1.24	1.87	......	8,141	14,654
				3	......	20.04	79.96	......	2.48	4.86	89.33	1.33	2.00	......	8,761	15,770
Same (pillar in room 3, off left entry 2, 3-foot 9-inch bed and cut).	276	12133	A	1	3.19	17.22	73.83	5.76	1.97	4.79	81.30	1.21	4.97	2.5	7,991	14,384
				2	......	17.79	76.26	5.95	2.04	4.59	83.98	1.25	2.19	......	8,255	14,859
				3	......	18.92	81.08	......	2.17	4.88	89.30	1.33	2.32	......	8,778	15,800
Portage, 2 miles north of; Forge slope No. 1 mine, Upper Kittanning or C′ bed (face of main south heading, 1 mile south of mine mouth, 3-foot 4¾-inch bed and cut).	277	W20396	A	1	2.0	16.3	74.1	7.6	.55	......	......	......	......	......	7,872	14,170
				2	......	16.6	75.6	7.8	.55	......	......	......	......	......	8,033	14,460
				3	......	18.0	82.0	......	.60	......	......	......	......	......	8,711	15,680
Same (on rib, near face of left entry 7, off main south entry, 1 mile west of mine mouth, 3-foot 5-inch bed and cut).	277	W20397	A	1	1.8	18.3	70.0	9.9	2.55	......	......	......	......	......	7,656	13,780
				2	......	18.6	71.3	10.1	2.60	......	......	......	......	......	7,794	14,030
				3	......	20.7	79.3	......	2.89	......	......	......	......	......	8,672	15,610
Same (on rib, near face of right entry 2, off left entry 7, off main south entry, 4,500 feet northwest of mine mouth, 3-foot 5½-inch bed and cut).	277	W20398	A	1	1.8	18.1	70.8	9.3	2.60	......	......	......	......	......	7,722	13,900
				2	......	18.4	72.1	9.5	2.65	......	......	......	......	......	7,867	14,160
				3	......	20.3	79.7	......	2.93	......	......	......	......	......	8,694	15,650
Same (on rib, near face of right entry 1, off left entry 7, off main south heading, 5,000 feet southwest of mine mouth, 3-foot 7¾-inch bed and cut).	277	W20399	A	1	2.3	15.1	73.5	9.1	2.25	......	......	......	......	......	7,717	13,890
				2	......	15.5	75.2	9.3	2.30	......	......	......	......	......	7,900	14,220
				3	......	17.1	82.9	......	2.54	......	......	......	......	......	8,711	15,680
Same (face of main north heading, 3,000 feet north of mine mouth, 3-foot 5¾-inch bed and cut).	277	W20400	A	1	1.9	16.7	73.4	8.0	1.40	......	......	......	......	......	7,828	14,090
				2	......	17.0	74.8	8.2	1.45	......	......	......	......	......	7,978	14,360
				3	......	18.5	81.5	......	1.58	......	......	......	......	......	8,694	15,650
Same (on rib near face of right entry 3, off main north entry, 2,600 feet north of mine mouth, 3-foot 7-inch bed and cut).	277	W20401	A	1	2.1	17.0	72.5	8.4	1.00	......	......	......	......	......	7,772	13,990
				2	......	17.4	74.0	8.6	1.06	......	......	......	......	......	7,939	14,290
				3	......	19.0	81.0	......	1.16	......	......	......	......	......	8,683	15,630
Same (composite of samples W20396–W20401)....	278	12369	A	1	.94	18.41	71.73	8.92	1.86	4.36	80.10	1.23	3.53	......	7,823	14,081
				2	......	18.58	72.42	9.00	1.88	4.30	80.86	1.24	2.72	......	7,897	14,215
				3	......	20.42	79.58	......	2.07	4.73	88.86	1.36	2.98	......	8,678	15,620
1 mile east of; Miller No. 1 shaft; Lower Kittanning, Miller, or B bed (face of second right dip, 2,800 feet southwest of shaft, 3-foot 2-inch bed, 2-foot 9½-inch cut).	278	11355	A	1	2.94	15.81	75.92	5.33	.70	......	......	......	......	2.4	......	......
				2	......	16.29	78.22	5.49	.72	......	......	......	......	......	......	......
				3	......	17.24	82.76	......	.76	......	......	......	......	......	......	......
Same (face of left heading 3, 4,100 feet southeast of shaft, 3-foot 11¾-inch bed, 3-foot 7¾-inch cut).	278	11356	A	1	2.91	17.58	72.69	6.82	1.68	......	......	......	......	2.4	......	......
				2	......	18.11	74.87	7.02	1.73	......	......	......	......	......	......	......
				3	......	19.48	80.52	......	1.86	......	......	......	......	......	......	......
Same (pillar of A-1 heading, 2,000 feet from foot of shaft, 3-foot 5-inch bed, 3-foot 2½-inch cut).	278	11357	A	1	1.85	19.83	71.53	6.79	1.49	......	......	......	......	1.4	......	......
				2	......	20.20	72.88	6.92	1.52	......	......	......	......	......	......	......
				3	......	21.70	78.30	......	1.63	......	......	......	......	......	......	......
Same (face of first right dip, 1,500 feet from foot of shaft, 3-foot 10⅝-inch bed, 3-foot 4⅝-inch cut).	278	11358	A	1	2.35	17.47	74.09	6.09	.73	......	......	......	......	1.8	......	......
				2	......	17.89	75.87	6.24	.75	......	......	......	......	......	......	......
				3	......	19.08	80.92	......	.80	......	......	......	......	......	......	......
Same (face of left heading 1, 3,000 feet from foot of shaft, 3-foot 6-inch bed, 3-foot ½-inch cut).	278	11359	A	1	3.43	17.69	72.25	6.63	1.41	......	......	......	......	2.9	......	......
				2	......	18.32	74.81	6.87	1.46	......	......	......	......	......	......	......
				3	......	19.67	80.33	......	1.57	......	......	......	......	......	......	......
Same (composite of samples 11355–11359).......	278	11360	A	1	2.87	17.63	73.21	6.29	1.22	4.51	81.69	1.23	5.06	2.2	7,940	14,292
				2	......	18.15	75.37	6.48	1.26	4.31	84.10	1.27	2.58	......	8,176	14,713
				3	......	19.41	80.59	......	1.35	4.61	89.93	1.36	2.75	......	8,740	15,732

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Portage, 2½ miles east of; Plymouth No. 1 mine, Upper Kittanning or C' bed (face of dip 5, 300 feet from main heading, 3-foot 7½-inch bed, 3-foot 7-inch cut).	279	W17110	A	1	3.3	17.5	72.6	6.6	.70	......	......	......	......	......	7,861	14,150
				2	......	18.0	75.2	6.8	.75	......	......	......	......	......	8,133	14,640
				3	......	19.3	80.7	......	.80	......	......	......	......	......	8,728	15,710
Same (face of left entry 4, 2,000 feet from main heading, 3-foot 8¼-inch bed, 3-foot 8-inch cut).	279	W17111	A	1	3.5	17.5	70.7	8.3	1.50	......	......	......	......	......	7,694	13,850
				2	......	18.1	73.3	8.6	1.60	......	......	......	......	......	7,972	14,350
				3	......	19.8	80.2	......	1.75	......	......	......	......	......	8,722	15,700
Same (face of main heading, 250 feet from left entry 8, 3-foot ½-inch cut).	279	W17112	A	1	3.3	16.5	72.9	7.3	.70	......	......	......	......	......	7,817	14,070
				2	......	17.0	75.5	7.5	.70	......	......	......	......	......	8,083	14,550
				3	......	18.4	81.6	......	.76	......	......	......	......	......	8,739	15,730
Same (composite samples of W17110, W17111, and W17112).	279	11705	A	1	1.95	......	......	7.60	1.05	4.45	78.35	1.18	7.37	......	......	......
				2	......	......	......	7.75	1.07	4.31	79.91	1.20	5.76	......	......	......
				3	......	......	......	......	1.16	4.67	86.62	1.30	6.25	......	......	......
1 mile east of; Sonman No. 2 shaft, Lower Kittanning or B bed (pillar near face of room 23, first right off plane, 3,000 feet southeast of shaft, 3-foot 11-inch bed, 3-foot 7-inch cut).	280	W19689	A	1	2.5	15.3	76.0	6.2	.65	......	......	......	......	......	8,039	14,470
				2	......	15.7	77.9	6.4	.65	......	......	......	......	......	8,239	14,830
				3	......	16.8	83.2	......	.69	......	......	......	......	......	8,800	15,840
Same (face of "No. 32 crook," off main north heading, 4-foot 3¾-inch bed, 3-foot 7¼-inch cut).	280	W19691	A	1	2.6	16.3	74.6	6.5	1.65	......	......	......	......	......	7,956	14,320
				2	......	16.7	76.6	6.7	1.70	......	......	......	......	......	8,172	14,710
				3	......	17.9	82.1	......	1.82	......	......	......	......	......	8,756	15,760
Same (face of "No. 28 crook," left heading 1, off dip, off main south heading, 3-foot 11¾-inch bed, 3-foot 4¼-inch cut).	280	W19692	A	1	2.5	15.9	76.1	5.5	.65	......	......	......	......	......	8,083	14,550
				2	......	16.3	78.1	5.6	.65	......	......	......	......	......	8,286	14,915
				3	......	17.3	82.7	......	.69	......	......	......	......	......	8,778	15,800
Same (composite of samples W19689, W19691, and W19692).	280	12217	A	1	1.43	16.92	75.22	6.43	1.08	4.50	83.40	1.30	3.29	......	8,087	14,557
				2	......	17.17	76.31	6.52	1.10	4.40	84.61	1.32	2.05	......	8,204	14,767
				3	......	18.37	81.63	......	1.18	4.71	90.51	1.41	2.19	......	8,776	15,797
Same (face of room 8, left heading 2, off dip off main south heading, 3-foot 3-inch bed, 3-foot ½-inch cut).	280	W19688	A	1	1.7	16.3	74.4	7.6	.65	......	......	......	......	......	7,933	14,280
				2	......	16.6	75.7	7.7	.65	......	......	......	......	......	8,072	14,530
				3	......	18.0	82.0	......	.70	......	......	......	......	......	8,744	15,740

Same (face of room 13, off left heading 1, off dip, off main south heading, 4-foot 4½-inch bed, 3-foot 10-inch cut).	280	W19690	A	1	2.5	15.4	76.6	5.5	1.15	......	......	......	......	......	8,094	14,570
				2	......	15.8	78.6	5.6	1.20	......	......	......	......	......	8,300	14,940
				3	......	16.7	83.3	......	1.27	......	......	......	......	......	8,794	15,830
Same (face of room 10, first right off plane 17, off main north heading, 4-foot 5-inch bed, 3-foot 9½-inch cut).	280	W19693	A	1	2.2	16.1	75.0	6.7	1.30	......	......	......	......	......	7,978	14,360
				2	......	16.5	76.6	6.9	1.35	......	......	......	......	......	8,156	14,680
				3	......	17.7	82.3	......	1.45	......	......	......	......	......	8,756	15,760
Same (face of right heading 3, off main dip, off main north heading, 4-foot 4-inch bed, 3-foot 10½-inch cut).	280	W19694	A	1	4.2	15.4	74.9	5.5	.65	......	......	......	......	......	7,967	14,340
				2	......	16.1	78.2	5.7	.70	......	......	......	......	......	8,317	14,970
				3	......	17.1	82.9	......	.74	......	......	......	......	......	8,817	15,870
Same (composite of samples W19688, W19690, W19693, and W19694).	280	12205	A	1	1.44	15.96	76.34	6.26	.98	4.44	83.35	1.19	3.78	......	8,101	14,582
				2	......	16.19	77.46	6.35	.99	4.34	84.57	1.21	2.54	......	8,219	14,794
				3	......	17.29	82.71	......	1.06	4.63	90.30	1.29	2.72	......	8,776	15,797
Spangler, 2 miles northwest of; Pennsylvania No. 21 mine, D or Lower Freeport bed (face of main heading, 5,000 feet southwest of mine mouth, 3-foot 9-inch bed, 3-foot 7-inch cut.	281	12144	A	1	4.54	22.74	67.62	5.10	.63	......	......	......	......	3.8	7,867	14,161
				2	......	23.82	70.84	5.34	.66	......	......	......	......	......	8,241	14,834
				3	......	25.16	74.84	......	.70	......	......	......	......	......	8,706	15,671
Same (face of second main entry, 6,500 feet from mine mouth, 3-foot 5¾-inch bed, 2-foot 10-inch cut).	281	12145	A	1	2.40	25.79	65.82	5.99	1.18	......	......	......	......	1.4	7,965	14,337
				2	......	26.42	67.44	6.14	1.21	......	......	......	......	......	8,161	14,690
				3	......	28.15	71.85	......	1.29	......	......	......	......	......	8,695	15,651
Same (face of back heading 17, 5,900 feet west of mine mouth, 3-foot 6-inch bed, 2-foot 8¼-inch cut).	281	12146	A	1	3.02	26.59	64.34	6.05	1.74	......	......	......	......	2.1	7,902	14,224
				2	......	27.42	66.34	6.24	1.79	......	......	......	......	......	8,148	14,666
				3	......	29.25	70.75	......	1.91	......	......	......	......	......	8,691	15,644
Same (face of right entry 4, off second main entry, 6,300 feet east of mine mouth, 3-foot 7½-inch bed, 3-foot 5-inch cut).	281	12147	A	1	2.65	26.48	62.86	8.01	1.56	......	......	......	......	1.9	7,771	13,988
				2	......	27.20	64.57	8.23	1.60	......	......	......	......	......	7,982	14,368
				3	......	29.64	70.36	......	1.74	......	......	......	......	......	8,698	15,656
Same (composite of samples 12144–12147..........	281	12148	A	1	3.14	26.01	64.40	6.45	1.37	5.08	79.55	1.30	6.25	2.3	7,866	14,159
				2	......	26.85	66.49	6.66	1.41	4.88	82.13	1.34	3.58	......	8,121	14,618
				3	......	28.77	71.23	......	1.51	5.23	87.99	1.44	3.83	......	8,701	15,662
2 miles northwest of; Pennsylvania No. 22 mine, Lower Freeport or D bed (face of ninth main entry, 7,200 feet southwest of mine mouth, 3-foot 9-inch cut).	281	12126	A	1	4.86	21.85	67,48	5.81	1.01	......	......	......	......	3.9	7,804	14,047
				2	......	22.97	70.92	6.11	1.06	......	......	......	......	......	8,203	14,765
				3	......	24.47	75.53	......	1.13	......	......	......	......	......	8,737	15,727
Same (rib of room 9, off right entry 4, off ninth main entry, 6,200 feet southwest of mine mouth, 3-foot 7¼-inch cut).	281	12127	A	1	2.55	22.53	68.11	6.81	1.20	......	......	......	......	1.7	7,894	14,209
				2	......	23.12	69.89	6.99	1.23	......	......	......	......	......	8,101	14,582
				3	......	24.86	75.14	......	1.32	......	......	......	......	......	8,710	15,678
Same (face of room 17, off right entry 6, off ninth main entry, 7,400 feet from mine mouth, 3-foot 9¾-inch cut).	281	12140	A	1	2.83	23.17	67.53	6.47	1.57	......	......	......	......	2.3	7,821	14,078
				2	......	23.84	69.50	6.66	1.62	......	......	......	......	......	8,049	14,488
				3	......	25.54	74.46	......	1.74	......	......	......	......	......	8,624	15,523
Same (face of main back heading, 7,100 feet from mine mouth, 3-foot 2-inch cut).	281	12141	A	1	2.70	23.73	66.86	6.71	1.39	......	......	......	......	2.0	7,904	14,227
				2	......	24.39	68.71	6.90	1.43	......	......	......	......	......	8,123	14,621
				3	......	26.20	73.80	......	1.54	......	......	......	......	......	8,725	15,705
Same (face of fourteenth right, off main heading, 7,200 feet from mine mouth, 3-foot 1½-inch cut).	281	12142	A	1	2.61	23.74	67.62	6.03	.95	......	......	......	......	1.9	7,968	14,342
				2	......	24.38	69.43	6.19	.98	......	......	......	......	......	8,182	14,728
				3	......	25.99	74.01	......	1.04	......	......	......	......	......	8,722	15,700
Same (composite of samples 12126, 12127, 12140, 12141, and 12142).	282	12143	A	1	3.08	22.77	67.60	6.55	1.24	4.90	80.54	1.36	5.41	2.4	7,856	14,141
				2	......	23.49	69.75	6.76	1.28	4.71	83.10	1.40	2.75	......	8,106	14,591
				3	......	25.19	74.81	......	1.37	5.05	89.12	1.50	2.96	......	8,694	15,649

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample. Laboratory No.	Sample. Kind.	Sample. Condition.	Proximate. Moisture.	Proximate. Volatile matter.	Proximate. Fixed carbon.	Proximate. Ash.	Ultimate. Sulphur.	Ultimate. Hydrogen.	Ultimate. Carbon.	Ultimate. Nitrogen.	Ultimate. Oxygen.	Air-drying loss.	Calorific value. Calories.	Calorific value. British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Twin Rocks, near: Big Bend No. 1 mine, Miller, B, or Lower Kittanning bed (face of room 11, off drainage heading, 2,600 feet northeast of main opening, 3-foot 7½-inch bed and cut).	282	W19668	A	1	2.4	20.0	71.7	5.9	1.70	......	......	......	......	......	8,094	14,570
				2	......	20.5	73.5	6.0	1.75	......	......	......	......	......	8,294	14,930
				3	......	21.8	78.2	......	1.86	......	......	......	......	......	8,828	15,890
Same (face of water-level heading, 2,000 feet northeast of main opening, 3-foot 6¾-inch bed and cut).	282	W19669	A	1	1.9	19.7	72.4	6.0	.60	......	......	......	......	......	8,078	14,540
				2	......	20.1	73.8	6.1	.60	......	......	......	......	......	8,239	14,830
				3	......	21.4	78.6	......	.64	......	......	......	......	......	8,772	15,790
Same (composite of samples W19668 and W19669)	282	12215	A	1	1.29	20.56	72.15	6.00	1.74	4.75	82.96	1.33	3.22	......	8,122	14,620
				2	......	20.83	73.09	6.08	1.76	4.67	84.05	1.35	2.09	......	8,228	14,810
				3	......	22.18	77.82	......	1.87	4.97	89.49	1.44	2.23	......	8,760	15,768
Same (pillar opposite room 16, off Claghorn section, 3,000 feet east of main opening, 3-foot 7½-inch bed).	282	W19672	A	1	2.0	19.5	73.0	5.5	1.50	......	......	......	......	......	8,078	14,540
				2	......	19.9	74.5	5.6	1.55	......	......	......	......	......	8,244	14,840
				3	......	21.1	78.9	......	1.64	......	......	......	......	......	8,739	15,730
Same (pillar to left of main heading, 2,500 feet east of main opening, 3-foot 7½-inch bed).	282	W19674	A	1	2.2	20.0	72.7	5.1	1.45	......	......	......	......	......	8,156	14,680
				2	......	20.5	74.3	5.2	1.50	......	......	......	......	......	8,344	15,020
				3	......	21.6	78.4	......	1.58	......	......	......	......	......	8,800	15,840
Same (pillar to left of main heading, opposite right headings 6 and 7, 2,000 feet east of main opening, 4-foot 3-inch bed and cut).	282	W19676	A	1	2.6	20.3	72.2	4.9	1.15	......	......	......	......	......	8,106	14,590
				2	......	20.8	74.2	5.0	1.20	......	......	......	......	......	8,322	14,980
				3	......	21.9	78.1	......	1.26	......	......	......	......	......	8,761	15,770
Same (composite of samples W19672, W19674, and W19676).	282	12208	A	1	1.32	20.33	73.01	5.34	1.39	4.66	83.45	1.33	3.83	......	8,201	14,762
				2	......	20.60	73.99	5.41	1.41	4.57	84.57	1.35	2.69	......	8,311	14,959
				3	......	21.78	78.22	......	1.49	4.83	89.41	1.43	2.84	......	8,786	15,815
1 mile north of: Big Bend No. 6 mine, Miller, B, or Lower Kittanning bed (room 1, left side of right heading 1, off No. 9 drift, 900 feet south of No. 9 opening, 3-foot 9½-inch bed, 3-foot 9½-inch cut).	283	W19670	A	1	2.7	18.9	72.6	5.8	1.75	......	......	......	......	......	8,017	14,430
				2	......	19.4	74.6	6.0	1.80	......	......	......	......	......	8,239	14,830
				3	......	20.6	79.4	......	1.91	......	......	......	......	......	8,767	15,780
Same (face of heading 13, 325 feet from outcrop and 3,500 feet southeast from main opening, 3-foot 9½-inch bed, 3-foot 8½-inch cut).	283	W19673	A	1	3.2	18.7	73.2	4.9	.95	......	......	......	......	......	8,011	14,420
				2	......	19.3	75.6	5.1	1.00	......	......	......	......	......	8,278	14,900
				3	......	20.3	79.7	......	1.05	......	......	......	......	......	8,722	15,700
Same (face of room 10 on left side of No. 9 drift, 700 feet southwest of No. 9 opening, 3-foot 8½-inch bed, 3-foot 8½-inch cut).	283	W19677	A	1	2.6	20.0	72.1	5.3	1.00	......	......	......	......	......	8,061	14,510
				2	......	20.5	74.1	5.4	1.05	......	......	......	......	......	8,272	14,890
					......	21.7	78.3	......	1.11	......	......	......	......	......	8,739	15,730

Same (composite of samples W19670, W 19673, and W19677).	283	12227	A	1	1.54	20.67	72.37	5.42	1.40	4.73	83.20	1.38	3.87	....	8,138	14,648
				2	....	20.99	73.51	5.50	1.42	4.63	84.50	1.40	2.55	....	8,265	14,877
				3	....	22.21	77.79	....	1.50	4.90	89.42	1.48	2.70	....	8,746	15,743
Same (pillar between rooms 23 and 24, off main heading, 1,500 feet northeast of main opening, 3-foot 7¾-inch bed).	283	W19671	A	1	2.8	19.4	72.5	5.3	1.10	....	....	....	....	....	8,006	14,410
				2	....	19.9	74.7	5.4	1.15	....	....	....	....	....	8,233	14,820
				3	....	21.0	79.0	....	1.22	....	....	....	....	....	8,717	15,690
Same (pillar between rooms 27 and 28, off main heading, 2,000 feet northeast of main opening, 3-foot 9½-inch cut).	283	W19675	A	1	2.2	19.7	72.7	5.4	1.10	....	....	....	....	....	8,106	14,590
				2	....	20.1	74.4	5.5	1.15	....	....	....	....	....	8,289	14,920
				3	....	21.3	78.7	....	1.22	....	....	....	....	....	8,767	15,780
Same (composite of samples W19671 and W19675)..	283	12207	A	1	1.47	20.18	72.91	5.44	1.20	4.75	83.43	1.27	3.91	....	8,169	14,704
				2	....	20.48	74.00	5.52	1.22	4.66	84.67	1.29	2.64	....	8,291	14,924
				3	....	21.68	78.32	....	1.29	4.93	89.61	1.37	2.80	....	8,775	15,795
Commercial No. 3 mine, Miller or B bed (face of main heading, 6,200 feet northwest of mine mouth, 5-foot ½-inch bed, 3-foot 10½-inch cut).	284	W20477	A	1	3.1	18.3	71.6	7.0	2.10	....	....	....	....	....	7,867	14,160
				2	....	18.9	73.9	7.2	2.15	....	....	....	....	....	8,117	14,610
				3	....	20.4	79.6	....	2.32	....	....	....	....	....	8,744	15,740
Same (face of right heading 7, 450 feet off main entry and 5,600 feet northwest of mine mouth, 3-foot 6½-inch bed, 3-foot 6-inch cut).	284	W20479	A	1	2.1	20.2	70.4	7.3	2.75	....	....	....	....	....	7,906	14,230
				2	....	20.6	71.9	7.5	2.80	....	....	....	....	....	8,072	14,530
				3	....	22.3	77.7	....	3.03	....	....	....	....	....	8,728	15,710
Same (face of left entry 7, 700 feet off main entry and 5,300 feet northwest of mine mouth, 3-foot 11⅞-inch bed and cut).	284	W20480	A	1	2.9	17.8	71.4	7.9	.60	....	....	....	....	....	8,028	14,450
				2	....	18.3	73.6	8.1	.60	....	....	....	....	....	8,267	14,880
				3	....	19.9	80.1	....	.65	....	....	....	....	....	8,800	15,840
Same (face of right heading 5, 5,000 feet off main entry and 4,200 feet north of mine mouth, 3-foot 8¼-inch bed and cut).	284	W20481	A	1	2.2	19.4	72.7	5.7	1.80	....	....	....	....	....	8,100	14,580
				2	....	19.8	74.4	5.8	1.85	....	....	....	....	....	8.278	14,900
				3	....	21.0	79.0	....	1.96	....	....	....	....	....	8,789	15,820
Same (composite of samples W20477, W20479, W20480, and W20481).	284	12377	A	1	.82	20.42	72.11	6.65	2.35	4.50	81.89	1.22	3.39	....	8,083	14,549
				2	....	20.59	72.70	6.71	2.37	4.45	82.57	1.23	2.67	....	8,150	14,670
				3	....	22.07	77.93	....	2.54	4.77	88.51	1.32	2.86	....	8,736	15,725
Same (pillar on right heading 4, 5,000 feet off main entry and 4,500 feet northeast of mine mouth, 3-foot 6-inch bed, 3-foot 5⅝-inch cut).	284	W20478	A	1	1.8	20.0	71.5	6.7	1.70	....	....	....	....	....	8,011	14,420
				2	....	20.4	72.8	6.8	1.75	....	....	....	....	....	8,156	14,680
				3	....	21.9	78.1	....	1.88	....	....	....	....	....	8,756	15,760
Commercial No. 4 mine, lower Kittanning or B bed (face of main entry 3, off No. 4 slope, 3-foot 7½-inch bed and cut).	285	W17061	A	1	3.3	17.0	73.9	5.8	1.60	....	....	....	....	....	7,978	14,360
				2	....	17.5	76.5	6.0	1.70	....	....	....	....	....	8,250	14,850
				3	....	18.6	81.4	....	1.81	....	....	....	....	....	8,778	15,800
Same (100 feet from face of main heading, 5,200 feet north of mine mouth, 3-foot 8-inch bed and cut).	285	W20501	A	1	3.8	18.1	71.9	6.2	1.85	....	....	....	....	....	7,878	14,180
				2	....	18.8	74.8	6.4	1.95	....	....	....	....	....	8,189	14,740
				3	....	20.1	79.9	....	2.08	....	....	....	....	....	8,744	15,740
Same (face of east entry 6, off right entry 3, off main entry, 4,000 feet northeast of mine mouth, 3-foot 8⅜-inch bed and cut).	285	W20505	A	1	2.4	19.0	72.3	6.3	1.85	....	....	....	....	....	7,956	14,320
				2	....	19.5	74.0	6.5	1.90	....	....	....	....	....	8,156	14,680
				3	....	20.9	79.1	....	2.03	....	....	....	....	....	8,722	15,700
Same (rib near face of east entry 7, off right entry 3, off main entry, 4,800 feet northeast of mine mouth, 3-foot 10⅝-inch bed and cut).	285	W20506	A	1	3.0	18.3	73.1	5.6	1.45	....	....	....	....	....	8,011	14,420
				2	....	18.9	75.3	5.8	1.50	....	....	....	....	....	8,256	14,860
				3	....	20.1	79.9	....	1.59	....	....	....	....	....	8,761	15,770
Same (face of right entry 3, off main entry, 4,900 feet northeast of mine mouth, 3-foot 7-inch cut).	285	W20508	A	1	3.3	18.2	72.9	5.6	1.75	....	....	....	....	....	7,961	14,330
				2	....	18.8	75.4	5.8	1.80	....	....	....	....	....	8,239	14,830
				3	....	20.0	80.0	....	1.91	....	....	....	....	....	8,750	15,750
Same (composite of samples W20501, W20505, W20506, and W20508).	285	12364	A	1	.89	19.73	73.30	6.08	1.95	4.58	83.16	1.28	2.95	....	8,090	14,562
				2	....	19.91	73.96	6.13	1.97	4.52	83.91	1.29	2.18	....	8,163	14,693
				3	....	21.21	78.79	....	2.10	4.82	89.39	1.37	2.32	....	8,696	15,653
Same (sixth pillar on east entry 9, off right entry 2, off main entry, 3,800 feet northeast of mine mouth, 3-foot 4¾-inch cut).	285	W20503	A	1	2.8	18.6	72.9	5.7	1.55	....	....	....	....	....	7,983	14,370
				2	....	19.1	75.0	5.9	1.60	....	....	....	....	....	8,217	14,790
				3	....	20.3	79.7	....	1.70	....	....	....	....	....	8,733	15,720

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Twin Rocks, 1 mile north of; Commercial No. 5 mine, Miller or B bed (face of room 1, off east heading 10, off main entry, 3-foot 8-inch bed).	286	W17062	A	1	3.3	20.0	72.3	4.4	.90	......	......	......	......	......	8,106	14,590
				2	......	20.5	74.9	4.6	.95	......	......	......	......	......	8,383	15,090
				3	......	21.5	78.5	......	1.00	......	......	......	......	......	8,789	15,820
Same (face of main heading, 4,700 feet northeast of mine mouth, 3-foot 7½-inch bed and cut).	286	W20499	A	1	2.8	18.8	72.4	6.0	2.05	......	......	......	......	......	7,972	14,350
				2	......	19.3	74.5	6.2	2.10	......	......	......	......	......	8,200	14,760
				3	......	20.6	79.4	......	2.24	......	......	......	......	......	8,744	15,740
Same (face of room 7, off right heading 1; 500 feet from second parallel entry and 3,400 feet northeast of mine mouth; 3-foot 5½-inch cut).	286	W20500	A	1	2.9	19.2	72.1	5.8	1.70	......	......	......	......	......	8,028	14,450
				2	......	19.8	74.2	6.0	1.75	......	......	......	......	......	8,267	14,880
				3	......	21.1	78.9	......	1.86	......	......	......	......	......	8,794	15,830
Same (face of east heading 12; 325 feet off main entry, 4,700 feet northeast of mine mouth, 3-foot 8½-inch bed, 3 foot 8 inch cut).	286	W20502	A	1	2.9	18.9	72.5	5.7	2.00	......	......	......	......	......	8,000	14,400
				2	......	19.5	74.6	5.9	2.05	......	......	......	......	......	8,239	14,830
				3	......	20.7	79.3	......	2.18	......	......	......	......	......	8,756	15,760
Same (east heading 11; 800 feet off main entry, 4,000 feet northeast of mine mouth, 3-foot 6⅜-inch cut).	286	W20504	A	1	2.8	19.2	71.3	6.7	2.15	......	......	......	......	......	7,956	14,320
				2	......	19.7	73.4	6.9	2.20	......	......	......	......	......	8,183	14,730
				3	......	21.2	78.8	......	2.36	......	......	......	......	......	8,789	15,820
Same (face of room 10, off east heading 10; 750 feet off main heading, 3,900 feet northeast of mine mouth, 3-foot 8½-inch bed and cut).	286	W20507	A	1	3.2	18.2	71.9	6.7	2.15	......	......	......	......	......	7,894	14,210
				2	......	18.8	74.3	6.9	2.25	......	......	......	......	......	8,156	14,680
				3	......	20.2	79.8	......	2.42	......	......	......	......	......	8,761	15,770
Same (composite of samples W20499, W20500, W20502, W20504, and W20507).	286	12375	A	1	.81	20.78	72.10	6.31	2.24	4.60	82.42	1.21	3.22	......	8,073	14,531
				2	......	20.95	72.69	6.36	2.26	4.55	83.10	1.22	2.51	......	8,139	14,650
				3	......	22.37	77.63	......	2.41	4.86	88.74	1.30	2.69	......	8,692	15,646
CENTER COUNTY.																
Osceola Mills, ½ mile south of; Electric mine, Lower Kittanning or B bed (face of right entry 1, off new drift entry, 4-foot 10-inch bed, 3-foot 9-inch cut).	287	12067	A	1	3.45	20.57	68.55	7.43	1.84	......	......	......	......	2.8	7,771	13,888
				2	......	21.30	71.00	7.70	1.91	......	......	......	......	......	8,048	14,486
				3.	......	23.08	76.92	......	2.07	......	......	......	......	......	8,719	15,694
Same (pillar at room 7, off right entry 4, off main heading, 4-foot 10¼-inch bed, 3-foot 11¼-inch cut).	287	12069	A	1	3.28	21.61	67.75	7.36	2.01	......	......	......	......	2.3	7,742	13,936
				2	......	22.34	70.05	7.61	2.08	......	......	......	......	......	8,004	14,407
				3	......	24.18	75.82	......	2.25	......	......	......	......	......	8,664	15,595
Same (pillar at room 5, off left entry 5, off main heading, 4-foot 10¼-inch bed, 3-foot 9¼-inch cut).	287	12070	A	1	3.09	20.73	67.90	8.28	2.53	......	......	......	......	2.4	7,654	13,777
				2	......	21.39	70.07	8.54	2.61	......	......	......	......	......	7,898	14,216
				3	......	23.39	76.61	......	2.85	......	......	......	......	......	8,636	15,545

Same (composite of samples 12067, 12069, and 12070).	287	12071	A	1	3.24	20.65	68.34	7.77	2.08	4.89	77.85	1.25	6.16	2.5	7,726	13,907
				2	....	21.34	70.63	8.03	2.15	4.68	80.46	1.29	3.39	....	7,985	14,373
				3	....	23.20	76.80	....	2.34	5.09	87.48	1.40	3.69	....	8,682	15,626
Same (pillar at room 9, off left entry 5, off main entry, 5-foot 2½-inch bed, 4 foot 1½-inch cut).	287	12068	A	1	3.48	20.26	68.55	7.71	1.63	4.82	78.45	1.26	6.13	2.5	7,715	13,887
				2	....	20.99	71.02	7.99	1.69	4.59	81.28	1.31	3.14	....	7,994	14,389
				3	....	22.81	77.19	....	1.84	4.99	88.34	1.42	3.41	....	8,688	15,638
Same (composite of samples 12068, 12069, and 12070).	287	12190	A	1	3.04	22.92	67.32	6.72	1.54	4.95	79.49	1.33	5.97	2.4	7,851	14,132
				2	....	23.64	69.43	6.93	1.59	4.75	81.99	1.37	3.37	....	8,098	14,576
				3	....	25.40	74.60	....	1.71	5.10	88.10	1.47	3.62	....	8,701	15,662
½ mile north of; Moshannon No. 10 mine, Lower Kittanning or B bed (on rib of room 18, off left entry 2, off right entry 4, 40 feet from left entry 2, 4-foot 6¾-inch bed, 3-foot 7½-inch cut).	287	12057	A	1	2.59	21.53	68.38	7.50	1.91	....	....	....	....	1.9	7,835	14,103
				2	....	22.10	70.20	7.70	1.96	....	....	....	....	....	8,043	14,477
				3	....	23.94	76.06	....	2.12	....	....	....	....	....	8,714	15,685
Same (face of room 21, off left main heading, 4-foot 5-inch bed, 3-foot 8½-inch cut).	287	12058	A	1	2.67	22.09	68.48	6.76	2.00	....	....	....	....	2.1	7,883	14,189
				2	....	22.70	70.35	6.95	2.05	....	....	....	....	....	8,099	14,578
				3	....	24.40	75.60	....	2.20	....	....	....	....	....	8,704	15,667
Same (face of room 16, off right main heading, near first crosscut to right, 4-foot 3¾-inch bed, 3-foot 8¼-inch cut).	287	12059	A	1	2.75	19.59	70.35	7.31	2.11	....	....	....	....	2.2	7,839	14,110
				2	....	20.14	72.34	7.52	2.17	....	....	....	....	....	8,061	14,510
				3	....	21.78	78.22	....	2.35	....	....	....	....	....	8,716	15,689
Same (face of right heading 1, 3-foot 8¾-inch bed, 3-foot 3-inch cut).	287	12060	A	1	3.80	19.98	69.09	7.13	1.73	....	....	....	....	3.2	7,779	14,002
				2	....	20.77	71.82	7.41	1.80	....	....	....	....	....	8,086	14,555
				3	....	22.43	77.57	....	1.94	....	....	....	....	....	8,733	15,719
Same (face of room 16, off right heading 2, 3-foot 5½-bed, 2-foot 6¾-inch cut).	287	12061	A	1	3.03	21.25	67.95	7.77	2.48	....	....	....	....	2.5	7,750	13,950
				2	....	21.91	70.08	8.01	2.56	....	....	....	....	....	7,992	14,386
				3	....	23.82	76.18	....	2.78	....	....	....	....	....	8,688	15,638
Same (composite of samples 12057-12061)..........	288	12062	A	1	3.05	21.65	67.95	7.35	1.98	4.99	79.16	1.31	5.21	2.4	7,798	14,036
				2	....	22.33	70.09	7.58	2.04	4.80	81.65	1.35	2.58	....	8,044	14,479
				3	....	24.16	75.84	....	2.21	5.19	88.35	1.46	2.79	....	8,704	15,667
⅓ mile south of; Weston mine, Lower Kittanning or B bed (face of left entry 4, off main entry, 4-foot 5¾-inch bed, 3-foot 7½-inch cut).	288	12063	A	1	3.20	19.52	68.74	8.54	2.35	....	....	....	....	2.5	7,664	13,795
				2	....	20.17	71.01	8.82	2.43	....	....	....	....	....	7,918	14,252
				3	....	22.12	77.88	....	2.66	....	....	....	....	....	8,684	15,631
Same (rib of main heading, at left entry 7, 5-foot 1½-inch bed, 4-foot 4-inch cut).	288	12064	A	1	2.91	19.53	70.53	7.03	1.86	....	....	....	....	2.2	7,832	14,098
				2	....	20.12	72.64	7.24	1.92	....	....	....	....	....	8,067	14,521
				3	....	21.69	78.31	....	2.07	....	....	....	....	....	8,697	15,655
Same (composite of samples 12063 and 12064).....	288	12066	A	1	3.08	20.27	68.83	7.82	2.11	4.85	79.00	1.28	4.94	2.4	7,754	13,957
				2	....	20.91	71.02	8.07	2.18	4.65	81.51	1.32	2.27	....	8,001	14,402
				3	....	22.75	77.25	....	2.37	5.06	88.67	1.44	2.46	....	8,703	15,665
Same (pillar at room 8, off right entry 1, off main heading, 4-foot 7-inch cut).	288	12065	A	1	2.94	19.89	69.68	7.49	1.93	4.98	79.67	1.23	4.70	2.2	7,822	14,080
				2	....	20.49	71.79	7.72	1.99	4.79	82.08	1.27	2.15	....	8,059	14,506
				3	....	22.21	77.79	....	2.16	5.19	88.95	1.38	2.32	....	8,734	15,721
CLEARFIELD COUNTY.																
Boardman; Potts Run No. 3 mine, Lower Kittanning or B bed (face of right entry 3, off main entry, 1,400 feet southwest of mine mouth, 3-foot 7½-inch bed, 3-foot 6½-inch cut).	289	12093	A	1	2.74	20.75	67.85	8.66	1.59	....	....	....	....	2.0	7,724	13,903
				2	....	21.34	69.76	8.90	1.63	....	....	....	....	....	7,942	14,296
				3	....	23.42	76.58	....	1.79	....	....	....	....	....	8,718	15,692
Same (on rib, 10 feet from face of right entry 4, off main entry 1,700 feet southwest of mine mouth, 3-foot 7¼-inch bed, 3-foot 7¼-inch cut).	289	12094	A	1	2.37	20.82	68.14	8.67	1.53	....	....	....	....	1.7	7,749	13,948
				2	....	21.33	69.79	8.88	1.57	....	....	....	....	....	7,937	14,287
				3	....	23.41	76.59	....	1.72	....	....	....	....	....	8,711	15,680
Same (face of left entry 3, off main entry, 1,680 feet northeast of mine mouth, 3-foot 6½-inch bed, 3-foot 6-inch cut).	289	12095	A	1	2.80	20.65	67.69	8.86	1.38	....	....	....	....	2.1	7,719	13,894
				2	....	21.24	69.64	9.12	1.42	....	....	....	....	....	7,941	14,294
				3	....	23.37	76.63	....	1.56	....	....	....	....	....	8,738	15,728

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CLEARFIELD COUNTY—continued.																
Boardman; Potts Run No. 3 mine, Lower Kittanning or B bed (face of main heading, 2,000 feet north of mine mouth, 3-foot 11½-inch bed, 3-foot 9-inch cut).	289	12096	A	1	3.38	20.50	67.75	8.37	1.08	......	......	......	......	2.7	7,697	13,855
				2	......	21.22	70.12	8.66	1.12	......	......	......	......	......	7,966	14,339
				3	......	23.23	76.77	......	1.23	......	......	......	......	......	8,721	15,698
Same (face of room 12, off right entry 2, off main entry, 3-foot 7¼-inch bed, 3-foot 5¾-inch cut).	289	12097	A	1	3.04	21.22	66.83	8.91	1.41	......	......	......	......	2.3	7,698	13,856
				2	......	21.89	68.92	9.19	1.45	......	......	......	......	......	7,940	14,292
				3	......	24.11	75.89	......	1.60	......	......	......	......	......	8,744	15,739
Same (composite of samples 12093–12097)..........	289	12098	A	1	2.95	21.29	66.92	8.84	1.35	4.74	78.51	1.19	5.37	2.2	7,723	13,901
				2	......	21.94	68.95	9.11	1.39	4.54	80.90	1.23	2.83	......	7,958	14,324
				3	......	24.14	75.86	......	1.53	4.99	89.01	1.35	3.12	......	8,755	15,759
Brisbin, 1 mile southwest of; Lenore No. 1 mine, Upper Freeport or E bed (face of left heading, off main heading, off No. 1 drift, 2-foot 7-inch bed and cut).	290	12046	A	1	3.00	21.59	67.61	7.80	.79	......	......	......	......	1.9	7,786	14,015
				2	......	22.26	69.70	8.04	.81	......	......	......	......	......	8,027	14,449
				3	......	24.21	75.79	......	.88	......	......	......	......	......	8,729	15,712
1 mile southwest of; Lenore No. 2 mine, Upper Freeport or E bed (face of left heading 3, off No. 2 drift, 2 foot 7½-inch bed and cut).	290	12044	A	1	3.14	21.77	66.12	8.97	1.08	......	......	......	......	2.0	7,690	13,842
				2	......	22.48	68.26	9.26	1.11	......	......	......	......	......	7,939	14,290
				3	......	24.77	75.23	......	1.22	......	......	......	......	......	8,749	15,748
Same (face of left heading 2, off No. 2 drift, 2-foot 6¾-inch bed and cut).	290	12045	A	1	2.83	21.78	67.89	7.50	.91	4.92	79.68	1.33	5.66	1.9	7,882	14,188
				2	......	22.41	69.87	7.72	.94	4.74	82.00	1.37	3.23	......	8,111	14,600
				3	......	24.29	75.71	......	1.02	5.14	88.86	1.48	3.50	......	8,788	15,818
Same (composite of samples 12044 and 12046).....	290	12047	A	1	3.07	21.47	67.26	8.20	.94	4.80	78.48	1.23	6.35	2.0	7,757	13,963
				2	......	22.15	69.39	8.46	.97	4.60	80.97	1.27	3.73	......	8,003	14,405
				3	......	24.20	75.80	......	1.06	5.03	88.45	1.39	4.07	......	8,742	15,736
Madera, 1 mile east of; Morgan Run mine, Lower Kittanning or B bed (face of south entry 4, off right entry 5, 3-foot 5½-inch bed, 3-foot 4½-inch cut).	290	W20402	A	1	1.6	18.6	69.4	10.4	2.55	......	......	......	......	......	7,678	13,820
				2	......	18.9	70.5	10.6	2.60	......	......	......	......	......	7,800	14,040
				3	......	21.1	78.9	......	2.91	......	......	......	......	......	8,722	15,700
2 miles north of; Bucher mine, Lower Kittanning or B bed (left entry 9, off cross heading 1, 3,400 feet northeast of mouth of main entry, 3-foot 1½-inch bed, 2-foot 11-inch cut).	290	W20435	A	1	2.3	18.7	68.5	10.5	1.00	......	......	......	......	......	7,606	13,690
				2	......	19.1	70.2	10.7	1.05	......	......	......	......	......	7,783	14,010
				3	......	21.4	78.6	......	1.18	......	......	......	......	......	8,711	15,680
Same (face of right heading 8, 3,800 feet northeast of mouth of main heading, 3-foot 3-inch bed, 3-foot 2½-inch cut).	290	W20436	A	1	2.1	20.0	68.9	9.0	1.45	......	......	......	......	......	7,767	13,980
				2	......	20.4	70.4	9.2	1.50	......	......	......	......	......	7,933	14,280
				3	......	22.5	77.5	......	1.65	......	......	......	......	......	8,739	15,730

Same (face of right heading 10, 4,500 feet northeast of mouth of main entry, 3-foot 3-inch bed, 3-foot 2-inch cut).	290	W 20437	A	1	2.4	19.9	68.2	9.5	1.20						7,700	13,860
				2		20.4	69.9	9.7	1.25						7,889	14,200
				3		22.6	77.4		1.38						8,739	15,730
Same (face of left heading 7, 3,000 feet northeast of mouth of main entry, 3-foot 6¾-inch bed, 3-foot 3½-inch cut).	290	W20438	A	1	2.3	19.5	70.2	8.0	1.15						7,839	14,110
				2		19.9	71.9	8.2	1.20						8,017	14,430
				3		21.7	78.3		1.31						8,733	15,720
Same (face of right heading 7, 3,200 feet northeast of mouth of main entry, 3-foot 1¼-inch cut).	290	W20439	A	1	2.3	20.4	69.0	8.3	1.20						7,756	13,960
				2		20.9	70.6	8.5	1.25						7,939	14,290
				3		22.8	77.2		1.37						8,772	15,790
Same (composite of samples W 20435–W 20439	290	12376	A	1	.80	21.36	68.48	9.36	1.38	4.49	80.07	1.09	3.61		7,790	14,022
				2		21.53	69.03	9.44	1.39	4.44	80.72	1.10	2.91		7,853	14,135
				3		23.77	76.23		1.53	4.90	89.13	1.21	3.23		8,671	15,608
Same (face of east entry 3, off right entry 5, 3-foot 3¾-inch bed, 3-foot 2-inch cut).	290	W20403	A	1	1.9	19.1	68.5	10.5	1.75						7,606	13,690
				2		19.5	69.8	10.7	1.80						7,750	13,950
				3		21.8	78.2		2.02						8,678	15,620
Same (face of south entry 3, off right entry 5, 3-foot 7¾-inch bed, 3-foot 7½-inch cut).	290	W20404	A	1	2.0	18.2	67.4	12.4	3.15						7,411	13,340
				2		18.6	68.7	12.7	3.20						7,561	13,610
				3		21.3	78.7		3.67						8,661	15,590
Same (face of south entry 1, off right entry 5, 3-foot 5½-inch bed, 3-foot 3¾-inch cut).	290	W20405	A	1	2.0	18.7	68.3	11.0	3.30						7,550	13,590
				2		19.1	69.7	11.2	3.35						7,700	13,860
				3		21.5	78.5		3.77						8,672	15,610
Same (face of main heading, 3-foot 6½-inch bed, 3-foot 5½-inch cut).	290	W20406	A	1	1.9	17.3	70.5	10.3	3.50						7,589	13,660
				2		17.6	71.9	10.5	3.55						7,733	13,920
				3		19.7	80.3		3.97						8,639	15,550
Same (composite of samples W20402–W20406)	291	12368	A	1	.82	20.40	67.86	10.92	3.08	4.34	77.62	1.08	2.96		7,630	13,734
				2		20.57	68.42	11.01	3.11	4.29	78.26	1.09	2.24		7,693	13,847
				3		23.11	76.89		3.49	4.82	87.94	1.22	2.53		8,645	15,561
Morrisdale; Morrisdale No. 1 mine, Middle Kittanning or C bed (face of sump heading, 3,100 feet from shaft No. 1, 5-foot bed, 4-foot 8½-inch cut).	293	12079	A	1	3.47	20.49	67.30	8.74	1.09					2.8	7,644	13,759
				2		21.23	69.72	9.05	1.13						7,918	14,252
				3		23.34	76.66		1.24						8,706	15,671
Same (at face of McKinley heading, off Truck main heading, 5,850 feet from bottom of shaft No. 1, 4-foot 6-inch bed, 4-foot 5¾-inch cut).	293	12080	A	1	4.19	19.70	67.64	8.47	1.07					3.1	7,554	13,597
				2		20.56	70.60	8.84	1.12						7,884	14,191
				3		22.55	77.45		1.23						8,649	15,568
Same (composite samples 12079 and 12080)	293	12082	A	1	3.86	19.79	67.54	8.81	1.08	4.85	77.44	1.29	6.53	3.0	7,565	13,617
				2		20.58	70.26	9.16	1.12	4.60	80.55	1.34	3.23		7,868	14,162
				3		22.65	77.35		1.23	5.06	88.67	1.48	3.56		8,661	15,590
Morrisdale No. 1 mine, Lower Kittanning or B bed (face of main heading, 5,240 feet from bottom of shaft No. 1, 4-foot ½-inch bed, 3-foot 7½-inch cut).	292	12074	A	1	4.15	20.01	66.83	9.01	3.77					3.7	7,526	13,547
				2		20.88	69.72	9.40	3.93						7,852	14,134
				3		23.05	76.95		4.34						8,667	15,601
Same (face of room 13, off right heading 8, off main heading, 5,040 feet from bottom of shaft No. 1, 4-foot 3¾-inch bed, 3-foot 9¾-inch cut).	292	12075	A	1	3.24	19.58	69.21	7.97	3.41					2.7	7,721	13,898
				2		20.24	71.52	8.24	3.52						7,980	14,364
				3		22.06	77.94		3.84						8,697	15,655
Same (face of room 45, off right entry 4, off main heading, 5,200 feet from bottom of shaft No. 1, 4-foot 1½-inch bed, 3-foot 6¾-inch cut).	292	12081	A	1	3.19	21.88	67.24	7.69	3.06					2.7	7,746	13,943
				2		22.60	69.46	7.94	3.16						8,002	14,404
				3		24.55	75.45		3.43						8,692	15,646
Same (composite of samples 12074, 12075, and 12081)	293	12090	A	1	3.61	20.86	67.07	8.46	3.38	4.92	77.06	1.28	4.90	3.0	7,679	13,822
				2		21.64	69.58	8.78	3.51	4.69	79.95	1.33	1.74		7,967	14,341
				3		23.72	76.28		3.85	5.14	87.65	1.46	1.90		8,734	15,721
Morrisdale No. 2 mine, Middle Kittaning or C bed (face of heading 6, 4,500 feet from bottom of shaft No. 2, 4-foot 10½-inch bed, 4-foot 9-inch cut).	293	12072	A	1	3.04	19.42	67.38	10.16	1.00					2.3	7,548	13,586
				2		20.03	69.49	10.48	1.03						7,785	14,013
				3		22.38	77.62		1.15						8,697	15,655

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CLEARFIELD COUNTY—continued.																
Morrisdale; Morrisdale No. 2 mine, Middle Kittanning or C bed (room 9, left heading 2, off Lucky strike, 2,200 feet from bottom of shaft No. 2, 4-foot 1-inch bed, 4-foot cut).	293	12073	A	1	2.52	20.73	66.96	9.79	1.74	....	....	....	....	2.0	7,640	13,752
				2	....	21.27	68.69	10.04	1.79	....	....	....	....	....	7,838	14,108
				3	....	23.64	76.36	....	1.99	....	....	....	....	....	8,715	15,688
Same (composite of samples 12072 and 12073)	293	12091	A	1	2.83	20.30	66.91	9.96	1.32	4.67	77.05	1.25	5.75	2.2	7,580	13,644
				2	....	20.89	68.86	10.25	1.36	4.49	79.29	1.29	3.32	....	7,801	14,042
				3	....	23.28	76.72	....	1.52	5.00	88.34	1.44	3.70	....	8,692	15,646
Morrisdale No. 3 mine, Lower Kittaning or B bed (room 5, off left heading 1, off new slope tunnel, 1,400 feet from shaft No. 3, 4-foot 1½-inch bed, 3-foot 6-inch cut).	292	12076	A	1	3.23	19.68	69.75	7.34	1.99	....	....	....	....	2.6	7,806	14,051
				2	....	20.34	72.07	7.59	2.06	....	....	....	....	....	8,067	14,521
				3	....	22.01	77.99	....	2.23	....	....	....	....	....	8,729	15,712
Same (face of shaft No. 2 heading, off main dip, 5,500 feet from shaft, 3-foot 8-inch bed, 2-foot 11½-inch cut).	292	12077	A	1	3.34	19.23	71.07	6.36	1.71	....	....	....	....	2.8	7,897	14,215
				2	....	19.90	73.52	6.58	1.77	....	....	....	....	....	8,170	14,706
				3	....	21.30	78.70	....	1.89	....	....	....	....	....	8,745	15,741
Same (face of Morrisdale heading, 6,300 feet from shaft, 3-foot 10½-inch bed, 3-foot 2½-inch cut).	292	12078	A	1	3.24	19.78	70.02	6.96	2.03	....	....	....	....	2.7	7,862	14,152
				2	....	20.44	72.37	7.19	2.10	....	....	....	....	....	8,125	14,623
				3	....	22.02	77.98	....	2.26	....	....	....	....	....	8,755	15,759
Same (composite samples 12076, 12077, and 12078)..	293	12092	A	1	3.30	20.13	69.57	7.00	1.91	4.66	79.93	1.36	5.14	2.7	7,862	14,152
				2	....	20.82	71.94	7.24	1.98	4.44	82.66	1.41	2.27	....	8,130	14,634
				3	....	22.45	77.55	....	2.13	4.79	89.12	1.52	2.44	....	8,765	15,777
Munson; Colorado No. 5 mine, Lower Kittaning or B bed (at face of left entry 6, off line heading, 4-foot 1-inch bed, 3-foot cut).	294	12109	A	1	3.74	19.77	69.68	6.81	1.66	....	....	....	....	2.7	7,835	14,108
				2	....	20.54	72.39	7.07	1.72	....	....	....	....	....	8,140	14,652
				3	....	22.10	77.90	....	1.86	....	....	....	....	....	8,759	15,766
Same (face of left entry 8, off main heading, 3,000 feet from drift mouth, 4-foot 2½-inch bed, 3-foot 2-inch cut).	294	12110	A	1	3.58	20.28	69.21	6.93	1.78	....	....	....	....	2.7	7,835	14,103
				2	....	21.03	71.78	7.19	1.85	....	....	....	....	....	8,126	14,627
				3	....	22.66	77.34	....	1.99	....	....	....	....	....	8,756	15,761
Same (face of main heading, 3,600 feet from drift mouth, 4-foot 4½-inch bed, 3-foot 2½-inch cut).	294	12111	A	1	3.31	20.02	69.34	7.33	1.64	....	....	....	....	2.4	7,817	14,071
				2	....	20.70	71.72	7.58	1.70	....	....	....	....	....	8,084	14,551
				3	....	22.40	77.60	....	1.84	....	....	....	....	....	8,747	15,745
Same (face of line heading, 4-foot 3½-inch bed, 3-foot 2-inch cut).	294	12112	A	1	3.27	19.91	68.31	8.51	2.92	....	....	....	....	2.5	7,688	13,838
				2	....	20.58	70.62	8.80	3.02	....	....	....	....	....	7,948	14,306
				3	....	22.57	77.43	....	3.31	....	....	....	....	....	8,715	15,687

Same (face of left heading 6, off main heading, 4-foot 10⅝-inch bed, 3-foot 9¾-inch cut).	294	12113.	A	1	3.40	19.44	68.75	8.41	2.02	......	......	......	......	2.4	7,673	13,811
				2	......	20.12	71.17	8.71	2.09	......	......	......	......	......	7,943	14,297
				3	......	22.04	77.96	......	2.29	......	......	......	......	......	8,701	15,662
Same (composite of samples 12109–12113)........	294	12114	A	1	3.27	19.91	69.02	7.80	2.02	4.84	78.95	1.32	5.07	2.5	7,789	14,020
				2	......	20.58	71.36	8.06	2.09	4.63	81.62	1.36	2.24	......	8,052	14,494
				3	......	22.38	77.62	......	2.27	5.04	88.78	1.48	2.43	......	8,758	15,764
Osceola Mills, 3 miles west of; Fairmount No. 2 mine, Lower Freeport or D bed (face of room 5, off left heading 2, 5-foot 1-inch bed and cut).	294	W17010	A	1	3.1	21.0	69.7	6.2	1.55	......	......	......	......	......	7,861	14,150
				2	......	21.5	72.1	6.4	1.60	......	......	......	......	......	8,117	14,610
				3	......	23.0	77.0	......	1.71	......	......	......	......	......	8,672	15,610
Same (face of room 14, off right heading 8, 5-foot 3½-inch bed and cut).	294	W17012	A	1	2.4	22.5	68.8	6.3	1.00	......	......	......	......	......	7,967	14,340
				2	......	23.0	70.6	6.4	1.00	......	......	......	......	......	8,161	14,690
				3	......	24.6	75.4	......	1.07	......	......	......	......	......	8,717	15,690
Same (composite of samples W17010 and W17012)	295	11711	A	1	1.80	......	......	6.44	1.31	4.71	81.71	1.28	4.55	......	......	......
				2	......	......	......	6.56	1.33	4.50	83.21	1.30	3.01	......	......	......
				3	......	......	......	......	1.42	4.91	89.05	1.39	3.23	......	......	......
Philipsburg, 2 miles west of; Guion mine, Lower Kittanning or B bed (face of right heading 1, off right entry 2, 4,000 feet from mine mouth, 3-foot 1½ inch bed and cut).	295	W20369	A	1	2.0	20.0	69.9	8.1	1.05	......	......	......	......	......	7,828	14,090
				2	......	20.4	71.3	8.3	1.10	......	......	......	......	......	7,983	14,370
				3	......	22.2	77.8	......	1.20	......	......	......	......	......	8,711	15,680
Same (face of right heading 2, 4,000 feet from mine mouth, 3-foot 2-inch bed and cut).	295	W20370	A	1	2.0	21.5	69.2	7.3	1.00	......	......	......	......	......	7,906	14,230
				2	......	21.9	70.7	7.4	1.05	......	......	......	......	......	8,067	14,520
				3	......	23.7	76.3	......	1.13	......	......	......	......	......	8,711	15,680
Same (back heading, off right entry 1, 2,500 feet from mine mouth, 3-foot 4-inch bed, 2-foot 8-inch cut).	295	W20371	A	1	2.9	18.4	69.1	9.6	2.15	......	......	......	......	......	7,600	13,680
				2	......	18.9	71.2	9.9	2.20	......	......	......	......	......	7,828	14,090
				3	......	21.0	79.0	......	2.44	......	......	......	......	......	8,683	15,630
Same (50 feet from face of left heading 1, off right entry 2, 3-foot 5½-inch bed and cut).	295	W20372	A	1	2.0	19.2	70.1	8.7	1.53	......	......	......	......	......	7,789	14,020
				2	......	19.6	71.5	8.9	1.60	......	......	......	......	......	7,950	14,310
				3	......	21.5	78.5	......	1.76	......	......	......	......	......	8,728	15,710
Same (face of left heading 1, off right entry 1, 2,200 feet from mine mouth, 3-foot 6¼-inch bed, 3-foot 2¾-inch cut).	295	W20373	A	1	2.4	19.6	67.5	10.5	1.85	......	......	......	......	......	7,539	13,570
				2	......	20.1	69.1	10.8	1.90	......	......	......	......	......	7,728	13,910
				3	......	22.5	77.5	......	2.13	......	......	......	......	......	8,667	15,600
Same (composite of samples W20369–W20373)....	295	12367	A	1	.90	21.59	68.49	9.02	1.99	4.57	79.49	1.31	3.62	......	7,811	14,060
				2	......	21.79	69.11	9.10	2.01	4.51	80.21	1.32	2.85	......	7,882	14,188
				3	......	23.97	76.03	......	2.21	4.96	88.24	1.45	3.14	......	8,671	15,608
Smoke Run, 3½ miles southeast of; Viola mine, Miller or B bed (near face of left heading 9, beyond room 18, 1,300 feet from main haulage way, 5-foot 8-inch bed, 4-foot 10-inch cut).	296	11748	A	1	3.83	20.40	68.87	6.90	1.11	......	......	......	......	3.1	7,812	14,062
				2	......	21.21	71.62	7.17	1.15	......	......	......	......	......	8,123	14,621
				3	......	22.85	77.15	......	1.24	......	......	......	......	......	8,750	15,750
Same (face of second slant to right just beyond room 3, 5-foot ½-inch bed, 4-foot ½-inch cut).	296	11749	A	1	3.67	20.36	68.87	7.10	.90	......	......	......	......	2.9	7,822	14,080
				2	......	21.14	71.49	7.37	.93	......	......	......	......	......	8,120	14,616
				3	......	22.82	77.18	......	1.00	......	......	......	......	......	8,766	15,779
Same (face of main entry, 4,750 feet southwest of opening, 5-foot 4½-inch bed, 4-foot 4½-inch cut).	296	11750	A	1	3.67	20.30	67.64	8.39	1.18	......	......	......	......	2.9	7,706	13,871
				2	......	21.07	70.22	8.71	1.22	......	......	......	......	......	8,000	14,400
				3	......	23.08	76.92	......	1.34	......	......	......	......	......	8,763	15,773
Same (composite of samples 11748, 11749, and 11750).	296	11751	A	1	3.73	20.29	68.41	7.57	1.33	4.86	78.92	1.22	6.10	3.0	7,761	13,970
				2	......	21.08	71.06	7.86	1.38	4.62	81.97	1.27	2.90	......	8,061	14,510
				3	......	22.88	77.12	......	1.50	5.01	88.96	1.38	3.15	......	8,749	15,748
Same (near face of left entry 10; 100 feet from room 12; 5-foot 4½-inch bed, 4-foot 5½-inch cut).	296	W18355	A	1	3.0	19.6	70.3	7.1	.75	......	......	......	......	......	7,822	14,080
				2	......	20.2	72.5	7.3	.75	......	......	......	......	......	8,061	14,510
				3	......	21.8	78.2	......	.81	......	......	......	......	......	8,694	15,650

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CLEARFIELD COUNTY—continued.																
Smoke Run, 3½ miles southeast of; Viola Mine, Miller or B bed (near face of third slant entry, 100 feet above room 1, 5-foot bed, 4 foot 11-inch cut).	296	W18356	A	1	3.0	19.2	69.7	8.1	1.60	....	....	....	....	....	7,767	13,980
				2	....	19.8	71.8	8.4	1.65	....	....	....	....	....	8,006	14,410
				3	....	21.6	78.4	....	1.80	....	....	....	....	....	8,739	15,730
Same (near face of left heading 9, 5-foot 7⅞-inch bed, 4-foot 7⅞-inch cut).	296	W18357	A	1	3.4	19.5	70.4	6.7	1.00	....	....	....	....	....	7,867	14,160
				2	....	20.2	72.9	6.9	1.05	....	....	....	....	....	8,144	14,660
				3	....	21.7	78.3	....	1.13	....	....	....	....	....	8,750	15,750
Same (face of main entry, 4,800 feet southwest of opening, 5-foot 1⅛-inch bed, 4-foot 10⅞-inch cut).	296	W18358	A	1	3.3	19.0	69.7	8.0	1.60	....	....	....	....	....	7,711	13,880
				2	....	19.7	72.0	8.3	1.65	....	....	....	....	....	7,978	14,360
				3	....	21.5	78.5	....	1.80	....	....	....	....	....	8,700	15,660
Same (face of second slant entry, 100 feet above third slant entry, 5-foot bed, 4-foot 1-inch cut).	296	W18359	A	1	3.3	19.0	70.2	7.5	1.00	....	....	....	....	....	7,833	14,100
				2	....	19.6	72.6	7.8	1.05	....	....	....	....	....	8,094	14,570
				3	....	21.3	78.7	....	1.14	....	....	....	....	....	8,778	15,800
HUNTINGDON COUNTY.																
Robertsdale, Robertsdale mine; Fulton (Lower Kittanning or B) bed (face of north heading 6, 6-foot 3¼-inch bed, 4-foot 5⅝-inch cut).	297	12115	A	1	2.26	14.81	76.54	6.39	1.36	....	....	....	....	1.6	8,038	14,468
				2	....	15.15	78.31	6.54	1.39	....	....	....	....	....	8,224	14,803
				3	....	16.21	83.79	....	1.49	....	....	....	....	....	8,800	15,840
Same (at face of main west heading, 5-foot 4-inch bed, 3-foot 9-inch cut).	297	12116	A	1	1.92	15.23	77.25	5.60	.80	....	....	....	....	1.2	8,130	14,634
				2	....	15.53	78.76	5.71	.82	....	....	....	....	....	8,289	14,920
				3	....	16.47	83.53	....	.87	....	....	....	....	....	8,791	15,824
Same (at face of south heading 6, 6-foot 1¼-inch bed, 4-foot 9¼-inch cut).	297	12117	A	1	1.95	14.58	76.24	7.23	.94	....	....	....	....	1.3	7,983	14,369
				2	....	14.87	77.76	7.37	.96	....	....	....	....	....	8,142	14,656
				3	....	16.05	83.95	....	1.04	....	....	....	....	....	8,790	15,822
Same (face of south heading 8, 5-foot 3-inch bed, 4-foot 3-inch cut).	297	12118	A	1	2.20	13.93	77.44	6.43	1.08	....	....	....	....	1.5	8,046	14,483
				2	....	14.24	79.19	6.57	1.10	....	....	....	....	....	8,227	14,809
				3	....	15.24	84.76	....	1.18	....	....	....	....	....	8,805	15,849
Same (face of south heading 5, 3-foot 11⅝-inch bed, 3-foot 4⅝-inch cut).	297	12119	A	1	2.34	15.64	75.55	6.47	1.02	....	....	....	....	1.7	8,008	14,414
				2	....	16.02	77.35	6.63	1.04	....	....	....	....	....	8,200	14,760
				3	....	17.16	82.84	....	1.11	....	....	....	....	....	8,782	15,808
Same (composite of samples 12115–12119).........	298	12120	A	1	2.14	15.47	75.96	6.43	1.05	4.44	82.83	1.27	3.98	1.5	8,039	14,470
				2	....	15.81	77.62	6.57	1.07	4.29	84.64	1.30	2.13	....	8,215	14,787
				3	....	16.92	83.08	....	1.15	4.59	90.59	1.39	2.28	....	8,793	15,827
Woodvale; Woodvale mine, Fulton (Lower Kittanning or B) bed (face of south heading 1, 4-foot 5⅝-inch bed, 3-foot 9⅝-inch cut).	298	12121	A	1	2.92	14.28	76.74	6.06	.94	....	....	....	....	2.2	7,997	14,395
				2	....	14.71	79.05	6.24	.97	....	....	....	....	....	8,238	14,828
				3	....	15.69	84.31	....	1.03	....	....	....	....	....	8,787	15,817

Same (face of third south or up-creek heading, 4-foot 6¾-inch bed, 3 foot 7½-inch cut).	298	12122	A	1	1.98	13.38	79.26	5.38	.86	....	....	....	....	1.3	8,159	14,686
				2	....	13.65	80.86	5.49	.88	....	....	....	....	....	8,324	14,982
				3	....	14.44	85.56	....	.92	....	....	....	....	....	8,808	15,854
Same (face of second west tunnel 4-foot 9½-inch bed, 3-foot 11¾-inch cut).	298	12123	A	1	1.85	14.14	78.09	5.92	.92	....	....	....	....	1.3	8,116	14,609
				2	....	14.41	79.56	6.03	1.00	....	....	....	....	....	8.269	14,884
				3	....	15.34	84.66	....	1.06	....	....	....	....	....	8,800	15,840
Same (No. 7 or Bolinger heading, 5-foot 10⅜-inch bed, 5-foot 1⅜-inch cut).	298	12124	A	1	2.43	15.31	75.48	6.78	1.79	....	....	....	....	1.7	7,939	14,290
				2	....	15.69	77.36	6.95	1.83	....	....	....	....	....	8,137	14,647
				3	....	16.86	83.14	....	1.97	....	....	....	....	....	8,745	15,741
Same (composite of samples 12121–12124).........	298	12125	A	1	2.18	14.07	77.62	6.13	1.14	4.46	83.33	1.30	3.64	1.6	8,085	14,553
				2	....	14.38	79.35	6.27	1.17	4.31	85.19	1.33	1.73	....	8,265	14,877
				3	....	15.34	84.66	....	1.25	4.60	90.89	1.42	1,84	....	8.18	15,872
INDIANA COUNTY.																
Clymer, 1½ miles north of; Empire M (McKean) mine; Lower Kittanning or B bed; corner of pillar of room 44, off right entry 1, off main entry, 3-foot 3½-inch cut).	299	W19682	A	1	2.2	26.4	64.6	6.8	1.95	....	....	....	....	....	7,956	14,320
				2	....	27.0	66.0	7.0	2.00	....	....	....	....	....	8,133	14,640
				3	....	29.0	71.0	....	2.15	....	....	....	....	....	8,744	15,740
Same (face of room 56, off right entry 2, off main entry, 4,000 feet east of mine mouth, 3-foot 2¾-inch bed and cut).	299	W19678	A	1	1.9	26.5	64.0	7.6	2.40	....	....	....	....	....	7,911	14,240
				2	....	27.0	65.2	7.8	2.50	....	....	....	....	....	8,061	14,510
				3	....	29.3	70.7	....	2.71	....	....	....	....	....	8,744	15,740
Same (corner of crosscut at room 40, off right entry 2, off main entry, 3,000 feet southeast of mine mouth, 3-foot 2½-inch cut).	299	W19680	A	1	2.1	26.5	63.4	8.0	2.25	....	....	....	....	....	7,811	14,060
				2	....	27.1	64.7	8.2	2.30	....	....	....	....	....	7,978	14,360
				3	....	29.5	70.5	....	2.51	....	....	....	....	....	8,694	15,650
Same (face of first parallel, off south heading, off right entry 1, off main entry, 2,800 feet southwest of mine mouth, 3-foot 5½-inch cut).	299	W19685	A	1	2.8	26.4	64.4	6.4	1.65	;......	....	....	....	....	7,917	14,250
				2	....	27.2	66.2	6.6	1.70	....	....	....	....	....	8,150	14,670
				3	....	29.1	70.9	....	1.82	....	....	....	....	....	8,728	15,710
Same (face of third parallel, off south heading, off right entry 1, off main entry, 3-foot 3½-inch cut).	299	W19687	A	1	2.5	26.9	63.9	6.7	1.70	....	....	....	....	....	7,917	14,250
				2	....	27.6	65.5	6.9	1.75	....	....	....	....	....	8,122	14,620
				3	....	29.6	70.4	....	1.88	....	....	....	....	....	8,722	15,700
Same (composite of samples W19678, W19680, W19685, and W19687).	299	12223	A	1	1.30	26.70	64.40	7.60	2.03	5.02	80.35	1.36	3.64	....	7,953	14,315
				2	....	27.05	65.25	7.70	2.06	4.94	81.41	1.38	2.51	....	8,058	14,504
				3	....	29.31	70.69	....	2.23	5.35	88.20	1.50	2.72	....	8,730	15,714
1 mile north of; Empire R mine, Lower Kittanning or B bed (pillar at room 28, in left entry 2, off main entry, 2,200 feet east of mine mouth, 3-foot 2-inch bed and cut).	300	W19681	A	1	2.1	26.5	64.9	6.5	1.70	....	....	....	....	....	7,972	14,350
				2	....	27.1	66.3	6.6	1.75	....	....	....	....	....	8,139	14,650
				3	....	29.0	71.0	....	1.87	....	....	....	....	....	8,717	15,690
Same (crosscut at corner of room 47, off left entry 1, off main entry, 4,700 feet southeast of mine mouth, 3-foot 2¼-inch bed, 3-foot 1⅝-inch cut).	300	W19679	A	1	2.3	26.6	63.3	7.8	2.70	....	....	....	....	....	7,839	14,110
				2	....	27.2	64.8	8.0	2.75	....	....	....	....	....	8,022	14,440
				3	....	29.6	70.4	....	2.99	....	....	....	....	....	8,722	15,700
Same (face of room 3, off right entry 1, off main entry, 1,100 feet south of mine mouth, 3-foot 10¾-inch bed and cut).	300	W19683	A	1	2.0	26.6	64.4	7.0	1.90	....	....	....	....	....	7,961	14,330
				2	....	27.1	65.8	7.1	1.95	....	....	....	....	....	8,122	14,620
				3	....	29.2	70.8	....	2.10	....	....	....	....	....	8,750	5,750
Same (face of room 11, left entry 5, off main entry, 1,900 feet southeast of mine mouth, 3-foot 2⅝-inch bed and cut).	300	W19684	A	1	2.2	26.0	64.5	7.3	2.15	....	....	....	....	....	7,850	14,130
				2	....	26.6	65.9	7.5	2.20	....	....	....	....	....	8,022	14,440
				3	....	28.8	71.2	....	2.38	....	....	....	....	....	8,672	15,610
Same (near face of room 43, off left entry 4, off main entry, 3,100 feet from mine mouth, 3-foot 2½-inch bed and cut).	300	W19686	A	1	1.9	26.6	64.7	6.8	2.15	....	....	....	....	....	7,972	14,350
				2	....	27.1	66.0	6.9	2.20	....	....	....	....	....	8,128	14,630
				3	....	29.1	70.9	....	2.36	....	....	....	....	....	8,728	15,710
Same (composite of samples W19679, W19683, W19684, and W19686).	300	12219	A	1	1.23	26.37	64.84	7.56	2.41	4.99	79.89	1.37	3.78	....	7,943	14,297
				2	....	26.70	65.65	7.65	2.44	4.91	80.89	1.39	2.72	....	8,042	14,476
				3	....	28.91	71.09	....	2.64	5.32	87.59	1.51	2.94	....	8,708	15,674

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Portage, 2½ miles east of; Plymouth No. 1 mine, Upper Kittanning or C' bed (face of dip 5, 300 feet from main heading, 3-foot 7½-inch bed, 3-foot 7-inch cut).	279	W17110	A	1	3.3	17.5	72.6	6.6	.70						7,861	14,150
				2		18.0	75.2	6.8	.75						8,133	14,640
				3		19.3	80.7		.80						8,728	15,710
Same (face of left entry 4, 2,000 feet from main heading, 3-foot 8½-inch bed, 3-foot 8-inch cut).	279	W17111	A	1	3.5	17.5	70.7	8.3	1.50						7,694	13,850
				2		18.1	73.3	8.6	1.60						7,972	14,350
				3		19.8	80.2		1.75						8,722	15,700
Same (face of main heading, 250 feet from left entry 8, 3-foot ½-inch cut).	279	W17112	A	1	3.3	16.5	72.9	7.3	.70						7,817	14,070
				2		17.0	75.5	7.5	.70						8,083	14,550
				3		18.4	81.6		.76						8,739	15,730
Same (composite samples of W17110, W17111, and W17112).	279	11705	A	1	1.95			7.60	1.05	4.45	78.35	1.18	7.37			
				2				7.75	1.07	4.31	79.91	1.20	5.76			
				3					1.16	4.67	86.62	1.30	6.25			
1 mile east of; Sonman No. 2 shaft, Lower Kittanning or B bed (pillar near face of room 23, first right off plane, 3,000 feet southeast of shaft, 3-foot 11-inch bed, 3-foot 7-inch cut).	280	W19689	A	1	2.5	15.3	76.0	6.2	.65						8,039	14,470
				2		15.7	77.9	6.4	.65						8,239	14,830
				3		16.8	83.2		.69						8,800	15,840
Same (face of "No. 32 crook," off main north heading, 4-foot 3¾-inch bed, 3-foot 7¾-inch cut).	280	W19691	A	1	2.6	16.3	74.6	6.5	1.65						7,956	14,320
				2		16.7	76.6	6.7	1.70						8,172	14,710
				3		17.9	82.1		1.82						8,756	15,760
Same (face of "No. 28 crook," left heading 1, off dip, off main south heading, 3-foot 11¾-inch bed, 3-foot 4¾-inch cut).	280	W19692	A	1	2.5	15.9	76.1	5.5	.65						8,083	14,550
				2		16.3	78.1	5.6	.65						8,286	14,915
				3		17.3	82.7		.69						8,778	15,800
Same (composite of samples W19689, W19691, and W19692).	280	12217	A	1	1.43	16.92	75.22	6.43	1.08	4.50	83.40	1.30	3.29		8,087	14,557
				2		17.17	76.31	6.52	1.10	4.40	84.61	1.32	2.05		8,204	14,767
				3		18.37	81.63		1.18	4.71	90.51	1.41	2.19		8,776	15,797
Same (face of room 8, left heading 2, off dip off main south heading, 3-foot 3-inch bed, 3-foot ½-inch cut).	280	W19688	A	1	1.7	16.3	74.4	7.6	.65						7,933	14,280
				2		16.6	75.7	7.7	.65						8,072	14,530
				3		18.0	82.0		.70						8,744	15,740

Same (face of room 13, off left heading 1, off dip, off main south heading, 4-foot 4½-inch bed, 3-foot 10-inch cut).	280	W19690	A	1	2.5	15.4	76.6	5.5	1.15	......	......	......	......	......	8,094	14,570
				2	......	15.8	78.6	5.6	1.20	......	......	......	......	......	8,300	14,940
				3	......	16.7	83.3	......	1.27	......	......	......	......	......	8,794	15,830
Same (face of room 10, first right off plane 17, off main north heading, 4-foot 5-inch bed, 3-foot 9½-inch cut).	280	W19693	A	1	2.2	16.1	75.0	6.7	1.30	......	......	......	......	......	7,978	14,360
				2	......	16.5	76.6	6.9	1.35	......	......	......	......	......	8,156	14,680
				3	......	17.7	82.3	......	1.45	......	......	......	......	......	8,756	15,760
Same (face of right heading 3, off main dip, off main north heading, 4-foot 4-inch bed, 3-foot 10½-inch cut).	280	W19694	A	1	4.2	15.4	74.9	5.5	.65	......	......	......	......	......	7,967	14,340
				2	......	16.1	78.2	5.7	.70	......	......	......	......	......	8,317	14,970
				3	......	17.1	82.9	......	.74	......	......	......	......	......	8,817	15,870
Same (composite of samples W19688, W19690, W19693, and W19694).	280	12205	A	1	1.44	15.96	76.34	6.26	.98	4.44	83.35	1.19	3.78	......	8,101	14,582
				2	......	16.19	77.46	6.35	.99	4.34	84.57	1.21	2.54	......	8,219	14,794
				3	......	17.29	82.71	......	1.06	4.63	90.30	1.29	2.72	......	8,776	15,797
Spangler, 2 miles northwest of; Pennsylvania No. 21 mine, D or Lower Freeport bed (face of main heading, 5,000 feet southwest of mine mouth, 3-foot 9-inch bed, 3-foot 7-inch cut.	281	12144	A	1	4.54	22.74	67.62	5.10	.63	......	......	......	......	3.8	7,867	14,161
				2	......	23.82	70.84	5.34	.66	......	......	......	......	......	8,241	14,834
				3	......	25.16	74.84	......	.70	......	......	......	......	......	8,706	15,671
Same (face of second main entry, 6,500 feet from mine mouth, 3-foot 5½-inch bed, 2-foot 10-inch cut).	281	12145	A	1	2.40	25.79	65.82	5.99	1.18	......	......	......	......	1.4	7,965	14,337
				2	......	26.42	67.44	6.14	1.21	......	......	......	......	......	8,161	14,690
				3	......	28.15	71.85	......	1.29	......	......	......	......	......	8,695	15,651
Same (face of back heading 17, 5,900 feet west of mine mouth, 3-foot 6-inch bed, 2-foot 8½-inch cut).	281	12146	A	1	3.02	26.59	64.34	6.05	1.74	......	......	......	......	2.1	7,902	14,224
				2	......	27.42	66.34	6.24	1.79	......	......	......	......	......	8,148	14,666
				3	......	29.25	70.75	......	1.91	......	......	......	......	......	8,691	15,644
Same (face of right entry 4, off second main entry, 6,300 feet east of mine mouth, 3-foot 7½-inch bed, 3-foot 5-inch cut).	281	12147	A	1	2.65	26.48	62.86	8.01	1.56	......	......	......	......	1.9	7,771	13,988
				2	......	27.20	64.57	8.23	1.60	......	......	......	......	......	7,982	14,368
				3	......	29.64	70.36	......	1.74	......	......	......	......	......	8,698	15,656
Same (composite of samples 12144–12147..........	281	12148	A	1	3.14	26.01	64.40	6.45	1.37	5.08	79.55	1.30	6.25	2.3	7,866	14,159
				2	......	26.85	66.49	6.66	1.41	4.88	82.13	1.34	3.58	......	8,121	14,618
				3	......	28.77	71.23	......	1.51	5.23	87.99	1.44	3.83	......	8,701	15,662
2 miles northwest of; Pennsylvania No. 22 mine, Lower Freeport or D bed (face of ninth main entry, 7,200 feet southwest of mine mouth, 3-foot 9-inch cut).	281	12126	A	1	4.86	21.85	67.48	5.81	1.01	......	......	......	......	3.9	7,804	14,047
				2	......	22.97	70.92	6.11	1.06	......	......	......	......	......	8,208	14,765
				3	......	24.47	75.53	......	1.13	......	......	......	......	......	8,737	15,727
Same (rib of room 9, off right entry 4, off ninth main entry, 6,200 feet southwest of mine mouth, 3-foot 7½-inch cut).	281	12127	A	1	2.55	22.53	68.11	6.81	1.20	......	......	......	......	1.7	7,894	14,209
				2	......	23.12	69.89	6.99	1.23	......	......	......	......	......	8,101	14,582
				3	......	24.86	75.14	......	1.32	......	......	......	......	......	8,710	15,678
Same (face of room 17, off right entry 6, off ninth main entry, 7,400 feet from mine mouth, 3-foot 9¾-inch cut).	281	12140	A	1	2.83	23.17	67.53	6.47	1.57	......	......	......	......	2.3	7,821	14,078
				2	......	23.84	69.50	6.66	1.62	......	......	......	......	......	8,049	14,488
				3	......	25.54	74.46	......	1.74	......	......	......	......	......	8,624	15,523
Same (face of main back heading, 7,100 feet from mine mouth, 3-foot 2-inch cut).	281	12141	A	1	2.70	23.73	66.86	6.71	1.39	......	......	......	......	2.0	7,904	14,227
				2	......	24.39	68.71	6.90	1.43	......	......	......	......	......	8,123	14,621
				3	......	26.20	73.80	......	1.54	......	......	......	......	......	8,725	15,705
Same (face of fourteenth right, off main heading, 7,200 feet from mine mouth, 3-foot 1½-inch cut).	281	12142	A	1	2.61	23.74	67.62	6.03	.95	......	......	......	......	1.9	7,968	14,342
				2	......	24.38	69.43	6.19	.98	......	......	......	......	......	8,182	14,728
				3	......	25.99	74.01	......	1.04	......	......	......	......	......	8,722	15,700
Same (composite of samples 12126, 12127, 12140, 12141, and 12142).	282	12143	A	1	3.08	22.77	67.60	6.55	1.24	4.90	80.54	1.36	5.41	2.4	7,856	14,141
				2	......	23.49	69.75	6.76	1.28	4.71	83.10	1.40	2.75	......	8,106	14,591
				3	......	25.19	74.81	......	1.37	5.05	89.12	1.50	2.96	......	8,694	15,649

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Twin Rocks, near; Big Bend No. 1 mine, Miller, B, or Lower Kittanning bed (face of room 11, off drainage heading, 2,600 feet northeast of main opening, 3-foot 7½-inch bed and cut).	282	W19668	A	1	2.4	20.0	71.7	5.9	1.70	....	....	....	....	....	8,094	14,570
				2	....	20.5	73.5	6.0	1.75	....	....	....	....	....	8,294	14,930
				3	....	21.8	78.2	....	1.86	....	....	....	....	....	8,828	15,890
Same (face of water-level heading, 2,000 feet northeast of main opening, 3-foot 6⅜-inch bed and cut).	282	W19669	A	1	1.9	19.7	72.4	6.0	.60	....	....	....	....	....	8,078	14,540
				2	....	20.1	73.8	6.1	.60	....	....	....	....	....	8,239	14,830
				3	....	21.4	78.6	....	.64	....	....	....	....	....	8,772	15,790
Same (composite of samples W19668 and W19669)	282	12215	A	1	1.29	20.56	72.15	6.00	1.74	4.75	82.96	1.33	3.22	....	8,122	14,620
				2	....	20.83	73.09	6.08	1.76	4.67	84.05	1.35	2.09	....	8,228	14,810
				3	....	22.18	77.82	....	1.87	4.97	89.49	1.44	2.23	....	8,760	15,768
Same (pillar opposite room 16, off Claghorn section, 3,000 feet east of main opening, 3-foot 7½-inch bed).	282	W19672	A	1	2.0	19.5	73.0	5.5	1.50	....	....	....	....	....	8,078	14,540
				2	....	19.9	74.5	5.6	1.55	....	....	....	....	....	8,244	14,840
				3	....	21.1	78.9	....	1.64	....	....	....	....	....	8,739	15,730
Same (pillar to left of main heading, 2,500 feet east of main opening, 3-foot 7⅜-inch bed).	282	W19674	A	1	2.2	20.0	72.7	5.1	1.45	....	....	....	....	....	8,156	14,680
				2	....	20.5	74.3	5.2	1.50	....	....	....	....	....	8,344	15,020
				3	....	21.6	78.4	....	1.58	....	....	....	....	....	8,800	15,840
Same (pillar to left of main heading, opposite right headings 6 and 7, 2,000 feet east of main opening, 4-foot 3-inch bed and cut).	282	W19676	A	1	2.6	20.3	72.2	4.9	1.15	....	....	....	....	....	8,106	14,590
				2	....	20.8	74.2	5.0	1.20	....	....	....	....	....	8,322	14,980
				3	....	21.9	78.1	....	1.26	....	....	....	....	....	8,761	15,770
Same (composite of samples W19672, W19674, and W19676).	282	12208	A	1	1.32	20.33	73.01	5.34	1.39	4.66	83.45	1.33	3.83	....	8,201	14,762
				2	....	20.60	73.99	5.41	1.41	4.57	84.57	1.35	2.69	....	8,311	14,959
				3	....	21.78	78.22	....	1.49	4.83	89.41	1.43	2.84	....	8,786	15,815
1 mile north of; Big Bend No. 6 mine, Miller, B, or Lower Kittanning bed (room 1, left side of right heading 1, off No. 9 drift, 900 feet south of No. 9 opening, 3-foot 9½-inch bed, 3-foot 9¼-inch cut).	283	W19670	A	1	2.7	18.9	72.6	5.8	1.75	....	....	....	....	....	8,017	14,430
				2	....	19.4	74.6	6.0	1.80	....	....	....	....	....	8,239	14,830
				3	....	20.6	79.4	....	1.91	....	....	....	....	....	8,767	15,780
Same (face of heading 13, 325 feet from outcrop and 3,500 feet southeast from main opening, 3-foot 9¾-inch bed, 3-foot 8½-inch cut).	283	W19678	A	1	3.2	18.7	73.2	4.9	.95	....	....	....	....	....	8,011	14,420
				2	....	19.3	75.6	5.1	1.00	....	....	....	....	....	8,278	14,900
				3	....	20.3	79.7	....	1.05	....	....	....	....	....	8,722	15,700
Same (face of room 10 on left side of No. 9 drift, 700 feet southwest of No. 9 opening, 3-foot 8½-inch bed, 3-foot 8¼-inch cut).	283	W19677	A	1	2.6	20.0	72.1	5.3	1.00	....	....	....	....	....	8,061	14,510
				2	....	20.5	74.1	5.4	1.05	....	....	....	....	....	8,272	14,890
					....	21.7	78.3	....	1.11	....	....	....	....	....	8,739	15,730

Same (composite of samples W19670, W 19673, and W19677).	283	12227	A	1	1.54	20.67	72.37	5.42	1.40	4.73	83.20	1.38	3.87	......	8,138	14,648
				2	......	20.99	73.51	5.50	1.42	4.63	84.50	1.40	2.55	......	8,265	14,877
				3	......	22.21	77.79	......	1.50	4.90	89.42	1.48	2.70	......	8,746	15,743
Same (pillar between rooms 23 and 24, off main heading, 1,500 feet northeast of main opening, 3-foot 7¼-inch bed).	283	W19671	A	1	2.8	19.4	72.5	5.3	1.10	......	......	......	......	......	8,006	14,410
				2	......	19.9	74.7	5.4	1.15	......	......	......	......	......	8,233	14,820
				8	......	21.0	79.0	......	1.22	......	......	......	......	......	8,717	15,690
Same (pillar between rooms 27 and 28, off main heading, 2,000 feet northeast of main opening, 3-foot 9½-inch cut).	283	W19675	A	1	2.2	19.7	72.7	5.4	1.10	......	......	......	......	......	8,106	14,590
				2	......	20.1	74.4	5.5	1.15	......	......	......	......	......	8,289	14,920
				3	......	21.3	78.7	......	1.22	......	......	......	......	......	8,767	15,780
Same (composite of samples W19671 and W19675)..	283	12207	A	1	1.47	20.18	72.91	5.44	1.20	4.75	83.43	1.27	3.91	......	8,169	14,704
				2	......	20.48	74.00	5.52	1.22	4.66	84.67	1.29	2.64	......	8,291	14,924
				3	......	21.68	78.32	......	1.29	4.93	89.61	1.37	2.80	......	8,775	15,795
Commercial No. 3 mine, Miller or B bed (face of main heading, 6,200 feet northwest of mine mouth, 5-foot ½-inch bed, 3-foot 10½-inch cut).	284	W20477	A	1	3.1	18.3	71.6	7.0	2.10	......	......	......	......	......	7,867	14,160
				2	......	18.9	73.9	7.2	2.15	......	......	......	......	......	8,117	14,610
				3	......	20.4	79.6	......	2.32	......	......	......	......	......	8,744	15,740
Same (face of right heading 7, 450 feet off main entry and 5,600 feet northwest of mine mouth, 3-foot 6¼-inch bed, 3-foot 6-inch cut).	284	W20479	A	1	2.1	20.2	70.4	7.3	2.75	......	......	......	......	......	7,906	14,230
				2	......	20.6	71.9	7.5	2.80	......	......	......	......	......	8,072	14,530
				3	......	22.3	77.7	......	3.03	......	......	......	......	......	8,728	15,710
Same (face of left entry 7, 700 feet off main entry and 5,300 feet northwest of mine mouth, 3-foot 11⅞-inch bed and cut).	284	W20480	A	1	2.9	17.8	71.4	7.9	.60	......	......	......	......	......	8,028	14,450
				2	......	18.3	73.6	8.1	.60	......	......	......	......	......	8,267	14,880
				3	......	19.9	80.1	......	.65	......	......	......	......	......	8,800	15,840
Same (face of right heading 5, 5,000 feet off main entry and 4,200 feet north of mine mouth, 3-foot 8⅛-inch bed and cut).	284	W20481	A	1	2.2	19.4	72.7	5.7	1.80	......	......	......	......	......	8,100	14,580
				2	......	19.8	74.4	5.8	1.85	......	......	......	......	......	8.278	14,900
				3	......	21.0	79.0	......	1.96	......	......	......	......	......	8,789	15,820
Same (composite of samples W20477, W20479, W20480, and W20481).	284	12377	A	1	.82	20.42	72.11	6.65	2.35	4.50	81.89	1.22	3.39	......	8,083	14,549
				2	......	20.59	72.70	6.71	2.37	4.45	82.57	1.23	2.67	......	8,150	14,670
				3	......	22.07	77.93	......	2.54	4.77	88.51	1.32	2.86	......	8,736	15,725
Same (pillar on right heading 4, 5,000 feet off main entry and 4,500 feet northeast of mine mouth, 3-foot 6-inch bed, 3-foot 5⅜-inch cut).	284	W20478	A	1	1.8	20.0	71.5	6.7	1.70	......	......	......	......	......	8,011	14.420
				2	......	20.4	72.8	6.8	1.75	......	......	......	......	......	8,156	14,680
				3	......	21.9	78.1	......	1.88	......	......	......	......	......	8,756	15,760
Commercial No. 4 mine, lower Kittanning or B bed (face of main entry 3, off No. 4 slope, 3-foot 7½-inch bed and cut).	285	W17061	A	1	3.3	17.0	73.9	5.8	1.60	......	......	......	......	......	7,978	14,360
				2	......	17.5	76.5	6.0	1.70	......	......	......	......	......	8,250	14,850
				3	......	18.6	81.4	......	1.81	......	......	......	......	......	8,778	15,800
Same (100 feet from face of main heading, 5,200 feet north of mine mouth, 3-foot 8-inch bed and cut).	285	W20501	A	1	3.8	18.1	71.9	6.2	1.85	......	......	......	......	......	7,878	14.180
				2	......	18.8	74.8	6.4	1.95	......	......	......	......	......	8.189	14,740
				3	......	20.1	79.9	......	2.08	......	......	......	......	......	8,744	15,740
Same (face of east entry 6, off right entry 3, off main entry, 4,000 feet northeast of mine mouth, 3-foot 8⅜-inch bed and cut).	285	W20505	A	1	2.4	19.0	72.3	6.3	1.85	......	......	......	......	......	7,956	14,320
				2	......	19.5	74.0	6.5	1.90	......	......	......	......	......	8,156	14,680
				3	......	20.9	79.1	......	2.03	......	......	......	......	......	8,722	15,700
Same (rib near face of east entry 7, off right entry 3, off main entry, 4,800 feet northeast of mine mouth, 3-foot 10⅜-inch bed and cut).	285	W20506	A	1	3.0	18.3	73.1	5.6	1.45	......	......	......	......	......	8,011	14,420
				2	......	18.9	75.3	5.8	1.50	......	......	......	......	......	8,256	14,860
				3	......	20.1	79.9	......	1.59	......	......	......	......	......	8,761	15,770
Same (face of right entry 3, off main entry, 4,900 feet northeast of mine mouth, 3-foot 7-inch cut).	285	W20508	A	1	3.3	18.2	72.9	5.6	1.75	......	......	......	......	......	7,961	14,330
				2	......	18.8	75.4	5.8	1.80	......	......	......	......	......	8,239	14,830
				3	......	20.0	80.0	......	1.91	......	......	......	......	......	8,750	15,750
Same (composite of samples W20501, W20505, W20506, and W20508).	285	12364	A	1	.89	19.73	73.30	6.08	1.95	4.58	83.16	1.28	2.95	......	8,090	14.562
				2	......	19.91	73.96	6.13	1.97	4.52	83.91	1.29	2.18	......	8,163	14,693
				3	......	21.21	78.79	......	2.10	4.82	89.39	1.37	2.32	......	8,696	15,653
Same (sixth pillar on east entry 9, off right entry 2, off main entry, 3,800 feet northeast of mine mouth, 3-foot 4½-inch cut).	285	W20503	A	1	2.8	18.6	72.9	5.7	1.55	......	......	......	......	......	7,983	14.370
				2	......	19.1	75.0	5.9	1.60	......	......	......	......	......	8,217	14,790
				3	......	20.3	79.7	......	1.70	......	......	......	......	......	8,733	15,720

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CAMBRIA COUNTY—continued.																
Twin Rocks, 1 mile north of; Commercial No. 5 mine, Miller or B bed (face of room 1, off east heading 10, off main entry, 3-foot 8-inch bed).	286	W17062	A	1	3.3	20.0	72.3	4.4	.90	......	......	......	......	......	8,106	14,590
				2	......	20.5	74.9	4.6	.95	......	......	......	......	......	8,383	15,090
				3	......	21.5	78.5	......	1.00	......	......	......	......	......	8,789	15,820
Same (face of main heading, 4,700 feet northeast of mine mouth, 3-foot 7½-inch bed and cut).	286	W20499	A	1	2.8	18.8	72.4	6.0	2.05	......	......	......	......	......	7,972	14,350
				2	......	19.3	74.5	6.2	2.10	......	......	......	......	......	8,200	14,760
				3	......	20.6	79.4	......	2.24	......	......	......	......	......	8,744	15,740
Same (face of room 7, off right heading 1; 500 feet from second parallel entry and 3,400 feet northeast of mine mouth; 3-foot 5½-inch cut).	286	W20500	A	1	2.9	19.2	72.1	5.8	1.70	......	......	......	......	......	8,028	14,450
				2	......	19.8	74.2	6.0	1.75	......	......	......	......	......	8,267	14,880
				3	......	21.1	78.9	......	1.86	......	......	......	......	......	8,794	15,830
Same (face of east heading 12; 325 feet off main entry, 4,700 feet northeast of mine mouth, 3-foot 8½-inch bed, 3 foot 8 inch cut).	286	W20502	A	1	2.9	18.9	72.5	5.7	2.00	......	......	......	......	......	8,000	14,400
				2	......	19.5	74.6	5.9	2.05	......	......	......	......	......	8,239	14,830
				3	......	20.7	79.3	......	2.18	......	......	......	......	......	8,756	15,760
Same (east heading 11; 800 feet off main entry, 4,000 feet northeast of mine mouth, 3-foot 6¾-inch cut).	286	W20504	A	1	2.8	19.2	71.3	6.7	2.15	......	......	......	......	......	7,956	14,320
				2	......	19.7	73.4	6.9	2.20	......	......	......	......	......	8,183	14,730
				3	......	21.2	78.8	......	2.36	......	......	......	......	......	8,789	15,820
Same (face of room 10, off east heading 10; 750 feet off main heading, 3,900 feet northeast of mine mouth, 3-foot 8½-inch bed and cut).	286	W20507	A	1	3.2	18.2	71.9	6.7	2.15	......	......	......	......	......	7,894	14,210
				2	......	18.8	74.3	6.9	2.25	......	......	......	......	......	8,156	14,680
				3	......	20.2	79.8	......	2.42	......	......	......	......	......	8,761	15,770
Same (composite of samples W20499, W20500, W20502, W20504, and W20507).	286	12375	A	1	.81	20.78	72.10	6.31	2.24	4.60	82.42	1.21	3.22	......	8,073	14,531
				2	......	20.95	72.69	6.36	2.26	4.55	83.10	1.22	2.51	......	8,139	14,650
				3	......	22.37	77.63	......	2.41	4.86	88.74	1.30	2.69	......	8,692	15,646
CENTER COUNTY.																
Osceola Mills, ½ mile south of; Electric mine, Lower Kittanning or B bed (face of right entry 1, off new drift entry, 4-foot 10-inch bed, 3-foot 9-inch cut).	287	12067	A	1	3.45	20.57	68.55	7.43	1.84	......	......	......	......	2.8	7,771	13,888
				2	......	21.30	71.00	7.70	1.91	......	......	......	......	......	8,048	14,486
				3.	......	23.08	76.92	......	2.07	......	......	......	......	......	8,719	15,694
Same (pillar at room 7, off right entry 4, off main heading, 4-foot 10½-inch bed, 3-foot 11½-inch cut).	287	12069	A	1	3.28	21.61	67.75	7.36	2.01	......	......	......	......	2.3	7,742	13,936
				2	......	22.34	70.05	7.61	2.08	......	......	......	......	......	8,004	14,407
				3	......	24.18	75.82	......	2.25	......	......	......	......	......	8,664	15,595
Same (pillar at room 5, off left entry 5, off main heading, 4-foot 10½-inch bed, 3-foot 9½-inch cut).	287	12070	A	1	3.09	20.73	67.90	8.28	2.53	......	......	......	......	2.4	7,654	13,777
				2	......	21.39	70.07	8.54	2.61	......	......	......	......	......	7,898	14,216
				3	......	23.39	76.61	......	2.85	......	......	......	......	......	8,636	15,545

Same (composite of samples 12067, 12069, and 12070).	287	12071	A	1	3.24	20.65	68.34	7.77	2.08	4.89	77.85	1.25	6.16	2.5	7,726	13,907
				2	......	21.34	70.63	8.03	2.15	4.68	80.46	1.29	3.39	......	7,985	14,373
				3	......	23.20	76.80	......	2.34	5.09	87.48	1.40	3.69	......	8,682	15,626
Same (pillar at room 9, off left entry 5, off main entry, 5-foot 2½-inch bed, 4 foot 1½-inch cut).	287	12068	A	1	3.48	20.26	68.55	7.71	1.63	4.82	78.45	1.26	6.13	2.5	7,715	13,887
				2	......	20.99	71.02	7.99	1.69	4.59	81.28	1.31	3.14	......	7,994	14,389
				3	......	22.81	77.19	......	1.84	4.99	88.34	1.42	3.41	......	8,688	15,638
Same (composite of samples 12068, 12069, and 12070).	287	12190	A	1	3.04	22.92	67.32	6.72	1.54	4.95	79.49	1.33	5.97	2.4	7,851	14,132
				2	......	23.64	69.43	6.93	1.59	4.75	81.99	1.37	3.37	......	8,098	14,576
				3	......	25.40	74.60	......	1.71	5.10	88.10	1.47	3.62	......	8,701	15,662
¼ mile north of; Moshannon No. 10 mine, Lower Kittanning or B bed (on rib of room 18, off left entry 2, off right entry 4, 40 feet from left entry 2, 4-foot 6¾-inch bed, 3-foot 7½-inch cut).	287	12057	A	1	2.59	21.53	68.38	7.50	1.91	......	......	......	......	1.9	7,835	14,103
				2	......	22.10	70.20	7.70	1.96	......	......	......	......	......	8,043	14,477
				3	......	23.94	76.06	......	2.12	......	......	......	......	......	8,714	15,685
Same (face of room 21, off left main heading, 4-foot 5-inch bed, 3-foot 8½-inch cut).	287	12058	A	1	2.67	22.09	68.48	6.76	2.00	......	......	......	......	2.1	7,883	14,189
				2	......	22.70	70.35	6.95	2.05	......	......	......	......	......	8,099	14,578
				3	......	24.40	75.60	......	2.20	......	......	......	......	......	8,704	15,667
Same (face of room 16, off right main heading, near first crosscut to right, 4-foot 3¾-inch bed, 3-foot 8¼-inch cut).	287	12059	A	1	2.75	19.59	70.35	7.31	2.11	......	......	......	......	2.2	7,839	14,110
				2	......	20.14	72.34	7.52	2.17	......	......	......	......	......	8,061	14,510
				3	......	21.78	78.22	......	2.35	......	......	......	......	......	8,716	15,689
Same (face of right heading 1, 3-foot 8¾-inch bed, 3-foot 3-inch cut).	287	12060	A	1	3.80	19.98	69.09	7.13	1.73	......	......	......	......	3.2	7,779	14,002
				2	......	20.77	71.82	7.41	1.80	......	......	......	......	......	8,086	14,555
				3	......	22.43	77.57	......	1.94	......	......	......	......	......	8,733	15,719
Same (face of room 16, off right heading 2, 3-foot 5½-bed, 2-foot 6¾-inch cut).	287	12061	A	1	3.03	21.25	67.95	7.77	2.48	......	......	......	......	2.5	7,750	13,950
				2	......	21.91	70.08	8.01	2.56	......	......	......	......	......	7,992	14,386
				3	......	23.82	76.18	......	2.78	......	......	......	......	......	8,688	15,638
Same (composite of samples 12057–12061)..........	288	12062	A	1	3.05	21.65	67.95	7.35	1.98	4.99	79.16	1.31	5.21	2.4	7,798	14,036
				2	......	22.33	70.09	7.58	2.04	4.80	81.65	1.35	2.58	......	8,044	14,479
				3	......	24.16	75.84	......	2.21	5.19	88.35	1.46	2.79	......	8,704	15,667
½ mile south of; Weston mine, Lower Kittanning or B bed (face of left entry 4, off main entry, 4-foot 5⅞-inch bed, 3-foot 7½-inch cut).	288	12063	A	1	3.20	19.52	68.74	8.54	2.35	......	......	......	......	2.5	7,664	13,795
				2	......	20.17	71.01	8.82	2.43	......	......	......	......	......	7,918	14,252
				3	......	22.12	77.88	......	2.66	......	......	......	......	......	8,684	15,631
Same (rib of main heading, at left entry 7, 5-foot 1½-inch bed, 4-foot 4-inch cut).	288	12064	A	1	2.91	19.53	70.53	7.03	1.86	......	......	......	......	2.2	7,832	14,098
				2	......	20.12	72.64	7.24	1.92	......	......	......	......	......	8,067	14,521
				3	......	21.69	78.31	......	2.07	......	......	......	......	......	8,697	15,655
Same (composite of samples 12063 and 12064).....	288	12066	A	1	3.08	20.27	68.83	7.82	2.11	4.85	79.00	1.28	4.94	2.4	7,754	13,957
				2	......	20.91	71.02	8.07	2.18	4.65	81.51	1.32	2.27	......	8,001	14,402
				3	......	22.75	77.25	......	2.37	5.06	88.67	1.44	2.46	......	8,703	15,665
Same (pillar at room 8, off right entry 1, off main heading, 4-foot 7-inch cut).	288	12065	A	1	2.94	19.89	69.68	7.49	1.93	4.98	79.67	1.23	4.70	2.2	7,822	14,080
				2	......	20.49	71.79	7.72	1.99	4.79	82.08	1.27	2.15	......	8,059	14,506
				3	......	22.21	77.79	......	2.16	5.19	88.95	1.38	2.32	......	8,734	15,721
CLEARFIELD COUNTY.																
Boardman; Potts Run No. 3 mine, Lower Kittanning or B bed (face of right entry 3, off main entry, 1,400 feet southwest of mine mouth, 3-foot 7½-inch bed, 3-foot 6½-inch cut).	289	12093	A	1	2.74	20.75	67.85	8.66	1.59	......	......	......	......	2.0	7,724	13,903
				2	......	21.34	69.76	8.90	1.63	......	......	......	......	......	7,942	14,296
				3	......	23.42	76.58	......	1.79	......	......	......	......	......	8,718	15,692
Same (on rib, 10 feet from face of right entry 4, off main entry 1,700 feet southwest of mine mouth, 3-foot 7⅞-inch bed, 3-foot 7¼-inch cut).	289	12094	A	1	2.37	20.82	68.14	8.67	1.53	......	......	......	......	1.7	7,749	13,948
				2	......	21.33	69.79	8.88	1.57	......	......	......	......	......	7,937	14,287
				3	......	23.41	76.59	......	1.72	......	......	......	......	......	8,711	15,680
Same (face of left entry 3, off main entry, 1,680 feet northeast of mine mouth, 3-foot 6¼-inch bed, 3-foot 6-inch cut).	289	12095	A	1	2.80	20.65	67.69	8.86	1.38	......	......	......	......	2.1	7,719	13,894
				2	......	21.24	69.64	9.12	1.42	......	......	......	......	......	7,941	14,294
				3	......	23.37	76.63	......	1.56	......	......	......	......	......	8,738	15,728

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CLEARFIELD COUNTY—continued.																
Boardman; Potts Run No. 3 mine, Lower Kittanning or B bed (face of main heading, 2,000 feet north of mine mouth, 3-foot 11½-inch bed, 3-foot 9-inch cut).	289	12096	A	1	3.38	20.50	67.75	8.37	1.08	......	......	......	......	2.7	7,697	13,855
				2	......	21.22	70.12	8.66	1.12	......	......	......	......	......	7,966	14,339
				3	......	23.23	76.77	......	1.23	......	......	......	......	......	8,721	15,698
Same (face of room 12, off right entry 2, off main entry, 3-foot 7¼-inch bed, 3-foot 5¾-inch cut).	289	12097	A	1	3.04	21.22	66.83	8.91	1.41	......	......	......	......	2.3	7,698	13,856
				2	......	21.89	68.92	9.19	1.45	......	......	......	......	......	7,940	14,292
				3	......	24.11	75.89	......	1.60	......	......	......	......	......	8,744	15,739
Same (composite of samples 12093–12097)	289	12098	A	1	2.95	21.29	66.92	8.84	1.35	4.74	78.51	1.19	5.37	2.2	7,723	13,901
				2	......	21.94	68.95	9.11	1.39	4.54	80.90	1.23	2.83	......	7,958	14,324
				3	......	24.14	75.86	......	1.53	4.99	89.01	1.35	3.12	......	8,755	15,759
Brisbin, 1 mile southwest of; Lenore No. 1 mine, Upper Freeport or E bed (face of left heading, off main heading, off No. 1 drift, 2-foot 7-inch bed and cut).	290	12046	A	1	3.00	21.59	67.61	7.80	.79	......	......	......	......	1.9	7,786	14,015
				2	......	22.26	69.70	8.04	.81	......	......	......	......	......	8,027	14,449
				3	......	24.21	75.79	......	.88	......	......	......	......	......	8,729	15,712
1 mile southwest of; Lenore No. 2 mine, Upper Freeport or E bed (face of left heading 3, off No. 2 drift, 2 foot 7½-inch bed and cut).	290	12044	A	1	3.14	21.77	66.12	8.97	1.08	......	......	......	......	2.0	7,690	13,842
				2	......	22.48	68.26	9.26	1.11	......	......	......	......	......	7,939	14,290
				3	......	24.77	75.23	......	1.22	......	......	......	......	......	8,749	15,748
Same (face of left heading 2, off No. 2 drift, 2-foot 6¾-inch bed and cut).	290	12045	A	1	2.83	21.78	67.89	7.50	.91	4.92	79.68	1.33	5.66	1.9	7,882	14,188
				2	......	22.41	69.87	7.72	.94	4.74	82.00	1.37	3.23	......	8,111	14,600
				3	......	24.29	75.71	......	1.02	5.14	88.86	1.48	3.50	......	8,788	15,818
Same (composite of samples 12044 and 12046)	290	12047	A	1	3.07	21.47	67.26	8.20	.94	4.80	78.48	1.23	6.35	2.0	7,757	13,963
				2	......	22.15	69.39	8.46	.97	4.60	80.97	1.27	3.73	......	8,003	14,405
				3	......	24.20	75.80	......	1.06	5.03	88.45	1.39	4.07	......	8,742	15,736
Madera, 1 mile east of; Morgan Run mine, Lower Kittanning or B bed (face of south entry 4, off right entry 5, 3-foot 5½-inch bed, 3-foot 4½-inch cut).	290	W20402	A	1	1.6	18.6	69.4	10.4	2.55	......	......	......	......	......	7,678	13,820
				2	......	18.9	70.5	10.6	2.60	......	......	......	......	......	7,800	14,040
				3	......	21.1	78.9	......	2.91	......	......	......	......	......	8,722	15,700
2 miles north of; Bucher mine, Lower Kittanning or B bed (left entry 9, off cross heading 1, 3,400 feet northeast of mouth of main entry, 3-foot 1¼-inch bed, 2-foot 11-inch cut).	290	W20435	A	1	2.3	18.7	68.5	10.5	1.00	......	......	......	......	......	7,606	13,690
				2	......	19.1	70.2	10.7	1.05	......	......	......	......	......	7,783	14,010
				3	......	21.4	78.6	......	1.18	......	......	......	......	......	8,711	15,680
Same (face of right heading 8, 3,800 feet northeast of mouth of main heading, 3-foot 3-inch bed, 3-foot 2½-inch cut).	290	W20436	A	1	2.1	20.0	68.9	9.0	1.45	......	......	......	......	......	7,767	13,980
				2	......	20.4	70.4	9.2	1.50	......	......	......	......	......	7,933	14,280
				3	......	22.5	77.5	......	1.65	......	......	......	......	......	8,739	15,730

Same (face of right heading 10, 4,500 feet northeast of mouth of main entry, 3-foot 3-inch bed, 3-foot 2-inch cut).	290	W 20437	A	1	2.4	19.9	68.2	9.5	1.20	......	......	......	......	......	7,700	13,860
				2	......	20.4	69.9	9.7	1.25	......	......	......	......	......	7,889	14,200
				3	......	22.6	77.4	......	1.38	......	......	......	......	......	8,739	15,730
Same (face of left heading 7, 3,000 feet northeast of mouth of main entry, 3-foot 6¾-inch bed, 3-foot 3½-inch cut).	290	W20438	A	1	2.3	19.5	70.2	8.0	1.15	......	......	......	......	......	7,839	14,110
				2	......	19.9	71.9	8.2	1.20	......	......	......	......	......	8,017	14,430
				3	......	21.7	78.3	......	1.31	......	......	......	......	......	8,733	15,720
Same (face of right heading 7, 3,200 feet northeast of mouth of main entry, 3-foot 1½-inch cut).	290	W20439	A	1	2.3	20.4	69.0	8.3	1.20	......	......	......	......	......	7,756	13,960
				2	......	20.9	70.6	8.5	1.25	......	......	......	......	......	7,939	14,290
				3	......	22.8	77.2	......	1.37	......	......	......	......	......	8,772	15,790
Same (composite of samples W 20435–W 20439......	290	12376	A	1	.80	21.36	68.48	9.36	1.38	4.49	80.07	1.09	3.61	......	7,790	14,022
				2	......	21.53	69.03	9.44	1.39	4.44	80.72	1.10	2.91	......	7,853	14,135
				3	......	23.77	76.23	......	1.53	4.90	89.13	1.21	3.23	......	8,671	15,608
Same (face of east entry 3, off right entry 5, 3-foot 3⅞-inch bed, 3-foot 2-inch cut).	290	W20403	A	1	1.9	19.1	68.5	10.5	1.75	......	......	......	......	......	7,606	13,690
				2	......	19.5	69.8	10.7	1.80	......	......	......	......	......	7,750	13,950
				3	......	21.8	78.2	......	2.02	......	......	......	......	......	8,678	15,620
Same (face of south entry 3, off right entry 5, 3-foot 7¾-inch bed, 3-foot 7½-inch cut).	290	W20404	A	1	2.0	18.2	67.4	12.4	3.15	......	......	......	......	......	7,411	13,340
				2	......	18.6	68.7	12.7	3.20	......	......	......	......	......	7,561	13,610
				3	......	21.3	78.7	......	3.67	......	......	......	......	......	8,661	15,590
Same (face of south entry 1, off right entry 5, 3-foot 5½-inch bed, 3-feot 3¾-inch cut).	290	W20405	A	1	2.0	18.7	68.3	11.0	3.30	......	......	......	......	......	7,550	13,590
				2	......	19.1	69.7	11.2	3.35	......	......	......	......	......	7,700	13,860
				3	......	21.5	78.5	......	3.77	......	......	......	......	......	8,672	15,610
Same (face of main heading, 3-foot 6½-inch bed, 3-foot 5½-inch cut).	290	W20406	A	1	1.9	17.3	70.5	10.3	3.50	......	......	......	......	......	7,589	13,660
				2	......	17.6	71.9	10.5	3.55	......	......	......	......	......	7,733	13,920
				3	......	19.7	80.3	......	3.97	......	......	......	......	......	8,639	15,550
Same (composite of samples W20402–W20406).....	291	12368	A	1	.82	20.40	67.86	10.92	3.08	4.34	77.62	1.08	2.96	......	7,630	13,734
				2	......	20.57	68.42	11.01	3.11	4.29	78.26	1.09	2.24	......	7,693	13,847
				3	......	23.11	76.89	......	3.49	4.82	87.94	1.22	2.53	......	8,645	15,561
Morrisdale; Morrisdale No. 1 mine, Middle Kittanning or C bed (face of sump heading, 3,100 feet from shaft No. 1, 5-foot bed, 4-foot 8¼-inch cut).	293	12079	A	1	3.47	20.49	67.30	8.74	1.09	......	......	......	......	2.8	7,644	13,759
				2	......	21.23	69.72	9.05	1.13	......	......	......	......	......	7,918	14,252
				3	......	23.34	76.66	......	1.24	......	......	......	......	......	8,706	15,671
Same (at face of McKinley heading, off Truck main heading, 5,850 feet from bottom of shaft No. 1, 4-foot 6-inch bed, 4-foot 5¼-inch cut).	293	12080	A	1	4.19	19.70	67.64	8.47	1.07	......	......	......	......	3.1	7,554	13,597
				2	......	20.56	70.60	8.84	1.12	......	......	......	......	......	7,884	14,191
				3	......	22.55	77.45	......	1.23	......	......	......	......	......	8,649	15,568
Same (composite samples 12079 and 12080)........	293	12082	A	1	3.86	19.79	67.54	8.81	1.08	4.85	77.44	1.29	6.53	3.0	7,565	13,617
				2	......	20.58	70.26	9.16	1.12	4.60	80.55	1.34	3.23	......	7,868	14,162
				3	......	22.65	77.35	......	1.23	5.06	88.67	1.48	3.56	......	8,661	15,590
Morrisdale No. 1 mine, Lower Kittanning or B bed (face of main heading, 5,240 feet from bottom of shaft No. 1, 4-foot ½-inch bed, 3-foot 7½-inch cut).	292	12074	A	1	4.15	20.01	66.83	9.01	3.77	......	......	......	......	3.7	7,526	13,547
				2	......	20.88	69.72	9.40	3.93	......	......	......	......	......	7,852	14,134
				3	......	23.05	76.95	......	4.34	......	......	......	......	......	8,667	15,601
Same (face of room 13, off right heading 8, off main heading, 5,040 feet from bottom of shaft No. 1, 4-foot 3⅝-inch bed, 3-foot 9⅝-inch cut).	292	12075	A	1	3.24	19.58	69.21	7.97	3.41	......	......	......	......	2.7	7,721	13,898
				2	......	20.24	71.52	8.24	3.52	......	......	......	......	......	7,980	14,364
				3	......	22.06	77.94	......	3.84	......	......	......	......	......	8,697	15,655
Same (face of room 45, off right entry 4, off main heading, 5,200 feet from bottom of shaft No. 1, 4-foot 1¾-inch bed, 3-foot 6¾-inch cut).	292	12081	A	1	3.19	21.88	67.24	7.69	3.06	......	......	......	......	2.7	7,746	13,943
				2	......	22.60	69.46	7.94	3.16	......	......	......	......	......	8,002	14,404
				3	......	24.55	75.45	......	3.43	......	......	......	......	......	8,692	15,646
Same (composite of samples 12074, 12075, and 12081)	293	12090	A	1	3.61	20.86	67.07	8.46	3.38	4.92	77.06	1.28	4.90	3.0	7,679	13,822
				2	......	21.64	69.58	8.78	3.51	4.69	79.95	1.33	1.74	......	7,967	14,341
				8	......	23.72	76.28	......	3.85	5.14	87.65	1.46	1.90	......	8,734	15,721
Morrisdale No. 2 mine, Middle Kittaning or C bed (face of heading 6, 4,500 feet from bottom of shaft No. 2, 4-foot 10½-inch bed, 4-foot 9-inch cut).	293	12072	A	1	3.04	19.42	67.38	10.16	1.00	......	......	......	......	2.3	7,548	13,586
				2	......	20.03	69.49	10.48	1.03	......	......	......	......	......	7,785	.4,013
				3	......	22.38	77.62	......	1.15	......	......	......	......	......	8,697	15,655

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CLEARFIELD COUNTY—continued.																
Morrisdale; Morrisdale No. 2 mine, Middle Kittanning or C bed (room 9, left heading 2, off Lucky strike, 2,200 feet from bottom of shaft No. 2, 4-foot 1-inch bed, 4-foot cut).	293	12073	A	1	2.52	20.73	66.96	9.79	1.74	......	......	......	......	2.0	7,640	13,752
				2	......	21.27	68.69	10.04	1.79	......	......	......	......	......	7,838	14,108
				3	......	23.64	76.36	......	1.99	......	......	......	......	......	8,715	15,683
Same (composite of samples 12072 and 12073)	293	12091	A	1	2.83	20.30	66.91	9.96	1.32	4.67	77.05	1.25	5.75	2.2	7,580	13,644
				2	......	20.89	68.86	10.25	1.36	4.49	79.29	1.29	3.32	......	7,801	14,042
				3	......	23.28	76.72	......	1.52	5.00	88.34	1.44	3.70	......	8,692	15,646
Morrisdale No. 3 mine, Lower Kittaning or B bed (room 5, off left heading 1, off new slope tunnel, 1,400 feet from shaft No. 3, 4-foot 1½-inch bed, 3-foot 6-inch cut).	292	12076	A	1	3.23	19.68	69.75	7.34	1.99	......	......	......	......	2.6	7,806	14,051
				2	......	20.34	72.07	7.59	2.06	......	......	......	......	......	8,067	14,521
				3	......	22.01	77.99	......	2.23	......	......	......	......	......	8,729	15,712
Same (face of shaft No. 2 heading, off main dip, 5,500 feet from shaft, 3-foot 8-inch bed, 2-foot 11½-inch cut).	292	12077	A	1	3.34	19.23	71.07	6.36	1.71	......	......	......	......	2.8	7,897	14,215
				2	......	19.90	73.52	6.58	1.77	......	......	......	......	......	8,170	14,706
				3	......	21.30	78.70	......	1.89	......	......	......	......	......	8,745	15,741
Same (face of Morrisdale heading, 6,300 feet from shaft, 3-foot 10½-inch bed, 3-foot 2½-inch cut).	292	12078	A	1	3.24	19.78	70.02	6.96	2.03	......	......	......	......	2.7	7,862	14,152
				2	......	20.44	72.37	7.19	2.10	......	......	......	......	......	8,125	14,623
				3	......	22.02	77.98	......	2.26	......	......	......	......	......	8,755	15,759
Same (composite samples 12076, 12077, and 12078)..	293	12092	A	1	3.30	20.13	69.57	7.00	1.91	4.66	79.93	1.36	5.14	2.7	7,862	14,152
				2	......	20.82	71.94	7.24	1.98	4.44	82.66	1.41	2.27	......	8,130	14,634
				3	......	22.45	77.55	......	2.13	4.79	89.12	1.52	2.44	......	8,765	15,777
Munson; Colorado No. 5 mine, Lower Kittaning or B bed (at face of left entry 6, off line heading, 4-foot 1-inch bed, 3-foot cut).	294	12109	A	1	3.74	19.77	69.68	6.81	1.66	......	......	......	......	2.7	7,835	14,103
				2	......	20.54	72.39	7.07	1.72	......	......	......	......	......	8,140	14,652
				3	......	22.10	77.90	......	1.85	......	......	......	......	......	8,759	15,766
Same (face of left entry 8, off main heading, 3,000 feet from drift mouth, 4-foot 2½-inch bed, 3-foot 2-inch cut).	294	12110	A	1	3.58	20.28	69.21	6.93	1.78	......	......	......	......	2.7	7,835	14,103
				2	......	21.03	71.78	7.19	1.85	......	......	......	......	......	8,126	14,627
				3	......	22.66	77.34	......	1.99	......	......	......	......	......	8,756	15,761
Same (face of main heading, 3,600 feet from drift mouth, 4-foot 4½-inch bed, 3-foot 2¾-inch cut).	294	12111	A	1	3.31	20.02	69.34	7.33	1.64	......	......	......	......	2.4	7,817	14,071
				2	......	20.70	71.72	7.58	1.70	......	......	......	......	......	8,084	14,551
				3	......	22.40	77.60	......	1.84	......	......	......	......	......	8,747	15,745
Same (face of line heading, 4-foot 3½-inch bed, 3-foot 2-inch cut).	294	12112	A	1	3.27	19.91	68.31	8.51	2.92	......	......	......	......	2.5	7,688	13,838
				2	......	20.58	70.62	8.80	3.02	......	......	......	......	......	7,948	14,306
				3	......	22.57	77.43	......	3.31	......	......	......	......	......	8,715	15,687

Same (face of left heading 6, off main heading, 4-foot 10¾-inch bed, 3-foot 9¾-inch cut).	294	12113.	A	1	3.40	19.44	68.75	8.41	2.02	......	......	......	......	2.4	7,673	13,811
				2	......	20.12	71.17	8.71	2.09	......	......	......	......	......	7,943	14,297
				3	......	22.04	77.96	......	2.29	......	......	......	......	......	8,701	15,662
Same (composite of samples 12109-12113)........	294	12114	A	1	3.27	19.91	69.02	7.80	2.02	4.84	78.95	1.32	5.07	2.5	7,789	14,020
				2	......	20.58	71.36	8.06	2.09	4.63	81.62	1.36	2.24	......	8,052	14,494
				3	......	22.38	77.62	......	2.27	5.04	88.78	1.48	2.43	......	8,758	15,764
Osceola Mills, 3 miles west of; Fairmount No. 2 mine, Lower Freeport or D bed (face of room 5, off left heading 2, 5-foot 1-inch bed and cut).	294	W17010	A	1	3.1	21.0	69.7	6.2	1.55	......	......	......	......	......	7,861	14,150
				2	......	21.5	72.1	6.4	1.60	......	......	......	......	......	8,117	14,610
				3	......	23.0	77.0	......	1.71	......	......	......	......	......	8,672	15,610
Same (face of room 14, off right heading 8, 5-foot 3½-inch bed and cut).	294	W17012	A	1	2.4	22.5	68.8	6.3	1.00	......	......	......	......	......	7,967	14,340
				2	......	23.0	70.6	6.4	1.00	......	......	......	......	......	8,161	14,690
				3	......	24.6	75.4	......	1.07	......	......	......	......	......	8,717	15,690
Same (composite of samples W17010 and W17012)	295	11711	A	1	1.80	......	......	6.44	1.31	4.71	81.71	1.28	4.55	......	......	......
				2	......	......	......	6.56	1.33	4.59	83.21	1.30	3.01	......	......	......
				3	......	......	......	......	1.42	4.91	89.05	1.39	3.23	......	......	......
Philipsburg, 2 miles west of; Guion mine, Lower Kittanning or B bed (face of right heading 1, off right entry 2, 4,000 feet from mine mouth, 3-foot 1½ inch bed and cut).	295	W20369	A	1	2.0	20.0	69.9	8.1	1.05	......	......	......	......	......	7,828	14,090
				2	......	20.4	71.3	8.3	1.10	......	......	......	......	......	7,983	14,370
				3	......	22.2	77.8	......	1.20	......	......	......	......	......	8,711	15,680
Same (face of right heading 2, 4,000 feet from mine mouth, 3-foot 2-inch bed and cut).	295	W20370	A	1	2.0	21.5	69.2	7.3	1.00	......	......	......	......	......	7,906	14,230
				2	......	21.9	70.7	7.4	1.05	......	......	......	......	......	8,067	14,520
				3	......	23.7	76.3	......	1.13	......	......	......	......	......	8,711	15,680
Same (back heading, off right entry 1, 2,500 feet from mine mouth, 3-foot 4-inch bed, 2-foot 8-inch cut).	295	W20371	A	1	2.9	18.4	69.1	9.6	2.15	......	......	......	......	......	7,600	13,680
				2	......	18.9	71.2	9.9	2.20	......	......	......	......	......	7,828	14,090
				3	......	21.0	79.0	......	2.44	......	......	......	......	......	8,683	15,630
Same (50 feet from face of left heading 1, off right entry 2, 3-foot 5½-inch bed and cut).	295	W20372	A	1	2.0	19.2	70.1	8.7	1.53	......	......	......	......	......	7,789	14,020
				2	......	19.6	71.5	8.9	1.60	......	......	......	......	......	7,950	14,310
				3	......	21.5	78.5	......	1.76	......	......	......	......	......	8,728	15,710
Same (face of left heading 1, off right entry 1, 2,200 feet from mine mouth, 3-foot 6½-inch bed, 3-foot 2¾-inch cut).	295	W20373	A	1	2.4	19.6	67.5	10.5	1.85	......	......	......	......	......	7,539	13,570
				2	......	20.1	69.1	10.8	1.90	......	......	......	......	......	7,728	13,910
				3	......	22.5	77.5	......	2.13	......	......	......	......	......	8,667	15,600
Same (composite of samples W20369-W20373)....	295	12367	A	1	.90	21.59	68.49	9.02	1.99	4.57	79.49	1.31	3.62	......	7,811	14,060
				2	......	21.79	69.11	9.10	2.01	4.51	80.21	1.32	2.85	......	7,882	14,188
				3	......	23.97	76.03	......	2.21	4.96	88.24	1.45	3.14	......	8,671	15,608
Smoke Run, 3½ miles southeast of; Viola mine, Miller or B bed (near face of left heading 9, beyond room 18, 1,300 feet from main haulage way, 5-foot 8-inch bed, 4-foot 10-inch cut).	296	11748	A	1	3.83	20.40	68.87	6.90	1.11	......	......	......	......	3.1	7,812	14,062
				2	......	21.21	71.62	7.17	1.15	......	......	......	......	......	8,123	14,621
				3	......	22.85	77.15	......	1.24	......	......	......	......	......	8,750	15,750
Same (face of second slant to right just beyond room 3, 5-foot ¼-inch bed, 4-foot ¾-inch cut).	296	11749	A	1	3.67	20.36	68.87	7.10	.90	......	......	......	......	2.9	7,822	14,080
				2	......	21.14	71.49	7.37	.93	......	......	......	......	......	8,120	14,616
				3	......	22.82	77.18	......	1.00	......	......	......	......	......	8,766	15,779
Same (face of main entry, 4,750 feet southwest of opening, 5-foot 4½-inch bed, 4-foot 4½-inch cut).	296	11750	A	1	3.67	20.30	67.64	8.39	1.18	......	......	......	......	2.9	7,706	13,871
				2	......	21.07	70.22	8.71	1.22	......	......	......	......	......	8,000	14,400
				3	......	23.08	76.92	......	1.34	......	......	......	......	......	8,763	15,773
Same (composite of samples 11748, 11749, and 11750).	296	11751	A	1	3.73	20.29	68.41	7.57	1.33	4.86	78.92	1.22	6.10	3.0	7,761	13,970
				2	......	21.08	71.06	7.86	1.38	4.62	81.97	1.27	2.90	......	8,061	14,510
				3	......	22.88	77.12	......	1.50	5.01	88.96	1.38	3.15	......	8,749	15,748
Same (near face of left entry 10; 100 feet from room 12; 5-foot 4½-inch bed, 4-foot 5½-inch cut).	296	W18355	A	1	3.0	19.6	70.3	7.1	.75	......	......	......	......	......	7,822	14,080
				2	......	20.2	72.5	7.3	.75	......	......	......	......	......	8,061	14,510
				3	......	21.8	78.2	......	.81	......	......	......	......	......	8,694	15,650

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
CLEARFIELD COUNTY—continued.																
Smoke Run, 3½ miles southeast of; Viola Mine, Miller or B bed (near face of third slant entry, 100 feet above room 1, 5-foot bed, 4 foot 11-inch cut).	296	W18356	A	1	3.0	19.2	69.7	8.1	1.60	......	......	......	......	......	7,767	13,980
				2	......	19.8	71.8	8.4	1.65	......	......	......	......	......	8,006	14,410
				3	......	21.6	78.4	......	1.80	......	......	......	......	......	8,739	15,730
Same (near face of left heading 9, 5-foot 7⅜-inch bed, 4-foot 7⅜-inch cut).	296	W18357	A	1	3.4	19.5	70.4	6.7	1.00	......	......	......	......	......	7,867	14,160
				2	......	20.2	72.9	6.9	1.05	......	......	......	......	......	8,144	14,660
				3	......	21.7	78.3	......	1.13	......	......	......	......	......	8,750	15,750
Same (face of main entry, 4,800 feet southwest of opening, 5-foot 1⅛-inch bed, 4-foot 10⅝-inch cut).	296	W18358	A	1	3.3	19.0	69.7	8.0	1.60	......	......	......	......	......	7,711	13,880
				2	......	19.7	72.0	8.3	1.65	......	......	......	......	......	7,978	14,360
				3	......	21.5	78.5	......	1.80	......	......	......	......	......	8,700	15,660
Same (face of second slant entry, 100 feet above third slant entry, 5-foot bed, 4-foot 1-inch cut).	296	W18359	A	1	3.3	19.0	70.2	7.5	1.00	......	......	......	......	......	7,833	14,100
				2	......	19.6	72.6	7.8	1.05	......	......	......	......	......	8,094	14,570
				3	......	21.3	78.7	......	1.14	......	......	......	......	......	8,778	15,800
HUNTINGDON COUNTY.																
Robertsdale, Robertsdale mine; Fulton (Lower Kittanning or B) bed (face of north heading 6, 6-foot 3½-inch bed, 4-foot 5⅜-inch cut).	297	12115	A	1	2.26	14.81	76.54	6.39	1.36	......	......	......	......	1.6	8,038	14,468
				2	......	15.15	78.31	6.54	1.39	......	......	......	......	......	8,224	14,803
				3	......	16.21	83.79	......	1.49	......	......	......	......	......	8,800	15,840
Same (at face of main west heading, 5-foot 4-inch bed, 3-foot 9-inch cut).	297	12116	A	1	1.92	15.23	77.25	5.60	.80	......	......	......	......	1.2	8,130	14,634
				2	......	15.53	78.76	5.71	.82	......	......	......	......	......	8,289	14,920
				3	......	16.47	83.53	......	.87	......	......	......	......	......	8,791	15,824
Same (at face of south heading 6, 6-foot 1¼-inch bed, 4-foot 9¼-inch cut).	297	12117	A	1	1.95	14.58	76.24	7.23	.94	......	......	......	......	1.3	7,983	14,369
				2	......	14.87	77.76	7.37	.96	......	......	......	......	......	8,142	14,656
				3	......	16.05	83.95	......	1.04	......	......	......	......	......	8,790	15,822
Same (face of south heading 8, 5-foot 3-inch bed, 4-foot 3-inch cut).	297	12118	A	1	2.20	13.93	77.44	6.43	1.08	......	......	......	......	1.5	8,046	14,483
				2	......	14.24	79.19	6.57	1.10	......	......	......	......	......	8,227	14,809
				3	......	15.24	84.76	......	1.18	......	......	......	......	......	8,805	15,849
Same (face of south heading 5, 3-foot 11⅝-inch bed, 3-foot 4⅝-inch cut).	297	12119	A	1	2.34	15.64	75.55	6.47	1.02	......	......	......	......	1.7	8,008	14,414
				2	......	16.02	77.35	6.63	1.04	......	......	......	......	......	8,200	14,760
				3	......	17.16	82.84	......	1.11	......	......	......	......	......	8,782	15,808
Same (composite of samples 12115–12119).........	298	12120	A	1	2.14	15.47	75.96	6.43	1.05	4.44	82.83	1.27	3.98	1.5	8,039	14,470
				2	......	15.81	77.62	6.57	1.07	4.29	84.64	1.30	2.13	......	8,215	14,787
				3	......	16.92	83.08	......	1.15	4.59	90.59	1.39	2.28	......	8,793	15,827
Woodvale; Woodvale mine, Fulton (Lower Kittanning or B) bed (face of south heading 1, 4-foot 5⅜-inch bed, 3-foot 9⅜-inch cut).	298	12121	A	1	2.92	14.28	76.74	6.06	.94	......	......	......	......	2.2	7,997	14,395
				2	......	14.71	79.05	6.24	.97	......	......	......	......	......	8,238	14,828
				3	......	15.69	84.31	......	1.03	......	......	......	......	......	8,787	15,817

Same (face of third south or up-creek heading, 4-foot 6¾-inch bed, 3 foot 7½-inch cut).	298	12122	A	1	1.98	13.38	79.26	5.38	.86					1.3	8,159	14,686
				2		13.65	80.86	5.49	.88						8,324	14,982
				3		14.44	85.56		.92						8,808	15,854
Same (face of second west tunnel 4-foot 9¾-inch bed, 3-foot 11¾-inch cut).	298	12123	A	1	1.85	14.14	78.09	5.92	.92					1.3	8,116	14,609
				2		14.41	79.56	6.03	1.00						8.269	14,884
				3		15.34	84.66		1.06						8,800	15,840
Same (No. 7 or Bolinger heading, 5-foot 10¾-inch bed, 5-foot 1¾-inch cut).	298	12124	A	1	2.43	15.31	75.48	6.78	1.79					1.7	7,939	14,290
				2		15.69	77.36	6.95	1.83						8,137	14,647
				3		16.86	83.14		1.97						8,745	15,741
Same (composite of samples 12121–12124).........	298	12125	A	1	2.18	14.07	77.62	6.13	1.14	4.46	83.33	1.30	3.64	1.6	8,085	14,553
				2		14.38	79.35	6.27	1.17	4.31	85.19	1.33	1.73		8,265	14,877
				3		15.34	84.66		1.25	4.60	90.89	1.42	1,84		8.18	15,872
INDIANA COUNTY.																
Clymer, 1½ miles north of: Empire M (McKean) mine; Lower Kittanning or B bed; corner of pillar of room 44, off right entry 1, off main entry, 3-foot 3¼-inch cut).	299	W19682	A	1	2.2	26.4	64.6	6.8	1.95						7,956	14,320
				2		27.0	66.0	7.0	2.00						8,133	14,640
				3		29.0	71.0		2.15						8,744	15,740
Same (face of room 56, off right entry 2, off main entry, 4,000 feet east of mine mouth, 3-foot 2¾-inch bed and cut).	299	W19678	A	1	1.9	26.5	64.0	7.6	2.40						7,911	14,240
				2		27.0	65.2	7.8	2.50						8,061	14,510
				3		29.3	70.7		2.71						8,744	15,740
Same (corner of crosscut at room 40, off right entry 2, off main entry, 3,000 feet southeast of mine mouth, 3-foot 2½-inch cut).	299	W19680	A	1	2.1	26.5	63.4	8.0	2.25						7,811	14,060
				2		27.1	64.7	8.2	2.30						7,978	14,360
				3		29.5	70.5		2.51						8,694	15,650
Same (face of first parallel, off south heading, off right entry 1, off main entry, 2,800 feet southwest of mine mouth, 3-foot 5½-inch cut).	299	W19685	A	1	2.8	26.4	64.4	6.4	1.65						7,917	14,250
				2		27.2	66.2	6.6	1.70						8,150	14,670
				3		29.1	70.9		1.82						8,728	15,710
Same (face of third parallel, off south heading, off right entry 1, off main entry, 3-foot 3¾-inch cut).	299	W19687	A	1	2.5	26.9	63.9	6.7	1.70						7,917	14,250
				2		27.6	65.5	6.9	1.75						8,122	14,620
				3		29.6	70.4		1.88						8,722	15,700
Same (composite of samples W19678, W19680, W19685, and W19687).	299	12223	A	1	1.30	26.70	64.40	7.60	2.03	5.02	80.35	1.36	3.64		7,953	14,315
				2		27.05	65.25	7.70	2.06	4.94	81.41	1.38	2.51		8,058	14,504
				3		29.31	70.69		2.23	5.35	88.20	1.50	2.72		8,730	15,714
1 mile north of: Empire R mine, Lower Kittanning or B bed (pillar at room 28, in left entry 2, off main entry, 2,200 feet east of mine mouth, 3-foot 2-inch bed and cut).	300	W19681	A	1	2.1	26.5	64.9	6.5	1.70						7,972	14,350
				2		27.1	66.3	6.6	1.75						8,139	14,650
				3		29.0	71.0		1.87						8,717	15,690
Same (crosscut at corner of room 47, off left entry 1, off main entry, 4,700 feet southeast of mine mouth, 3-foot 2½-inch bed, 3-foot 1¾-inch cut).	300	W19679	A	1	2.3	26.6	63.3	7.8	2.70						7,839	14,110
				2		27.2	64.8	8.0	2.75						8,022	14,440
				3		29.6	70.4		2.99						8,722	15,700
Same (face of room 3, off right entry 1, off main entry, 1,100 feet south of mine mouth, 3-foot 10¾-inch bed and cut).	300	W19683	A	1	2.0	26.6	64.4	7.0	1.90						7,961	14,330
				2		27.1	65.8	7.1	1.95						8,122	14,620
				3		29.2	70.8		2.10						8,750	5,750
Same (face of room 11, left entry 5, off main entry, 1,900 feet southeast of mine mouth, 3-foot 2¾-inch bed and cut).	300	W19684	A	1	2.2	26.0	64.5	7.3	2.15						7,850	14,130
				2		26.6	65.9	7.5	2.20						8,022	14,440
				3		28.8	71.2		2.38						8,672	15,610
Same (near face of room 43, off left entry 4, off main entry, 3,100 feet from mine mouth, 3-foot 2½-inch bed and cut).	300	W19686	A	1	1.9	26.6	64.7	6.8	2.15						7,972	14,350
				2		27.1	66.0	6.9	2.20						8,128	14,630
				3		29.1	70.9		2.36						8,728	15,710
Same (composite of samples W19679, W19683, W19684, and W19686).	300	12219	A	1	1.23	26.37	64.84	7.56	2.41	4.99	79.89	1.37	3.78		7,943	14,297
				2		26.70	65.65	7.65	2.44	4.91	80.89	1.39	2.72		8,042	14,476
				3		28.91	71.09		2.64	5.32	87.59	1.51	2.94		8,708	15,674

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
INDIANA COUNTY—continued.																
Glen Campbell, 2½ miles north of Huvershurst Junction; Indiana No. 6 mine, Upper Freeport or E bed (face of left heading 7, off main entry, 3,000 feet northwest of mine mouth, 4-foot 3¾-inch bed, 3-foot 8¼-inch cut).	301	12102	A	1	3.01	27.42	61.30	8.27	1.30					1.8	7,650	13,770
				2		28.27	63.20	8.53	1.34						7,887	14,197
				3		30.91	69.09		1.47						8,623	15,521
Same (face of room 10 off left entry 1, off main heading, 1,000 feet northeast of mine mouth, 4-foot 10¼-inch bed, 3-foot 10¼-inch cut).	301	12103	A	1	2.89	26.89	61.69	8.53	1.20					1.7	7,622	13,720
				2		27.69	63.53	8.78	1.23						7,849	14,128
				3		30.36	69.64		1.35						8,606	15,491
Same (rib near face of room 14, in right heading 7, off main entry, 2,000 feet northwest of mine mouth, 4-foot 11½-inch bed, 4-foot 1½-inch cut).	301	12104	A	1	3.66	26.95	61.24	8.15	1.19					2.5	7,611	13,700
				2		27.97	63.57	8.46	1.24						7,900	14,220
				3		30.55	69.45		1.35						8,630	15,534
Same (face of left heading 6, off main entry, 2,200 feet southwest of mine mouth, 4-foot 8¾-inch bed, 3-foot 11-inch cut).	301	12105	A	1	2.88	27.93	60.25	8.94	1.70					1.7	7,610	13,698
				2		28.76	62.03	9.21	1.75						7,836	14,105
				3		31.68	68.32		1.93						8,631	15,536
Same (near face of room 1, on right heading 8, off main entry, 2,300 feet northwest of mine mouth, 4-foot 10½-inch bed, 4-foot ⅞-inch cut).	301	12106	A	1	3.23	27.44	61.05	8.28	1.26					1.9	7,609	13,696
				2		28.36	63.08	8.56	1.30						7,863	14,153
				3		31.01	68.99		1.42						8,599	15,478
Same (composite of samples 12102–12106)	301	12107	A	1	3.08	27.32	61.16	8.44	1.29	4.99	76.71	1.27	7.30	1.9	7,651	13,772
				2		28.19	63.10	8.71	1.33	4.80	79.15	1.31	4.70		7,894	14,209
				3		30.88	69.12		1.46	5.26	86.70	1.43	5.15		8,647	15,565
Homer City, 1 mile northwest of; Tearing Run mine, E or Upper Freeport bed (No. 4 stump, o n right entry 1, off main entry, 5-foot 2¼-inch cut).	301	W20431	A	1	2.6	24.5	64.8	8.1	2.50						7,739	13,930
				2		25.2	66.5	8.3	2.55						7,944	14,300
				3		27.5	72.5		2.78						8,756	15,760
Same (rib of room 10, off left entry 1, off main entry, 3-foot 7½-inch cut).	301	W20428	A	1	2.6	26.5	61.7	9.2	2.80						7,622	13,720
				2		27.2	63.4	9.4	2.90						7,828	14,090
				3		30.0	70.0		3.20						8,639	15,550
Same (face of right entry 2, off main entry, 5-foot 1-inch cut).	301	W20429	A	1	2.8	23.8	64.1	9.3	2.65						7,550	13,590
				2		24.5	65.9	9.6	2.75						7,767	13,980
				3		27.1	72.9		3.04						8,589	15,460

Same (rib near face of left entry 2, off main entry, 5-foot 1¼-inch cut).	301	W20430	A	1	2.3	24.6	64.6	8.5	2.20	......	......	......	......	......	7,728	13,910
				2	......	25.2	66.1	8.7	2.25	......	......	......	......	......	7,911	14,240
				3	......	27.6	72.4	......	2.46	......	......	......	......	......	8,667	15,600
Same ("shoo-fly," 100 feet below left entry 3, off main entry, 5-foot 3⅜-inch cut).	301	W20432	A	1	2.0	24.4	65.2	8.4	2.15	......	......	......	......	......	7,711	13,880
				2	......	24.9	66.5	8.6	2.20	......	......	......	......	......	7,872	14,170
				3	......	27.2	72.8	......	2.41	......	......	......	......	......	8,611	15,500
Same (composite of samples 20428, 20429, 20430, and 20432).	301	12373	A	1	1.00	26.07	63.79	9.14	2.69	4.67	78.12	1.12	4.26	......	7,747	13,945
				2	......	26.33	64.44	9.23	2.72	4.61	78.91	1.13	3.40	......	7,825	14,085
				3	......	29.01	70.99	......	3.00	5.08	86.94	1.24	3.74	......	8,621	15,518
JEFFERSON COUNTY.																
Sykesville, 1 mile south of; Sykesville mine, D or Lower Freeport bed (pillar at last cut-through between eighth left main and the back heading, off main south heading, 5-foot 10½-inch bed, 5-foot 9-inch cut).	302	12455	A	1	2.44	28.44	60.68	8.44	1.32	5.07	76.91	1.31	6.95	1.6	7,629	13,732
				2	......	29.15	62.20	8.65	1.35	4.92	78.83	1.34	4.91	......	7,820	14,076
				3	......	31.91	68.09	......	1.48	5.39	86.30	1.47	5.36	......	8,561	15,410
Same (from cavity in roof between third right butt heading and second back heading in eighth main off main south heading; 6-foot ½-inch bed, 3-foot 7-inch cut).	303	12457	A	1	2.05	28.44	53.51	16.00	4.01	4.65	68.85	1.16	5.33	1.3	6,890	12,402
				2	......	29.03	54.64	16.33	4.09	4.51	70.29	1.18	3.60	......	7,034	12,661
				3	......	34.70	65.30	......	4.89	5.39	84.01	1.41	4.30	......	8,407	15,133
Same (face of third right butt heading off eighth left, off main south heading, 5-foot 8½-inch bed, 5-foot 7-inch cut).	302	12458	A	1	2.53	29.50	60.48	7.49	1.31	5.07	78.06	1.38	6.69	1.5	7,684	13,831
				2	......	30.27	62.05	7.68	1.34	4.91	80.09	1.42	4.56	......	7,884	14,191
				3	......	32.79	67.21	......	1.45	5.32	86.75	1.54	4.94	......	8,540	15,372
Same (cavity in roof of third right butt heading off eighth left main, about 200 feet from face of main heading, 2-foot 7½-inch cut).	303	12471	A	1	1.94	29.01	51.36	17.69	2.28	4.52	67.80	1.15	6.56	1.2	6,721	12,098
				2	......	29.58	52.38	18.04	2.33	4.39	69.14	1.17	4.93	......	6,854	12,337
				3	......	36.09	63.91	......	2.84	5.36	84.36	1.43	6.01	......	8,363	15,053
LACKAWANNA COUNTY.																
Dunmore, 3 miles east of Scranton; Sloan mine, Dunmore No. 2 bed (7-foot cut).	303	11440	B	1	3.43	6.79	78.25	11.53	.46	2.52	78.85	.77	5.87	2.5	7,101	12,782
				2	......	7.03	81.03	11.94	.48	2.22	81.65	.80	2.91	......	7,353	13,235
				3	......	7.98	92.02	......	.55	2.52	92.72	.91	3.30	......	8,350	15,030
LUZERNE COUNTY.																
Pittston, 2 miles southwest of; Colliery No. 14 mine, Pittston bed (working face of rear station 4695, on Brannagan slope).	303	11441	B	1	2.19	5.67	86.24	5.90	.57	2.70	86.37	.91	3.55	1.6	7,682	13,828
				2	......	5.80	88.17	6.03	.58	2.52	88.30	.93	1.64	......	7,854	14,137
				3	......	6.17	93.83	......	.62	2.68	93.97	.99	1.74	......	8,358	15,044
Wilkes-Barre, near; Dorrance mine, Red Ash bed (end of first west gangway, 600 feet northwest of shaft).	303	11782	B	1	1.31	5.68	85.87	7.14	.42	2.35	86.76	.68	2.65	.3	7,654	13,777
				2	......	5.76	87.01	7.23	.43	2.23	87.91	.69	1.51	......	7,756	13,961
				3	......	6.21	93.79	......	.46	2.40	94.76	.74	1.64	......	8,360	15,048
SOMERSET COUNTY.																
Acosta, ¼ mile north of; Belmont No. 1 mine, Upper Kittanning or C′ bed (face of room 4 off main heading, 1,300 feet from mine mouth, 3-foot 6-inch bed, 3-foot 4½-inch cut).	304	W19536	A	1	2.0	16.1	72.8	9.1	1.35	......	......	......	......	......	7,778	14,000
				2	......	16.4	74.3	9.3	1.40	......	......	......	......	......	7,939	14,290
				3	......	18.1	81.9	......	1.54	......	......	......	......	......	8,756	15,760
Same (face of third left entry, 60 feet from main entry, 3-foot 5¾-inch bed, 3-foot 4¾-inch cut).	304	W19537	A	1	2.2	15.7	73.4	8.7	1.25	......	......	......	......	......	7,811	14,060
				2	......	16.1	75.0	8.9	1.30	......	......	......	......	......	7,983	14,370
				3	......	17.7	82.3	......	1.43	......	......	......	......	......	8,756	15,760
Same (face of room 3, off first right entry, 3-foot 9½-inch bed and cut).	304	W19538	A	1	2.0	15.5	72.7	9.8	1.60	......	......	......	......	......	7,683	13,830
				2	......	15.8	74.2	10.0	1.65	......	......	......	......	......	7,844	14,120
				3	......	17.6	82.4	......	1.83	......	......	......	......	......	8,711	15,680

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
SOMERSET COUNTY—continued.																
Acosta, ¼ mile north of; Belmont No. 1 mine, Upper Kittanning or C′ bed (face of left entry 2; 250 feet from main entry; 3-foot 3½-inch bed and cut).	304	W19539	A	1	2.0	16.2	73.1	8.7	1.30	......	......	......	......	......	7,828	14,090
				2	......	16.5	74.6	8.9	1.35	......	......	......	......	......	7,989	14,380
				3	......	18.1	81.9	......	1.48	......	......	......	......	......	8,767	15,780
Same (face of main heading; 1,550 feet from mine mouth, 150 feet above left entry 3; 3-foot 2½-inch bed, 3-foot 1-inch cut).	304	W19543	A	1	1.4	16.4	73.7	8.5	1.05	......	......	......	......	......	7,872	14,170
				2	......	16.6	74.8	8.6	1.10	......	......	......	......	......	7,983	14,370
				3	......	18.2	81.8	......	1.20	......	......	......	......	......	8,733	15,720
Same (composite of samples W19536, W19537, W19538, W19539, and W19543).	304	12209	A	1	1.2	15.65	74.18	8.97	1.39	4.31	80.74	1.36	3.23	......	7,831	14,096
				2	......	15.84	75.08	9.08	1.41	4.23	81.72	1.38	2.18	......	7,926	14,267
				3	......	17.42	82.58	......	1.55	4.65	89.88	1.52	2.40	......	8,718	15,692
Berlin, 1½ miles east of; Coronet No. 3 mine, "Smokeless" or Upper Freeport bed (pillar between rooms 2 and 3 off main air course, between right entries 3 and 4; 1,400 feet from mine mouth).	305	W19535	A	1	3.0	17.7	68.6	10.7	1.50	......	......	......	......	......	7,467	13,440
				2	......	18.2	70.8	11.0	1.55	......	......	......	......	......	7,700	13,860
				3	......	20.4	79.6	......	1.74	......	......	......	......	......	8,656	15,580
Same (face of left entry 8, 15 feet beyond room 15; 2,950 feet from mouth of main entry, 3-foot 9½-inch bed and cut).	305	W19532	A	1	2.3	18.5	66.4	12.8	1.80	......	......	......	......	......	7,406	13,330
				2	......	18.9	68.0	13.1	1.85	......	......	......	......	......	7,583	13,650
				3	......	21.7	78.3	......	2.13	......	......	......	......	......	8,728	15,710
Same (face of right entry 6; 125 feet from main heading, 2,500 feet from mouth of main entry; 3-foot 5½-inch bed, 3-foot 4¾-inch cut).	305	W19533	A	1	2.3	18.4	68.4	10.9	1.35	......	......	......	......	......	7,561	13,610
				2	......	18.8	70.0	11.2	1.40	......	......	......	......	......	7,739	13,930
				3	......	21.2	78.8	......	1.58	......	......	......	......	......	8,711	15,680
Same (face of room 13, off left entry 7, 2,450 feet from mouth of main entry, 3-foot 6½-inch bed, 3-foot 5-inch cut).	305	W19534	A	1	2.4	18.0	68.3	11.3	1.90	......	......	......	......	......	7,522	13,540
				2	......	18.4	70.0	11.6	1.95	......	......	......	......	......	7,706	13,870
				3	......	20.8	79.2	......	2.21	......	......	......	......	......	8,717	15,690
Same (face of right entry 7, 50 feet beyond room 8; 2,800 feet from mouth of main entry, 3-foot 8-inch bed, 3-foot 4½-inch cut).	305	W19542	A	1	2.4	19.0	67.6	11.0	1.90	......	......	......	......	......	7,600	13,680
				2	......	19.5	69.2	11.3	1.95	......	......	......	......	......	7,789	14,020
				3	......	22.0	78.0	......	2.20	......	......	......	......	......	8,783	15,810
Same (composite of samples W19542, W19533, W19532, and W19534).	305	12214	A	1	1.44	18.56	68.44	11.56	1.88	4.39	77.31	1.64	3.22	......	7,558	13,604
				2	......	18.83	69.44	11.73	1.91	4.29	78.44	1.66	1.97	......	7,668	13,802
				3	......	21.33	78.67	......	2.16	4.86	88.86	1.88	2.24	......	8,687	15,637
Confluence, 4¼ miles southeast of; Linmer mine, C′ bed (at end of back subentry, off main entry 1, about 2,500 feet southeast of mine mouth, 3-foot 6½-inch bed and cut).	306	13631	B	1	2.8	25.94	63.06	8.20	2.35	......	......	......	......	2.2	7,715	13,887
				2	......	26.69	64.87	8.44	2.42	......	......	......	......	......	7,937	14,287
				3	......	29.15	70.85	......	2.64	......	......	......	......	......	8,669	15,604

Elklick, 2½ miles east of; Eagle mine, D (?) bed (room in mine, about 600 feet east of mine mouth, 3-foot 1½-inch bed, 2-foot 10½-inch cut).	306	13983	B	1	3.49	18.21	69.67	8.63	1.53	....	....	....	....	2.6	7,593	13,667
				2	....	18.87	72.19	8.94	1.59	....	....	....	....	....	7,868	14,162
				3	....	20.72	79.28	....	1.75	....	....	....	....	....	8,641	15,554
2½ miles northwest of; Grassy Run No. 1 mine, 4-foot bed (about 1,200 feet southwest of mine mouth, 4-foot 7-inch bed, 4-foot 6-inch cut).	306	13664	B	1	2.73	19.63	64.99	12.65	2.30	....	....	....	....	2.2	7,357	13,243
				2	....	20.18	66.81	13.01	2.36	....	....	....	....	....	7,564	13,615
				3	....	23.20	76.80	....	2.71	....	....	....	....	....	8,696	15,653
Holsopple, ½ mile north of; Lenore mine, B or Miller bed (rib of right entry 1, 200 feet from main entry, 3-foot 4½-inch bed, 3-foot 3½-inch cut).	307	15078	A	1	2.75	15.98	74.03	7.24	2.15	....	....	....	....	2.0	7,762	13,972
				2	....	16.43	76.13	7.44	2.21	....	....	....	....	....	7,982	14,368
				3	....	17.75	82.25	....	2.39	....	....	....	....	....	8,624	15,523
Same (face of south entry 3, 3-foot 7-inch bed and cut).	307	15079	A	1	2.32	16.73	72.98	7.97	2.64	....	....	....	....	1.9	7,796	14,033
				2	....	17.13	74.71	8.16	2.70	....	....	....	....	....	7,982	14,368
				3	....	18.65	81.35	....	2.94	....	....	....	....	....	8,691	15,644
Same (30 feet from face of main entry, 3-foot 3¾-inch bed and cut).	307	15080	A	1	1.94	17.07	72.41	8.58	2.61	....	....	....	....	1.5	7,761	13,970
				2	....	17.41	73.84	8.75	2.66	....	....	....	....	....	7,915	14,247
				3	....	19.08	80.92	....	2.92	....	....	....	....	....	8,674	15,613
Same (30 feet from face of south entry 7, 3-foot 1/16-inch bed and cut).	307	15081	A	1	1.81	17.36	73.16	7.67	2.23	....	....	....	....	1.4	7,900	14,220
				2	....	17.68	74.51	7.81	2.27	....	....	....	....	....	8,045	14,481
				3	....	19.18	80.82	....	2.46	....	....	....	....	....	8,726	15,707
Same (composite of samples 15079, 15080, and 15081).	307	15082	A	1	1.99	16.58	73.28	8.15	2.46	4.37	80.24	1.28	3.50	1.6	7,839	14,110
				2	....	16.92	74.76	8.32	2.51	4.23	81.87	1.31	1.76	....	7,998	14,396
				3	....	18.46	81.54	....	2.74	4.61	89.30	1.43	1.92	....	8,724	15,703
¾ mile north of; Oneida mine, C′ bed (on pillar in room 6, off left entry 5, 3-foot 4-inch bed and cut).	308	15073	A	1	2.23	15.62	72.83	9.32	1.87	....	....	....	....	1.7	7,645	13,761
				2	....	15.98	74.49	9.53	1.91	....	....	....	....	....	7,819	14,074
				3	....	17.66	82.34	....	2.11	....	....	....	....	....	8,642	15,556
Same (face of room 3, off left entry 6, 3-foot 5¾-inch bed and cut).	308	15074	A	1	1.99	17.73	69.54	10.74	2.90	....	....	....	....	1.5	7,539	13,570
				2	....	18.09	70.95	10.96	2.96	....	....	....	....	....	7,692	13,846
				3	....	20.32	79.68	....	3.32	....	....	....	....	....	8,639	15,550
Same (15 feet from face of left entry 5, 3-foot 7¾-inch bed and cut).	308	15075	A	1	2.27	16.71	69.63	11.39	3.47	....	....	....	....	1.8	7,414	13,345
				2	....	17.10	71.25	11.65	3.55	....	....	....	....	....	7,586	13,655
				3	....	19.36	80.64	....	4.02	....	....	....	....	....	8,587	15,457
Same (face of room 23, off left entry 3, 4-foot bed and cut).	308	15076	A	1	1.85	17.34	68.94	11.87	2.77	....	....	....	....	1.4	7,430	13,374
				2	....	17.67	70.24	12.09	2.82	....	....	....	....	....	7,570	13,626
				3	....	20.10	79.90	....	3.21	....	....	....	....	....	8,611	15,500
Same (composite of samples 15074, 15075, and 15076).	308	15077	A	1	2.03	17.28	69.31	11.38	3.00	4.19	76.77	1.25	3.41	1.6	7,482	13,468
				2	....	17.64	70.74	11.62	3.06	4.04	78.36	1.28	1.64	....	7,637	13,747
				3	....	19.96	80.04	....	3.46	4.57	88.66	1.45	1.86	....	8,641	15,554
Jerome, near; Jerome No. 2 mine, C′ or Upper Kittanning bed (south entry 7, off east entry 1½, 6-foot 6-inch bed, 5-foot 3½-inch cut).	308	W19638	A	1	2.0	14.7	73.8	9.5	.90	....	....	....	....	....	7,650	13,770
				2	....	15.0	75.3	9.7	.90	....	....	....	....	....	7,806	14,050
				3	....	16.6	83.4	....	1.00	....	....	....	....	....	8,644	15,560
Same (face of south entry 4, east entry 2, 6-foot ½-inch bed, 5-foot 5-inch cut).	308	W19639	A	1	2.1	14.5	74.1	9.3	.70	....	....	....	....	....	7,722	13,900
				2	....	14.8	75.7	9.5	.70	....	....	....	....	....	7,889	14,200
				3	....	16.4	83.6	....	.77	....	....	....	....	....	8,722	15,700
Same (face of east entry 2, 6-foot ½-inch bed, 4-foot 9½-inch cut).	308	W19642	A	1	2.7	14.3	72.2	10.8	.90	....	....	....	....	....	7,506	13,510
				2	....	14.7	74.2	11.1	.95	....	....	....	....	....	7,717	13,890
				3	....	16.5	83.5	....	1.07	....	....	....	....	....	8,683	15,630
Same (composite of samples W19638, W19639, and W19642).	309	12224	A	1	1.44	15.21	73.38	9.97	.90	4.17	79.43	1.34	4.19	....	7,666	13,799
				2	....	15.43	74.45	10.12	.91	4.07	80.59	1.36	2.95	....	7,778	14,000
				3	....	17.17	82.83	....	1.01	4.53	89.66	1.51	3.29	....	8,654	15,577
Same (face of east entry 1).......................	309	12210	A	1	1.27	14.23	70.06	14.44	1.92	3.98	75.15	1.25	3.26	....	7,274	13,093
				2	....	14.41	70.96	14.63	1.94	3.89	76.12	1.27	2.15	....	7,368	13,262
				3	....	16.88	83.12	....	2.27	4.56	89.17	1.49	2.51	....	8,631	15,536

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
SOMERSET COUNTY—continued.																
Pinehill, 2 miles southwest of Berlin; Consolidation No. 112 mine, Little Pittsburgh bed (room 44, off right subentry 8, off west main entry, about 3,000 feet southwest of mine mouth, 3-foot 7-inch bed and cut).	309	13981	B	1	2.21	21.48	67.29	9.02	2.29	......	......	......	......	1.3	7,658	13,784
				2	......	21.97	68.81	9.22	2.34	......	......	......	......	......	7,831	14,096
				3	......	24.20	75.80	......	2.58	......	......	......	......	......	8,627	15,529
Same (room 8, right subentry 1, off left entry 12, about 5,000 feet southwest of mine mouth, 3-foot 1-inch cut).	309	13982	B	1	3.09	21.58	68.51	6.82	1.04	......	......	......	......	1.9	7,767	13,981
				2	......	22.27	70.69	7.04	1.07	......	......	......	......	......	8,015	14,427
				3	......	23.96	76.04	......	1.15	......	......	......	......	......	8,622	15,520
Same (composite of samples 13981 and 13982).....	309	14037	B	1	2.58	21.53	67.97	7.92	1.66	4.69	78.88	1.72	5.13	1.6	7,724	13,903
				2	......	22.10	69.77	8.13	1.70	4.52	80.97	1.77	2.91	......	7,929	14,272
				3	......	24.06	75.94	......	1.85	4.92	88.14	1.93	3.16	......	8,631	15,536
2 miles southwest of Berlin; Consolidation No. 113 mine, "Stoner" bed (from mine car brought from room 2, off right entry 10, about 3,700 feet southwest of mine mouth.	309	13984	B	1	3.43	19.14	66.61	10.82	1.89	......	......	......	......	2.7	7,398	13,316
				2	......	19.82	68.98	11.20	1.96	......	......	......	......	......	7,661	13,790
				3	......	22.32	77.68	......	2.21	......	......	......	......	......	8,627	15,529
Ralphton, ½ mile east of; Ralphton No. 3 mine, C′ or Upper Kittanning bed (third crosscut off south entry 2, 3,050 feet from mine mouth, 3-foot 11½-inch bed, 3-foot 3-inch cut).	310	W19627	A	1	2.2	15.8	74.2	7.8	.60	......	......	......	......	......	7,850	14,130
				2	......	16.2	75.8	8.0	.60	......	......	......	......	......	8,028	14,450
				3	......	17.6	82.4	......	.65	......	......	......	......	......	8,728	15,710
Same (face of room 11, off straight main entry, 3,150 feet from mine mouth, 3-foot 5½-inch cut).	310	W19628	A	1	2.1	15.2	73.9	8.8	.80	......	......	......	......	......	7,794	14,030
				2	......	15.5	75.5	9.0	.80	......	......	......	......	......	7,967	14,340
				3	......	17.0	83.0	......	.88	......	......	......	......	......	8,756	15,760
Same (face of room 4, off right entry 6, 2,800 feet from mine mouth, 3-foot 4½-inch cut).	310	W19629	A	1	1.9	16.4	73.7	8.0	.70	......	......	......	......	......	7,861	14,150
				2	......	16.7	75.1	8.2	.70	......	......	......	......	......	8,011	14,420
				3	......	18.2	81.8	......	.76	......	......	......	......	......	8,722	15,700
Same (face of room 29, off right heading 4, 2,900 feet from mine mouth, 3-foot 6½-inch cut).	310	W19630	A	1	2.1	15.8	74.2	7.9	.75	......	......	......	......	......	7,850	14,130
				2	......	16.1	75.8	8.1	.75	......	......	......	......	......	8,017	14,430
				3	......	17.5	82.5	......	.82	......	......	......	......	......	8,722	15,700
Same (third crosscut, off south entry 3, 3,700 feet from mine mouth, 3-foot 4½-inch cut).	210	W19631	A	1	2.0	15.5	73.2	9.5	.65	......	......	......	......	......	7,772	13,990
				2	......	15.6	74.7	9.7	.65	......	......	......	......	......	7,933	14,280
				3	......	17.3	82.7	......	.72	......	......	......	......	......	8,783	15,810

Same (composite of samples W19627, W19628, W19629, W19630, and W19631).	310	12228	A	1	1.23	15.49	74.77	8.51	.71	4.35	81.14	1.45	3.84	......	7,872	14,170
				2	......	15.68	75.70	8.62	.72	4.26	82.15	1.47	2.78	......	7,970	14,346
				3	......	17.16	82.84	......	.79	4.66	89.90	1.61	3.04	......	8,722	15,700
1¾ miles north of; Ralphton No. 4 mine, C′ or Upper Kittanning bed (face of south main entry, 3-foot 10¾-inch bed and cut).	311	W19633	A	1	2.4	15.2	74.5	7.9	.65	......	......	......	......	......	7,822	14,080
				2	......	15.6	76.3	8.1	.65	......	......	......	......	......	8,011	14,420
				3	......	17.0	83.0	......	.71	......	......	......	......	......	8,717	15,690
Same (face of room 7, off south main entry, 3-foot 7½-inch bed and cut).	311	W19635	A	1	2.2	15.3	74.7	7.8	.55	......	......	......	......	......	7,894	14,210
				2	......	15.6	76.4	8.0	.55	......	......	......	......	......	8,072	14,530
				3	......	17.0	83.0	......	.60	......	......	......	......	......	8,772	15,790
Same (face of left entry 4, off south main entry, 3-foot 11-inch bed and cut).	311	W19637	A	1	2.1	14.3	75.9	7.7	.60	......	......	......	......	......	7,900	14,220
				2	......	14.6	77.5	7.9	.60	......	......	......	......	......	8,072	14,530
				3	......	15.9	84.1	......	.65	......	......	......	......	......	8,767	15,780
Same (composite of samples W19633, W19635, and W19637).	311	12202	A	1	1.33	14.97	75.75	7.95	.66	4.28	81.59	1.35	4.17	......	7,892	14,206
				2	......	15.17	76.77	8.06	.67	4.19	82.69	1.37	3.02	......	7,999	14,398
				3	......	16.50	83.50	......	.73	4.56	89.94	1.49	3.28	......	8,701	15,662
Same (pillar between south main heading and air course, opposite right entry 1, 3-foot 11½-inch bed and cut).	311	W19632	A	1	1.9	15.7	74.5	7.9	.65	......	......	......	......	......	7,894	14,210
				2	......	16.0	75.9	8.1	.65	......	......	......	......	......	8,044	14,480
				3	......	17.4	82.6	......	.71	......	......	......	......	......	8,756	15,760
Same (on pillar, at second "shoo-fly," off first main entry, 3-foot 7-inch bed and cut).	311	W19636	A	1	2.6	15.2	75.1	7.1	.70	......	......	......	......	......	7,911	14,240
				2	......	15.6	77.1	7.3	.70	......	......	......	......	......	8,128	14,630
				3	......	16.8	83.2	......	.76	......	......	......	......	......	8,767	15,780
Same (composite of samples W19632 and W19636).	311	12218	A	1	1.52	15.78	74.93	7.77	.66	4.36	82.09	1.43	3.69	......	7,936	14,285
				2	......	16.02	76.09	7.89	.67	4.25	83.35	1.45	2.39	......	8,058	14,504
				3	......	17.39	82.61	......	.73	4.61	90.49	1.57	2.60	......	8,749	15,748
WASHINGTON COUNTY.																
Avella, 2 miles west of; Penobscott mine, Pittsburgh bed (room 15, off eighth left entry, off main entry, 2,500 feet northwest of mine mouth; 5-foot 4½-inch bed, 4-foot 11-inch cut).	311	15673	B	1	4.31	37.00	52.85	5.84	1.90	5.31	74.14	1.53	11.28	2.1	7,391	13,304
				2	......	38.67	55.23	6.10	1.99	5.05	77.48	1.60	7.78	......	7,724	13,903
				3	......	41.18	58.82	......	2.12	5.38	82.52	1.70	8.28	......	8,226	14,807
Same (room off right entry 7, 3,100 feet northeast of mine mouth; 4-foot 2-inch bed, 3-foot 10-inch cut).	311	15674	B	1	4.27	37.84	51.55	6.34	2.15	......	......	......	......	1.7	7,398	13,316
				2	......	39.53	53.85	6.62	2.25	......	......	......	......	......	7,728	13,910
				3	......	42.33	57.67	......	2.41	......	......	......	......	......	8,276	14,897
Baird, near; Schoenberger mine, Pittsburgh bed (face of room 10, off entry 32, off face entry 12, 4,000 feet southeast of mine mouth, 5-foot 6¼-inch bed, 5-foot 5-inch cut).	312	11164	A	1	2.29	34.62	57.52	5.57	1.04	......	......	......	......	1.1	......	......
				2	......	35.43	58.87	5.70	1.06	......	......	......	......	......	......	......
				3	......	37.57	62.43	......	1.12	......	......	......	......	......	......	......
Same (face of room 4, off entry 32, off face entry 11, 4,000 feet southeast of mine mouth, 5-foot 9⅝-inch bed, 5-foot 8¾-inch cut).	312	11165	A	1	2.68	34.05	57.40	5.87	1.23	......	......	......	......	1.6	......	......
				2	......	34.99	58.98	6.03	1.26	......	......	......	......	......	......	......
				3	......	37.24	62.76	......	1.34	......	......	......	......	......	......	......
Same (face of right main entry, 7,000 feet south of mine mouth, 5-foot 2¼-inch bed, 5-foot 2⅛-inch cut).	312	11166	A	1	9.48	30.96	53.52	6.04	.97	......	......	......	......	8.4	......	......
				2	......	34.20	59.13	6.67	1.07	......	......	......	......	......	......	......
				3	......	36.65	63.35	......	1.15	......	......	......	......	......	......	......
Same (face of room 11, off entry 32, off right main entry, 6,000 feet south of mine mouth, 5-foot 7½-inch bed, 5-foot 6½-inch cut).	312	11167	A	1	3.54	33.01	57.85	5.60	.98	......	......	......	......	2.3	......	......
				2	......	34.22	59.97	5.81	1.02	......	......	......	......	......	......	......
				3	......	36.33	63.67	......	1.08	......	......	......	......	......	......	......
Same (face of face entry 11, 4,800 feet southeast of mine mouth, 5-foot 6¾-inch bed, 5-foot 5⅜-inch cut).	312	11168	A	1	3.94	33.63	56.63	5.80	.90	......	......	......	......	2.7	......	......
				2	......	35.01	58.95	6.04	.94	......	......	......	......	......	......	......
				3	......	37.26	62.74	......	1.00	......	......	......	......	......	......	......

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
PENNSYLVANIA—Continued.																
WASHINGTON COUNTY—continued.																
Baird, near; Schoenberger mine, Pittsburgh bed (pillar, in right entry 29, off main entry, 5,500 feet south of mine mouth, 5-foot 2-inch bed, 5-foot ¾-inch cut).	312	11169	A	1	2.63	34.32	57.60	5.45	1.09	......	......	......	......	1.4	......	......
				2	......	35.25	59.15	5.60	1.12	......	......	......	......	......	......	......
				3	......	37.34	62.66	......	19.1	......	......	......	......	......	......	......
Same (composite of samples 11164-11691).........	312	11170	A	1	3.94	33.55	56.81	5.70	1.04	5.33	77.21	1.37	9.35	2.9	7,676	13,817
				2	......	34.93	59.14	5.93	1.08	5.09	80.38	1.43	6.09	......	7,991	14,384
				3	......	37.13	62.87	......	1.15	5.41	85.44	1.52	6.48	......	8,494	15,290
Finleyville, 2 miles from; Cincinnati mine, Pittsburgh bed (butt entry 6, off face entry 14, 60 feet outby last cut-through, 6,200 feet northwest of Mingo slope mouth, 5-foot 8½-inch bed and cut).	313	17082	A	1	2.79	35.17	55.30	6.74	1.05	......	......	......	......	1.6	7,564	13,615
				2	......	36.18	56.89	6.93	1.08	......	......	......	......	......	7,781	14,006
				3	......	38.88	61.12	......	1.16	......	......	......	......	......	8,361	15,050
Same (butt entry 10, off face entry 14, 60 feet inby room 2, 6,800 feet northwest of Mingo slope mouth, 6-foot 1½-inch bed, 5-foot 5½-inch cut).	313	17083	A	1	3.22	35.10	55.76	5.92	1.13	......	......	......	......	2.1	7,592	13,666
				2	......	36.27	57.61	6.12	1.17	......	......	......	......	......	7,845	14,121
				3	......	38.63	61.37	......	1.25	......	......	......	......	......	8,356	15,041
Same (butt entry 12, off face entry 14, inby first crosscut, 7,400 feet northwest of Mingo slope mouth, 6-foot 3½-inch bed and cut).	313	17084	A	1	2.87	35.75	54.88	6.50	1.32	......	......	......	......	1.5	7,609	13,696
				2	......	36.80	56.51	6.69	1.36	......	......	......	......	......	7,833	14,099
				3	......	39.44	60.56	......	1.46	......	......	......	......	......	8,395	15,111
Same (composite of samples 17082, 17083, and 17084).	313	17085	A	1	3.03	35.22	55.36	6.39	1.19	5.52	77.05	1.41	8.44	1.8	7,586	13,655
				2	......	36.32	57.09	6.59	1.23	5.34	79.45	1.45	5.94	......	7,823	14,081
				3	......	38.88	61.12	......	1.32	5.72	85.05	1.55	6.36	......	8,375	15,075
Monongahela City; Catsburg mine, Pittsburgh bed (face of butt entry 66, 2½ miles northeast of mine mouth, 5-foot 6-inch bed, 5-foot 5-inch cut).	314	11157	A	1	2.78	33.58	57.15	6.49	1.06	......	......	......	......	1.7	......	......
				2	......	34.54	58.78	6.68	1.09	......	......	......	......	......	......	......
				3	......	37.01	62.99	......	1.17	......	......	......	......	......	......	......
Same (rib of room 17, off entry 75, 2¼ miles northeast of mine mouth, 4-foot 5⅝-inch bed, 4-foot 3½-inch cut).	314	11158	A	1	2.68	33.44	58.07	5.81	.82	......	......	......	......	1.6	......	......
				2	......	34.36	59.67	5.97	.84	......	......	......	......	......	......	......
				3	......	36.54	63.46	......	.89	......	......	......	......	......	......	......
Same (face of room 23, off butt entry 79, 2¾ miles north of mine mouth, 5-foot 2½-inch bed, 5-foot 1½-inch cut).	314	11159	A	1	3.10	33.27	57.79	5.84	1.02	......	......	......	......	1.9	......	......
				2	......	34.33	59.64	6.03	1.05	......	......	......	......	......	......	......
				3	......	36.53	63.47	......	1.12	......	......	......	......	......	......	......

Same (chain pillar at room 46, off butt entry 74, 2½ miles northwest of mine mouth), 5-foot 6⅜-inch bed, 5-foot 5½-inch cut).	314	11160	A	1	2.70	32.78	58.01	6.51	.84	....	....	....	....	1.6	....	....
				2	....	33.69	59.61	6.70	.86	....	....	....	....	....	....	....
				3	....	36.11	63.89	....	.92	....	....	....	....	....	....	....
Same (face of north second main entry, 2½ miles north of mine mouth, 5-foot 6-inch bed, 5-foot 4¾-inch cut).	314	11161	A	1	2.94	33.69	55.73	7.64	2.03	....	....	....	....	1.9	....	....
				2	....	34.71	57.42	7.87	2.09	....	....	....	....	....	....	....
				3	....	37.67	62.33	....	2.27	....	....	....	....	....	....	....
Same (face of room 101, off butt entry 68, 2¾ miles northwest of mine mouth, 5-foot 7⅜-inch cut).	314	11162	A	1	2.93	33.39	57.60	6.08	1.07	....	....	....	....	1.9	....	....
				2	....	34.40	59.34	6.26	1.10	....	....	....	....	....	....	....
				3	....	36.70	63.30	....	1.17	....	....	....	....	....	....	....
Same (composite of samples 11157-11162).........	314	11163	A	1	2.83	33.51	57.29	6.37	1.11	5.38	77.17	1.33	8.64	1.8	7,704	13,867
				2	....	34.49	58.95	6.56	1.14	5.22	79.42	1.37	6.29	....	7,928	14,270
				3	....	36.91	63.09	....	1.22	5.59	85.00	1.47	6.72	....	8,485	15,273
Same (from car of slack, sampled at Pittsburgh experiment station).	314	11172	C	1	2.82	31.58	55.13	10.47	1.21	4.98	73.53	1.32	8.49	1.7	7,314	13,165
				2	....	32.50	56.73	10.77	1.25	4.81	75.66	1.36	6.15	....	7,526	13,547
				3	....	36.42	63.58	....	1.40	5.39	84.79	1.52	6.90	....	8,434	15,181
Same (from car of slack, sampled at Pittsburgh experiment station).	314	11541	C	1	3.82	32.34	52.23	11.61	1.81	5.03	72.16	1.28	8.11	2.6	7,115	12,807
				2	....	33.62	54.31	12.07	1.88	4.79	75.02	1.33	4.91	....	7,397	13,315
				3	....	38.24	61.76	....	2.14	5.45	85.32	1.51	5.58	....	8,413	15,143
Same (from car of slack, sampled at Pittsburgh experiment station).	314	12281	C	1	2.08	31.48	54.66	11.78	1.17	4.91	72.65	1.45	8.04	.6	7,248	13,046
				2	....	32.15	55.82	12.03	1.19	4.78	74.19	1.48	6.33	....	7,402	13,324
				3	....	36.55	63.45	....	1.35	5.43	84.34	1.68	7.20	....	8,415	15,147
WESTMORELAND COUNTY.																
Greensburg, 4 miles north of; No. 4 mine, Pittsburgh or No. 8 bed (from car of lump coal, sampled at Pittsburgh experiment station).	314	12265	C	1	2.14	30.02	58.81	9.03	1.17	5.03	76.33	1.56	6.88	.8	7,590	13,662
				2	....	30.68	60.09	9.23	1.20	4.89	78.00	1.59	5.09	....	7,756	13,961
				3	....	33.80	66.20	....	1.32	5.39	85.93	1.75	5.61	....	8,545	15,381
Same (from car of lump coal, sampled at Pittsburgh experiment station).	314	12853	C	1	2.51	30.28	56.82	10.39	1.37	4.98	74.65	1.51	7.10	1.8	7,425	13,365
				2	....	31.06	58.28	10.66	1.41	4.82	76.57	1.55	4.99	....	7,616	13,709
				3	....	34.77	65.23	....	1.58	5.40	85.70	1.73	5.59	....	8,525	15,345
Lucesco, ¾ mile south of; Lucesco mine, Upper Freeport bed (face of left entry 9; 4,000 feet south of mine mouth, 3-foot 6-inch bed, 3-foot 4½-inch cut).	315	12393	A	1	2.91	33.68	50.09	13.32	4.10	....	....	....	....	1.4	....	....
				2	....	34.69	51.59	13.72	4.22	....	....	....	....	....	....	....
				3	....	40.21	59.79	....	4.89	....	....	....	....	....	....	....
Same (face of left entry 5, 165 feet beyond room 46; 3,600 feet from mine mouth, 3-foot 10-inch bed, 3-foot 8½-inch cut).	315	12394	A	1	2.92	35.46	52.29	9.33	3.17	....	....	....	....	1.4	....	....
				2	....	36.53	53.86	9.61	3.27	....	....	....	....	....	....	....
				3	....	40.41	59.59	....	3.62	....	....	....	....	....	....	....
Same (face of room 48, off right entry 1, 3,200 feet southeast of mine mouth, 3-foot 11-inch bed, 3-foot 8-inch cut).	315	12395	A	1	2.65	36.27	52.09	8.99	2.39	....	....	....	....	1.1	....	....
				2	....	37.26	53.51	9.23	2.46	....	....	....	....	....	....	....
				3	....	41.05	58.95	....	2.71	....	....	....	....	....	....	....
Same (face of room 20, off right entry; 3 2,300 feet southeast of mine mouth, 3-foot 8-inch bed, 3-foot 6-inch cut).	315	12396	A	1	2.57	34.83	52.28	10.32	3.03	....	....	....	....	1.0	....	....
				2	....	35.75	53.66	10.59	3.11	....	....	....	....	....	....	....
				3	....	39.98	60.02	....	3.48	....	....	....	....	....	....	....
Same (composite of samples 12393-12396).........	315	12397	A	1	2.71	35.72	50.87	10.70	3.11	5.00	72.08	1.24	7.87	1.2	7,280	13,104
				2	....	36.72	52.28	11.00	3.20	4.83	74.09	1.27	5.61	....	7,483	13,469
				3	....	41.26	58.74	....	3.60	5.43	83.25	1.43	6.29	....	8,408	15,134
Madison, Keystone mine, Pittsburgh bed (slack).........	315	16270	C	1	3.26	30.68	55.01	11.05	1.49	4.96	72.97	1.41	8.12	2.5	7,231	13,016
				2	....	31.71	56.87	11.42	1.54	4.76	75.43	1.46	5.39	....	7,475	13,455
				3	....	35.80	64.20	....	1.74	5.37	85.15	1.65	6.09	....	8,439	15,190
Same (run of mine).........................	315	16271	C	1	1.94	32.75	55.78	9.53	1.88	5.05	75.40	1.58	6.56	1.2	7,514	13,525
				2	....	33.40	56.88	9.72	1.92	4.93	76.89	1.61	4.93	....	7,663	13,793
				3	....	37.00	63.00	....	2.13	5.46	85.17	1.78	5.46	....	8,488	15,278

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
SOUTH DAKOTA.																
HARDING COUNTY.																
Buffalo, 18 miles east of; sec. 1, T. 17 N., R. 7 E., open pit (face of pit, 6-foot 7-inch bed, 6-foot 1½-inch cut).	316	*13220	B	1	41.49	24.00	24.35	10.16	0.55	......	......	......	......	34.7	3,140	5,652
				2	......	41.02	41.62	17.36	.94	......	......	......	......	......	5,367	9,661
				3	......	49.64	50.36	......	1.14	......	......	......	......	......	6,495	11,691
10 miles north of, sec. 6, T. 20 N., R. 5 E., Hilton mine (face of entry, 120 feet from outcrop at mouth of entry).	316	*13221	B	1	39.80	25.31	23.81	11.08	.96	......	......	......	......	33.0	3,046	5,483
				2	......	42.04	39.56	18.40	1.59	......	......	......	......	......	5,060	9,108
				3	......	51.52	48.48	......	1.95	......	......	......	......	......	6,201	11,162
18 miles east of; sec. 1, T. 17 N., R. 7 E., open pit (face of pit, 11-foot 5-inch bed, 6-foot 3-inch cut).	316	*13222	B	1	41.09	25.84	23.95	9.12	1.14	......	......	......	......	34.3	2,953	5,315
				2	......	43.86	40.66	15.48	1.94	......	......	......	......	......	5,013	9,023
				3	......	51.90	48.10	......	2.30	......	......	......	......	......	5,931	10,676
Ralph, 5 miles southeast of; sec. 10, T. 20 N., R. 9 E.; Newcomb mine, Widow Clark bed (fresh face of open pit, 10-foot 10-inch bed, 10-foot 1-inch cut).	317	*15062	B	1	34.75	27.19	28.96	9.10	.95	......	......	......	......	26.6	3,474	6,253
				2	......	41.67	44.38	13.95	1.46	......	......	......	......	......	5,324	9,583
				3	......	48.42	51.58	......	1.70	......	......	......	......	......	6,187	11,137
PERKINS COUNTY.																
Lodgepole, 3 miles southwest of; sec. 29, T. 21 N., R. 12 E.; Nelson mine, (in entry 140 feet northwest of entry mouth, 7-foot cut).	317	*14354	B	1	33.30	28.95	27.26	10.49	.76	......	......	......	......	21.2	3,864	6,955
				2	......	43.40	40.87	15.73	1.14	......	......	......	......	......	5,793	10,427
				3	......	51.50	48.50	......	1.35	......	......	......	......	......	6,875	12,375
Strool, 6 miles northwest of; sec. 35, T. 19 N., R. 10 E.; Jones mine (end of main entry, 5-foot 4½-inch bed, 4-foot 2-inch cut).	318	*12453	B	1	39.16	24.68	27.81	8.35	2.22	6.60	38.02	.53	44.28	26.0	3,504	6,307
				2	......	40.57	45.71	13.72	3.65	3.70	62.49	.87	15.57	......	5,760	10,368
				3	......	47.02	52.98	......	4.23	4.29	72.43	1.01	18.04	......	6,676	12,017
5 miles southwest of; sec. 2, T. 17 N., R. 10 E.; Knudsen mine (end of second entry, 3-foot 6-inch bed, 2-foot 7-inch cut).	318	*12454	B	1	34.69	26.42	27.64	11.25	1.08	......	......	......	......	19.6	......	......
				2	......	40.45	42.32	17.23	1.65	......	......	......	......	......	......	......
				3	......	48.87	51.13	......	1.99	......	......	......	......	......	......	......
5 miles south of; sec. 17, T. 17 N., R. 11 W.; Phillips mine (in mine, 150 feet from mine mouth, 2-foot 5-inch bed).	318	*12488	B	1	42.46	23.24	25.30	9.00	1.16	7.07	35.22	0.62	46.93	30.1	3,308	5,954
				2	......	40.39	43.97	15.64	2.02	4.08	61.21	1.08	15.97	......	5,749	10,348
				3	......	47.88	52.12	......	2.39	4.84	72.56	1.28	18.93	......	6,815	12,267

TENNESSEE.

ANDERSON COUNTY.																
Briceville, ½ mile west of; Cross Mountain No. 1 mine, Coal Creek bed (on right rib of left entry 26, outby last break-through to the right, 3-foot 8-inch bed, 3-foot 7-inch cut).	319	13237	A	1	1.55	35.36	56.49	6.60	.75	5.08	78.39	1.83	7.35	0.5	7,804	14,047
				2		35.92	57.38	6.70	.76	4.99	79.62	1.86	6.07		7,927	14,269
				3		38.50	61.50		.81	5.35	85.34	1.99	6.51		8,496	15,293
Same, chain pillar on right entry 27 (18 feet outby twelfth and last break-through, 3-foot 10-inch bed and cut).	319	13238	A	1	1.86	33.00	53.61	11.53	.76	4.78	74.21	1.76	6.96	.7	7,306	13,151
				2		33.63	54.62	11.75	.77	4.66	75.62	1.79	5.41		7,445	13,401
				3		38.11	61.89		.87	5.28	85.09	2.03	6.13		8,436	15,185
Same (room 3 off left entry 26)	319	13264	A	1	1.70	35.02	53.14	10.14	1.06	4.97	75.32	1.80	6.71	.5	7,479	13,462
				2		35.63	54.05	10.32	1.08	4.86	76.62	1.83	5.29		7,608	13,694
				3		39.73	60.27		1.20	5.42	85.44	2.04	5.90		8,484	15,271
Same (room 2 off left entry 26)	319	13265	A	1	1.47	35.93	53.39	9.21	1.08	5.13	75.03	1.84	7.76	.5	7,564	13,615
				2		36.47	54.18	9.35	1.05	5.04	76.15	1.87	6.54		7,677	13,819
				3		40.23	59.77		1.16	5.56	84.00	2.06	7.22		8,468	15,242
Same (right rib of right entry 27, about 30 feet from main entry, 4-foot bed and cut).	319	13290	A	1	1.88	34.96	54.58	8.58	.78	5.15	76.79	1.81	6.89	.5	7,654	13,777
				2		35.63	55.63	8.74	.79	5.03	78.26	1.84	5.34		7,801	14,042
				3		39.04	60.96		.87	5.51	85.76	2.02	5.84		8,548	15,386
BLEDSOE COUNTY.																
Herbert, near; prospect, Sewanee (?) bed (in drift near mouth, 6-foot 4-inch cut).	320	10915	B	1	3.44	31.10	57.63	7.83	.56					1.9	7,485	13,473
				2		32.21	59.69	8.10	.58						7,751	13,952
				3		35.05	64.95		.63						8,434	15,181
near; prospect, Sewanee (?) bed (face of new opening, 10 feet in, 3-foot cut).	320	10916	B	1	5.32	28.81	59.62	6.25	.51					3.7	7,447	13,405
				2		30.43	62.97	6.60	.54						7,866	14,159
				3		32.58	67.42		.58						8,421	15,158
Litton, 2 miles northwest of, 15 miles northeast of Pikeville; A. L. Hale mine, Angel bed (8-foot bed and cut).	320	12584	B	1	3.33	27.89	63.17	5.61	1.88					2.2	7,850	14,130
				2		28.85	65.35	5.80	1.94						8,120	14,616
				3		30.63	69.37		2.06						8,620	15,516
4 miles north of; prospect (in main entry, 60 feet from entry mouth).	320	10808	B	1	5.04	25.48	61.81	7.67	.59					3.9	7,493	13,487
				2		26.83	65.09	8.08	.62						7,891	14,204
				3		29.19	70.81		.67						8,585	15,453
Pikeville; 6 miles southwest of, J. W. McFarland prospect, Morgan Springs bed (in drift entry, 50 feet from outcrop, 21½-inch bed).	320	10731	B	1	2.17	33.80	53.40	10.63	3.53					1.0	7,361	13,250
				2		34.55	54.58	10.87	3.61						7,524	13,543
				3		38.77	61.23		4.05						8,442	15,196
About 10 miles west of; Vaughn Bros. prospect; Sewanee (?) bed (outcrop, 3-foot 9-inch bed).	321	11049	B	1	4.41	27.16	59.85	8.58	.93					2.8	7,302	13,144
				2		28.41	62.61	8.98	.97						7,639	13,750
				3		31.21	68.79		1.07						8,393	15,107
About 10 miles west of; Vaughn Bros. prospect; Sewanee (?) bed (outcrop, 3-foot 2 -inchbed).	321	11050	B	1	4.38	26.66	62.37	6.59	.68					2.6	7,431	13,376
				2		27.88	65.23	6.89	.71						7,771	13,988
				3		29.94	70.06		.76						8,346	15,023
CUMBERLAND COUNTY.																
Litton, 4 miles north of; J. H. Hale & Son prospect, Sewanee (?) bed (in entry, 30 feet from entry mouth).	321	10800	B	1	4.30	25.71	62.51	7.48	.56					3.3	7,573	13,631
				2		26.86	65.33	7.81	.59						7,913	14,243
				3		29.14	70.86		.64						8,583	15,449
4 miles north of; J. H. Hale & Son prospect, Morgan Springs bed (in opening, 3-foot 6-inch bed).	321	10802	B	1	6.05	27.75	55.65	10.55	3.11					5.0	7,071	12,728
				2		29.54	59.23	11.23	3.31						7,526	13,547
				3		33.28	66.72		3.73						8,478	15,260

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
TENNESSEE—Continued.																
RHEA COUNTY.																
Dayton, near; North Pole mine, Richland bed (main heading of Craig workings, 3,000 feet from mine mouth, 25½-inch bed).	322	10689	B	1	1.91	29.67	54.39	14.03	1.43	4.71	71.50	1.41	6.92	0.7	7,109	12,796
				2	......	30.25	55.45	14.30	1.46	4.59	72.89	1.44	5.32	......	7,248	13,046
				3	......	35.30	64.70	......	1.70	5.36	85.06	1.68	6.20	......	8,458	15,224
Same (face of room off left heading 5, Craig workings, about 2,600 feet from mine mouth, 23-inch cut).	322	10690	B	1	1.90	29.68	53.44	14.98	1.65	4.68	71.09	1.40	6.20	.7	7,042	12,676
				2	......	30.26	54.47	15.27	1.68	4.56	72.46	1.43	4.60	......	7,179	12,922
				3	......	35.71	64.29	......	1.98	5.38	85.52	1.69	5.43	......	8,473	15,251
Same (floor of room off left heading 5, Craig workings, 2,600 feet from mine mouth, 30-inch cut).	322	10691	B	1	2.19	14.38	16.23	67.20	.33	2.25	21.55	.45	8.22	.8	2,026	3,647
				2	......	14.70	16.59	68.71	.34	2.06	22.03	.46	6.40	......	2,071	3,728
				3	......	46.98	53.02	......	1.09	6.58	70.41	1.47	20.45	......	6,619	11,914
Same (face of left heading 8, 3,000 feet from mine mouth, 22-inch bed).	322	10692	B	1	1.96	28.79	57.46	11.79	1.78	4.85	74.43	1.26	5.89	.9	7,343	13,217
				2	......	29.37	58.60	12.03	1.82	4.72	75.92	1.29	4.22	......	7,490	13,482
				3	......	33.39	66.61	......	2.06	5.37	86.31	1.47	4.79	......	8,515	15,327
Same (face of right heading 7, between 2,700 and 2,800 feet from mine mouth, 2-foot bed).	322	10693	B	1	2.08	27.91	55.10	14.91	.65	4.76	71.49	1.18	7.01	1.0	7,017	12,631
				2	......	28.50	56.27	15.23	.66	4.63	73.01	1.21	5.26	......	7,166	12,899
				3	......	33.62	66.38	......	.78	5.46	86.13	1.43	6.20	......	8,454	15,217
New Prospect mine, Nelson bed (face of room off left heading 3, 1,200 feet from mine mouth, 3-foot cut).	322	10696	B	1	1.76	27.86	49.57	20.81	.49	4.51	66.24	1.19	6.76	.8	6,481	11,666
				2	......	28.36	50.46	21.18	.50	4.39	67.43	1.21	5.29	......	6,597	11,875
				3	......	35.98	64.02	......	.63	5.57	85.55	1.54	6.71	......	8,370	15,066
Same (face of room off left heading 2, 2-foot 3-inch cut).	322	10697	B	1	3.08	29.77	52.75	14.40	.50	......	......	......	......	2.0	......	......
				2	......	30.72	54.42	14.86	.52	......	......	......	......	......	......	......
				3	......	36.08	63.92	......	.61	......	......	......	......	......	......	......
Same (end of right heading 3, 3-foot 2-inch cut)...	322	10698	B	1	1.93	27.86	49.17	21.04	.46	......	......	......	......	.8	......	......
				2	......	28.40	50.15	21.45	.47	......	......	......	......	......	......	......
				3	......	36.16	63.84	......	.60	......	......	......	......	......	......	......
Same (main heading, 1,500 to 1,600 feet from mine mouth).	322	10699	B	1	4.34	28.21	51.82	15.63	.46	......	......	......	......	3.2	......	......
				2	......	29.49	54.17	16.34	.48	......	......	......	......	......	......	......
				3	......	35.25	64.75	......	.57	......	......	......	......	......	......	......

UTAH.

CARBON COUNTY.

Hiawatha; Hiawatha No. 1 mine, Hiawatha bed (face of room 7, off left entry 1, 11-foot 2-inch bed and cut).	323	10904	A	1	7.57	39.10	47.80	5.53	.71	......	......	......	......	4.3	......	......
				2	......	42.30	51.72	5.98	.77	......	......	......	......	......	......	......
				3	......	44.99	55.01	......	.82	......	......	......	......	......	......	......
Same (face of left air course 2, 11-foot 5-inch bed and cut).	323	10905	A	1	7.36	39.67	47.09	5.88	.66	......	......	......	......	3.9	......	......
				2	......	42.82	50.83	6.35	.71	......	......	......	......	......	......	......
				3	......	45.72	54.28	......	.75	......	......	......	......	......	......	......
Same (composite of samples 10904 and 10905).....	323	10906	A	1	7.49	39.72	47.17	5.62	.64	6.09	69.12	1.35	17.18	4.1	6,956	12,521
				2	......	42.94	50.98	6.08	.69	5.69	74.72	1.46	11.36	......	7,519	13,534
				3	......	45.72	54.28	......	.73	6.06	79.55	1.55	12.11	......	8,005	14,409
Hiawatha No. 2 mine, Hiawatha bed (face of room 2, off dip entry 2, 13-foot 3-inch bed, 10-foot 3-inch cut).	323	10907	A	1	5.84	40.10	46.31	7.75	.53	......	......	......	......	2.6	......	......
				2	......	42.59	49.18	8.23	.56	......	......	......	......	......	......	......
				3	......	46.41	53.59	......	.61	......	......	......	......	......	......	......
Same (face of room 1, off rise entry 2, 11-foot 7-inch bed, 10-foot 5½-inch cut).	323	10908	A	1	5.41	41.25	46.94	6.40	.58	......	......	......	......	2.2	......	......
				2	......	43.61	49.62	6.77	.61	......	......	......	......	......	......	......
				3	......	46.78	53.22	......	.65	......	......	......	......	......	......	......
Same (composite of samples 10907 and 10908).....	323	10909	A	1	5.65	40.79	46.61	6.95	.56	5.96	69.46	1.39	15.68	2.4	7,026	12,647
				2	......	43.23	49.40	7.37	.59	5.66	73.62	1.47	11.29	......	7,447	13,405
				3	......	46.67	53.33	......	.64	6.11	79.48	1.59	12.18	......	8,040	14,472
Price, 9 miles northeast of; sec. 7, T. 13 S., R. 11 E.; Jesse Knight prospect (face of entry, 90 feet northwest of entry mouth, 6-foot 8-inch bed, 6-foot 5-inch cut).	324	14678	B	1	5.68	41.87	47.68	4.77	.58	......	......	......	......	.3	7,021	12,638
				2	......	44.39	50.55	5.06	.61	......	......	......	......	......	7,444	13,399
				3	......	46.76	53.24	......	.64	......	......	......	......	......	7,841	14,114
Sunnyside, sec. 32, T. 14 S., R. 14 E.; No. 1 mine, lower bed (face of Fowler slope; 10-foot bed, 5-foot 5-inch cut).	324	12632	B	1	5.96	38.68	48.77	6.59	1.73	5.43	71.28	1.52	13.45	3.4	7,134	12,841
				2	......	41.13	51.86	7.01	1.84	5.07	75.80	1.62	8.66	......	7,586	13,655
				3	......	44.23	55.77	......	1.98	5.45	81.52	1.74	9.31	......	8,158	14,684
1 mile southeast of; sec. 4, T. 15 S., R. 14 E.; No. 3 mine, upper bed (rib of air course, 50 feet northwest of outlet, 6-foot 2-inch bed).	325	12546	B	1	5.96	38.10	48.82	7.13	.56	......	......	......	......	1.0	6,663	11,993
				2	......	40.51	51.91	7.58	.60	......	......	......	......	......	7,085	12,753
				3	......	43.83	56.17	......	.65	......	......	......	......	......	7,666	13,799
Same (face of second crosscut, off right entry 1, off No. 1 rise, 1¾ miles southeast of mine mouth, 4-foot 8-inch bed).	325	12630	B	1	5.06	38.45	48.68	7.81	.51	5.54	70.72	1.58	13.84	1.5	7,011	12,620
				2	......	40.50	51.27	8.23	.54	5.25	74.49	1.66	9.83	......	7,385	13,293
				3	......	44.13	55.87	......	.59	5.72	81.17	1.81	10.71	......	8,047	14,485
1 mile southeast of, No. 3 mine, lower bed (on dip entry, about 2¾ miles southeast of opening at Sunnyside, 6-foot 10-inch bed).	325	12631	B	1	5.08	39.94	47.69	7.29	1.37	......	......	......	......	2.1	7,094	12,769
				2	......	42.08	50.24	7.68	1.44	......	......	......	......	......	7,474	13,453
				3	......	45.58	54.42	......	1.56	......	......	......	......	......	8,096	14,573
¾ mile south of; sec. 5, T. 15 S., R. 14 E.; prospect, lower bed (85 feet northeast of mouth of prospect tunnel, 13-foot 10-inch bed, 6-foot cut).	325	*12545	B	1	15.17	33.87	46.19	4.77	.60	......	......	......	......	2.5	5,293	9,527
				2	......	39.93	54.45	5.62	.71	......	......	......	......	......	6,239	11,230
				3	......	42.31	57.69	......	.75	......	......	......	......	......	6,610	11,898
About 8½ miles northwest of; sec. 32, T. 13 S., R. 13 E.; prospect (face of tunnel, 200 feet northwest of opening, 5-foot 7-inch bed).	325	12792	B	1	6.67	38.58	49.54	5.21	1.04	......	......	......	......	2.9	6,728	12,110
				2	......	41.34	53.08	5.58	1.11	......	......	......	......	......	7,209	12,976
				3	......	43.78	56.22	......	1.18	......	......	......	......	......	7,635	13,743
12 miles northwest of; sec. 23, T. 13 S., R. 12 E.; prospect (in entry, 200 feet southeast of entry mouth, 8-foot 6-inch bed, 8-foot 2½-inch cut).	326	14801	B	1	8.54	39.76	47.86	3.84	.48	......	......	......	......	5.3	7,115	12,807
				2	......	43.47	52.33	4.20	.52	......	......	......	......	......	7,780	14,004
				3	......	45.37	54.63	......	.54	......	......	......	......	......	8,121	14,618

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
UTAH—Continued.																
EMERY COUNTY.																
Emery, 4 miles south of; NE. ¼ NE. ¼ sec. 33, T. 22 S., R. 6 E.; Browning mine, I bed (back of entry, 165 feet northeast of mouth, 22-foot bed, 12-foot cut.)	326	12627	B	1	3.93	40.92	49.22	5.93	.89	5.52	73.02	1.25	13.89	0.3	7,203	12,965
				2	....	42.59	51.24	6.17	.41	5.29	76.01	1.30	10.82	....	7,498	13,496
				3	....	45.39	54.61	....	.44	5.64	81.01	1.39	11.52	....	7,991	14,384
4 miles southeast of; SE. ¼ NW. ¼ sec. 26, T. 22 S., R. 6 E., Casper mine, C bed, back end of main entry, 200 feet northwest of entry mouth, 9-foot 10-inch bed, 7-foot 3½-inch cut).	327	12652	B	1	5.21	39.06	41.61	14.12	.81	....	....	....	....	1.8	6,263	11,273
				2	....	41.21	43.89	14.90	.85	....	....	....	....	....	6,607	11,893
				3	....	48.43	51.57	....	1.00	....	....	....	....	....	7,764	13,975
12 miles south of; SW. ¼ SW. ¼ sec. 32, T. 23 S., R. 6 E.; surface prospect, I bed (cave in rock ledge, 12-foot 3-inch bed, 10-foot 5-inch cut).	327	* 14903	B	1	16.67	34.29	41.94	7.10	1.12	....	....	....	....	8.5	5,277	9,499
				2	....	41.15	50.33	8.52	1.34	....	....	....	....	....	6,332	11,398
				3	....	44.98	55.02	....	1.46	....	....	....	....	....	6,922	12,460
2½ miles east of; SE. ¼, SW. ¼ sec. 12, T. 22 S., R. 6 E.; Williams mine, I bed (back of main entry, 200 feet southeast of mine mouth).	328	12613	B	1	3.95	41.83	42.60	11.62	4.66	....	....	....	....	.9	6,661	11,990
				2	....	43.55	44.35	12.10	4.85	....	....	....	....	....	6,935	12,483
				3	....	49.55	50.45	....	5.52	....	....	....	....	....	7,890	14,202
SEVIER COUNTY.																
Emery, 15 miles south of; SW. ¼ NW. ¼ sec. 25, T. 24 S., R. 5 E.; surface prospect, A bed (near outcrop at prospect, 11-foot 3-inch bed, 11-foot cut).	328	* 15061	B	1	18.41	33.86	42.16	5.57	.41	....	....	....	....	11.6	5,270	9,486
				2	....	41.50	51.67	6.83	.50	....	....	....	....	....	6,459	11,626
				3	....	44.54	55.46	....	.54	....	....	....	....	....	6,932	12,478
Fremont, 15 miles northeast of; SW. ¼ NW. ¼ sec. 3, T. 26 S., R. 4 E.; Hogan prospect, A bed (back end of drift, about 20 feet northwest of mouth, 5-foot 1-inch bed, 3-foot cut).	328	* 15090	B	1	23.57	32.61	33.25	10.57	2.88	....	....	....	....	17.0	4,346	7,823
				2	....	42.67	43.50	13.83	3.77	....	....	....	....	....	5,686	10,235
				3	....	49.52	50.48	....	4.38	....	....	....	....	....	6,599	11,878
Salina, about 15 miles east of; sec. 7, T. 22 S., R. 3 E.; SW. ¼ SW. ¼; Kearns & Duggins mine, Duggins bed (in room off main entry, about 175 feet north of entry mouth, 3-foot 9-inch bed, 3-foot 7-inch cut).	329	15045	B	1	6.38	45.10	36.28	12.24	.44	....	....	....	....	3.1	6,529	11,752
				2	....	48.17	38.76	13.07	.47	....	....	....	....	....	6,974	12,553
				3	....	55.41	44.59	....	.54	....	....	....	....	....	8,022	14,440

SUMMIT COUNTY.																
Coalville, 6 miles northeast of; SW. ¼ sec. 24, T. 3 N., R. 5 E.; Rees-Grass Creek mine, Wasatch bed (in mine, 400 feet below surface, and 500 feet east of main slope, 8-foot 6½-inch bed and cut).	329	13216	B	1	12.18	42.18	42.25	3.39	1.90	5.80	63.77	1.27	23.87	4.5	6,254	11,257
				2	....	48.03	48.11	3.86	2.16	5.07	72.01	1.45	14.85	....	7,121	12,818
				3	....	49.96	50.04	....	2.25	5.27	75.52	1.51	15.45	....	7,407	13,333
1 mile southeast of; sec. 16, T. 2 N., R. 5 E.; Superior mine, Wasatch bed (in mine, 1,000 feet northwest of foot of slope, 7-foot 6-inch bed and cut).	329	13217	B	1	17.08	36.94	41.24	4.74	1.53	....	....	....	....	7.8	5,655	10,179
				2	....	44.55	49.73	5.72	1.85	....	....	....	....	....	6,820	12,276
				3	....	47.25	52.75	....	1.96	....	....	....	....	....	7,234	13,021
3 miles northeast of; SW. ¼ sec. 3, T. 2 N., R. 5 E; Wasatch mine, Wasatch bed (800-foot level, 3,000 feet northeast of main slope, 10-foot 6-inch bed and cut).	329	13218	B	1	13.61	41.28	41.03	4.08	1.37	5.84	62.11	1.06	25.54	5.3	6,100	10,980
				2	....	47.78	47.50	4.72	1.59	5.01	71.89	1.23	15.56	....	7,061	12,710
				3	....	50.15	49.85	....	1.67	5.26	75.45	1.29	16.33	....	7,411	13,340
UINTA COUNTY.																
Vernal, 7 miles north of; sec. 8, T. 3 S., R. 22 E.; Blue Bell mine (face of main entry, about 225 feet from mine mouth, 7-foot 8½-inch bed, 6-foot 5½-inch cut).	330	10712	B	1	9.78	37.19	43.50	9.53	1.49	....	....	....	....	1.8	....	....
				2	....	41.22	48.22	10.56	1.65	....	....	....	....	....	....	....
				3	....	46.09	53.91	....	1.84	....	....	....	....	....	....	....
8 miles east of; Green mine (face of main entry, 75 feet from mine mouth, 4-foot 8¾-inch bed, 2-foot 6-inch cut).	330	10713	B	1	8.76	38.84	44.16	8.24	2.36	....	....	....	....	1.1	....	....
				2	....	42.57	48.40	9.03	2.59	....	....	....	....	....	....	....
				3	....	46.80	53.20	....	2.85	....	....	....	....	....	....	....
About 20 miles west of; Reynolds mine, Mancos bed (at face of main entry about 210 feet from surface).	331	10812	B	1	11.25	35.61	46.43	6.71	.95	....	....	....	....	4.5	6,324	11,383
				2	....	40.13	52.31	7.56	1.07	....	....	....	....	....	7,126	12,827
				3	....	43.41	56.59	....	1.16	....	....	....	....	....	7,709	13,876
WASATCH COUNTY.																
Hanna, about 15 miles southwest of; sec. 26, T. 1 S., R. 9 W.; prospect (face of entry, 100 feet from mouth of entry, 24-foot 8-inch cut).	331	10096	B	1	14.58	38.48	41.00	5.94	.82	....	....	....	....	9.0	5,768	10,382
				2	....	45.05	48.00	6.95	.96	....	....	....	....	....	6,753	12,155
				3	....	48.42	51.58	....	1.08	....	....	....	....	....	7,257	13,063
About 3 miles east of; sec. 7, T. 1 S., R. 7 W.; Winchester prospect, Mancos bed (in entry, about 10 feet from entry mouth).	332	10097	B	1	27.40	32.14	37.87	2.59	.60	....	....	....	....	19.0	4,125	7,425
				2	....	44.27	52.16	3.57	.83	....	....	....	....	....	5,682	10,226
				3	....	45.91	54.09	....	.86	....	....	....	....	....	5,892	10,606
Heber, about 28 miles southeast of; sec. 31, T. 1 S., R. 10 E.; Cummings mine (surface exposure, 13-foot bed, 10-foot 6-inch cut).	332	11058	B	1	19.15	37.04	37.60	6.21	.69	....	....	....	....	5.6	4,846	8,723
				2	....	45.81	46.51	7.68	.85	....	....	....	....	....	5,994	10,789
				3	....	49.62	50.38	....	.92	....	....	....	....	....	6,493	11,687
VIRGINIA.																
DICKENSON COUNTY.																
Clintwood, 3 miles southeast of; Elbert Powers mine, Clintwood (?) bed (in main entry, 20 feet from mine mouth, 7-foot 10½-inch bed, 4-foot 8-inch cut).	332	14767	B	1	4.28	29.40	62.11	4.21	1.02	....	....	....	....	3.3	....	....
				2	....	30.71	64.89	4.40	1.07	....	....	....	....	....	....	....
				3	....	32.12	67.88	....	1.12	....	....	....	....	....	....	....
Near, 20 miles north of Coeburn; Yeates mine, Clintwood bed (at end of right entry 1, off main entry, 100 feet from main entry, 4-foot 2¾-inch bed and cut).	333	14766	B	1	2.18	34.19	60.44	3.19	.85	....	....	....	....	1.2	8,108	14,594
				2	....	34.95	61.79	3.26	.87	....	....	....	....	....	8,289	14,920
				3	....	36.13	63.87	....	.90	....	....	....	....	....	8,568	15.422

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
VIRGINIA—Continued.																
HENRICO COUNTY.																
Gayton, 1 mile from, 5 miles north of Lorraine; Carbon Hill mine, C bed (in main gangway, opposite room 45, 3-foot 6-inch bed, 3-foot 4-inch cut).	333	15050	A	1	2.81	25.70	62.47	9.02	1.43	4.90	76.55	1.81	6.29	1.9	7,496	13,493
				2	......	26.44	64.28	9.28	1.47	4.72	78.76	1.86	3.91	......	7,713	13,883
				3	......	29.14	70.86	......	1.62	5.20	86.82	2.05	4.31	......	8,502	15,304
Same (in main gangway opposite room 32, 3-foot 3-inch bed, 2-foot 3-inch cut).	333	15052	A	1	2.54	16.28	70.24	10.94	1.31	......	......	......	......	1.9	......	......
				2	......	16.70	72.07	11.23	1.34	......	......	......	......	......	......	......
				3	......	18.81	81.19	......	1.51	......	......	......	......	......	......	......
1 mile from, 5 miles north of Lorraine; Carbon Hill mine, B bed (face of right gangway off slope 3, 20 feet beyond room 21, 4-foot 11-inch bed, 4-foot 9-inch cut).	333	15051	A	1	2.11	23.58	56.95	17.36	2.16	4.44	69.22	1.59	5.23	1.4	6,778	12,200
				2	......	24.09	58.18	17.73	2.21	4.30	70.72	1.62	3.42	......	6,924	12,463
				3	......	29.28	70.72	......	2.69	5.23	85.96	1.97	4.15	......	8,416	15,149
RUSSELL COUNTY.																
Dante, 2 miles north of; No. 2 mine, Upper Banner bed (face of butt heading off fourth left drift off McClure entry, 5-foot 2½-inch bed, 5-foot 1-inch cut).	334	10734	A	1	2.34	35.23	56.16	6.27	.59	......	......	......	......	1.5	7,832	14,098
				2	......	36.08	57.50	6.42	.60	......	......	......	......	......	8,020	14,436
				3	......	38.56	61.44	......	.64	......	......	......	......	......	8,570	15,426
Same (room 16, off right heading 3, off first drift off McClure entry, 5-foot 1½-inch bed, 4-foot 9½-inch cut).	334	10735	A	1	2.99	35.16	56.66	5.19	.59	......	......	......	......	2.1	7,860	14,148
				2	......	36.24	58.41	5.35	.61	......	......	......	......	......	8,102	14,584
				3	......	38.29	61.71	......	.64	......	......	......	......	......	8,560	15,408
Same (composite of samples 10734 and 10735).....	334	10736	A	1	2.76	34.96	56.51	5.77	.59	5.32	80.18	1.43	6.76	1.8	7,860	14,148
				2	......	35.95	58.12	5.93	.61	5.15	82.41	1.47	4.43	......	8,063	14.549
				3	......	38.21	61.79	......	.65	5.47	87.60	1.56	4.72	......	8,591	15,464
2 miles north of; No. 5 mine, Upper Banner bed (room 14, on right heading 5, off ninth drift, 5-foot 2½-inch bed, 5-foot ½-inch cut).	334	10737	A	1	2.29	35.18	57.79	4.74	.55	......	......	......	......	1.4	8,017	14,431
				2	......	36.00	59.15	4.85	.56	......	......	......	......	......	8,205	14,769
				3	......	37.84	62.16	......	.59	......	......	......	......	......	8,623	15.521
Same (butt on left heading 9, off ninth drift, 5-foot 5½-inch bed, 5-foot 3-inch cut).	334	10738	A	1	2.11	35.11	57.13	5.65	.56	......	......	......	......	1.2	7,931	14,276
				2	......	35.87	58.36	5.77	.58	......	......	......	......	......	8,102	14,584
				3	......	38.06	61.94	......	.62	......	......	......	......	......	8,598	15,476
Same (composite of samples 10737 and 10738).....	334	10739	A	1	2.07	35.90	56.70	5.33	.57	5.45	80.24	1.43	6.98	1.3	7,964	14,335
				2	......	36.66	57.90	5.44	.58	5.33	81.93	1.46	5.26	......	8,132	14,638
				3	......	38.77	61.23	......	.61	5.64	86.64	1.54	5.57	......	8,600	15,480

WISE COUNTY.																
Flat Gap, 2 miles east of; Reuben Bolling mine, Lower Bolling bed (in entry, 15 feet from mine mouth, 4-foot 1-inch bed, 2-foot 11-inch cut).	335	15174	B	1	6.92	30.37	54.57	8.14	.95	....	....	....	....	5.8	7,209	12,976
				2	....	32.63	58.63	8.74	1.02	....	....	....	....	....	7,745	13,941
				3	....	35.76	64.24	....	1.12	....	....	....	....	....	8,487	15,277
Glamorgan; Glamorgan No. 3 mine, Glamorgan bed (face of dip entry 2; 6,000 feet from mine mouth, 4-foot 5½-inch bed, 4-foot 3-inch cut).	335	15100	B	1	2.66	33.08	59.27	4.99	1.37	....	....	....	....	1.6	7,897	14,215
				2	....	33.98	60.89	5.13	1.41	....	....	....	....	....	8,113	14,603
				3	....	35.82	64.18	....	1.49	....	....	....	....	....	8,552	15,394
Same (face of Glady entry), 7,000 feet from mine mouth, 4-foot 4-inch bed, 4-foot 3-inch cut).	335	15101	B	1	3.26	31.30	59.07	6.37	.87	5.27	78.02	1.65	7.82	2.0	7,728	13,910
				2	....	32.35	61.07	6.58	.90	5.08	80.65	1.71	5.08	....	7,988	14,378
				3	....	34.63	65.37	....	.96	5.44	86.33	1.83	5.44	....	8,550	15,390
Pardee; Pardee mine, Pardee bed (face of main entry, 2,000 feet from mine mouth, 9-foot 7-inch bed and cut).	336	15099	B	1	2.29	33.84	54.66	9.21	1.56	4.99	74.46	1.57	8.21	1.0	7,421	13,358
				2	....	34.63	55.94	9.43	1.60	4.85	76.20	1.61	6.31	....	7,595	13,671
				3	....	38.23	61.77	....	1.77	5.35	84.13	1.78	6.97	....	8,386	15,095
WASHINGTON.																
KING COUNTY.																
Grand Ridge, 3 miles east of; NE. ¼ sec. 26 N., T. 24, R. 6 E.; Grand Ridge mine, No. 3 or Reynolds bed (left side of chute 4, 6-foot 2-inch bed, 3-foot 6-inch cut).	336	11736	B	1	15.92	36.04	38.47	9.57	.49	....	....	....	....	9.0	5,537	9,967
				2	....	42.86	45.76	11.38	.58	....	....	....	....	....	6,585	11,853
				3	....	48.36	51.64	....	.65	....	....	....	....	....	7,431	13,376
Same (No. 4 bed, north side of rock tunnel, where it cuts bed, 3-foot 5-inch bed, 3-foot 1-inch cut).	336	11737	B	1	15.60	33.41	30.36	20.63	2.27	....	....	....	....	9.6	4,662	8,392
				2	....	39.58	35.98	24.44	2.69	....	....	....	....	....	5,524	9,943
				3	....	52.38	47.62	....	3.56	....	....	....	....	....	7,311	13,160
Same (No. 7 bed, 7-foot 7½-inch bed, 5-foot 2-inch cut).	336	11738	B	1	16.45	34.63	36.38	12.54	.38	....	....	....	....	10.6	5,323	9,581
				2	....	41.45	43.54	15.01	.45	....	....	....	....	....	6,371	11,468
				3	....	48.77	51.23	....	.53	....	....	....	....	....	7,496	13,493
KITTITAS COUNTY.																
Ellensburg, 25 miles west of, sec. 22, T. 18 N., R. 15 E.; prospect, No. 7 bed (face of entry, 30 feet from entry mouth, 2-foot 10½-inch bed, 2-foot 9½-inch cut).	337	13266	B	1	10.42	30.33	36.43	22.82	1.13	....	....	....	....	6.7	4,988	8,978
				2	....	33.86	40.67	25.47	1.26	....	....	....	....	....	5,568	10,022
				3	....	45.43	54.57	....	1.69	....	....	....	....	....	7,471	13,448
25 miles west of, sec. 16, T. 18 N., R. 15 E.; prospect, No. 10d bed (face of entry, 35 feet from entry mouth, 3-foot 9½-inch bed, 3-foot 1½-inch cut).	337	13267	B	1	5.08	31.87	40.43	22.62	1.14	....	....	....	....	2.9	5,751	10,352
				2	....	33.58	42.59	23.83	1.20	....	....	....	....	....	6,059	10,906
				3	....	44.09	55.91	....	1.58	....	....	....	....	....	7,955	14,319
25 miles west of, sec. 22, T. 18 N., R. 15 E.; prospect, 9c bed (face of entry, 40 feet from entry mouth, 3-foot 6-inch bed, 3-foot 2-inch cut).	338	13268	B	1	7.45	37.52	47.88	7.15	.64	....	....	....	....	4.1	6,701	12,062
				2	....	40.54	51.73	7.73	.69	....	....	....	....	....	7,240	13,032
				3	....	43.94	56.06	....	.75	....	....	....	....	....	7,847	14,125
Roslyn, secs. 8 and 9, T. 20 N., R. 15 E.; Roslyn No. 1 mine, Roslyn bed (face of southwest slant entry, 4-foot 2½-inch bed and cut).	338	11522	A	1	4.22	36.59	46.28	12.91	.38	....	....	....	....	2.4	....	....
				2	....	38.20	48.32	13.48	.40	....	....	....	....	....	....	....
				3	....	44.15	55.85	....	.46	....	....	....	....	....	....	....
Same (face of east entry 1, 3-foot 11½-inch bed, 3-foot 9½-inch cut).	338	11523	A	1	3.91	36.86	46.48	12.75	.37	....	....	....	....	2.1	....	....
				2	....	38.36	48.37	13.27	.39	....	....	....	....	....	....	....
				3	....	44.23	55.77	....	.45	....	....	....	....	....	....	....
Same (face of west entry 2, 3-foot 11¾-inch bed and cut).	338	11524	A	1	4.08	37.23	46.96	11.73	.39	....	....	....	....	2.5	....	....
				2	....	38.81	48.96	12.23	.41	....	....	....	....	....	....	....
				3	....	44.22	55.78	....	.47	....	....	....	....	....	....	....
Same (composite of samples 11522–11524)	338	11525	A	1	3.89	37.00	46.49	12.62	.37	5.58	68.55	1.31	11.57	2.3	6,908	12,434
				2	....	38.50	48.37	13.13	.38	5.36	71.33	1.36	8.44	....	7,188	12,938
				3	....	44.32	55.68	....	.44	6.17	82.11	1.57	9.71	....	8,274	14,893

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WASHINGTON—Continued.																
KITTITAS COUNTY—continued.																
Roslyn; Roslyn No. 2 mine, Roslyn bed (face of main slope, 4-foot 3-inch bed and cut).	338	11526	A	1	3.77	37.69	47.05	11.49	.47	5.58	70.17	1.52	10.77	2.2	7,090	12,762
				2		39.17	48.89	11.94	.49	5.36	72.92	1.58	7.71		7,368	13,262
				3		44.48	55.52		.56	6.09	82.81	1.79	8.75		8,367	15,061
Thorp, 10 miles west of, sec. 33, T. 19 N., R. 16 E.; Joseph Wilson mine (run-of-mine coal taken from bunkers).	339	13269	B	1	7.69	39.63	36.51	16.17	.43					3.1	5,547	9,985
				2		42.93	39.55	17.52	.47						6,009	10,816
				3		52.05	47.95		.57						7,285	13,113
LEWIS COUNTY.																
Mendota; Mendota No. 1 mine, No. 3 bed (rib of second level; 10-foot bed).	339	12305	A	1	7.88	61.57	15.11	15.44	.29	6.73	58.92	.50	18.12	3.7	6,622	11,920
				2		66.83	16.41	16.76	.31	6.35	63.96	.54	12.08		7,188	12,938
				3		80.28	19.72		.37	7.63	76.84	.65	14.51		8,635	15,543
Same (rib in south level, 10-foot bed)	339	14084	A	1	3.18	80.47	12.04	4.31	.28					.4		
				2		83.11	12.44	4.45	.29							
				3		86.98	13.02		.30							
Morton, 2 miles southeast of, SW. ¼ SW. ¼ sec. 12, T. 12 N., R. 4 E.; prospect, Edlund bed (4 feet from surface, 4-foot cut).	340	*12563	B	1	12.62	28.59	36.89	21.90	.73					5.1		
				2		32.72	42.22	25.06	.84							
				3		43.66	56.34		1.12							
2 miles east of, SW. ¼ sec. 6, T. 12 N., R. 5 E.; prospect (in entry, 20 feet from entry mouth, 2-foot 10-inch bed, 2-foot cut).	340	*12564	B	1	8.02	22.74	53.24	16.00	.67					4.1		
				2		24.72	57.88	17.40	.73							
				3		29.93	70.07		.88							
4 miles north of, sec. 14, T. 13 N., R. 4 E.; prospect (about 60 feet from entry mouth, 14+-foot bed, 6+-foot cut).	341	12565	B	1	11.84	8.40	57.30	22.46	.79					6.2	5,384	9,691
				2		9.53	64.99	25.48	.90						6,107	10,993
				3		12.79	87.21		1.21						8,195	14,751
PIERCE COUNTY.																
Ashford; NW. ¼ sec. 15, T. 15 N., R. 6 E.; prospect, Misqualy Chief bed (in entry, 100 feet from mouth of entry, 13-foot 4-inch bed, 11-foot 9-inch cut).	341	12581	B	1	5.81	15.28	54.68	24.23	0.41					3.6	5,782	10,408
				2		16.22	58.06	25.72	.44						6,139	11,050
				3		21.84	78.16		.59						8,265	14,877
Fairfax, 2 miles south of; SW. ¼ sec. 2, T. 17 N., R. 6 E.; prospect, Montezuma bed (face of entry, 45 feet from mouth of entry, 9-foot 7-inch bed, 4-foot cut).	342	12478	B	1	3.57	20.60	41.93	33.90	.55					2.3	3,455	6,219
				2		21.36	43.49	35.15	.57						3,583	6,449
				3		32.94	67.06		.88						5,525	9,945

½ mile south of; NE. ¼ NW. ¼ sec. 34, T. 18 N., R. 6 E.; prospect No. 1, No. 1 bed (in entry, 35 feet from mouth of entry, 4-foot 1-inch bed, 2-foot 10-inch cut).	342	12496	B	1	4.75	26.37	60.82	8.06	1.13					4.0	7,574	13,633
				2		27.69	63.85	8.46	1.19						7,952	14,314
				3		30.25	69.75		1.30						8,687	15,637
½ mile south of; NE. ¼ NW. ¼ sec. 34, T. 18 N., R. 6 E.; prospect No. 2, No. 2 bed (in entry, 35 feet from mouth of entry, 5-foot 5-inch bed, 2-foot 11-inch cut).	342	12495	B	1	2.56	24.81	52.79	19.84	.68					1.7	6,586	11,855
				2		25.46	54.18	20.36	.70						6,759	12,166
				3		31.97	68.03		.88						8,487	15,277
Wilkeson; Gale Creek mine, Queen bed (face of north gangway 1, off new slope, 3-foot 8½-inch bed, 3-foot 8¾-inch cut).	343	W31921	A	1	2.4	35.7	52.1	9.8	.85						7,556	13,600
				2		36.6	53.4	10.0	.90						7,744	13,940
				3		40.7	59.3		1.00						8,606	15,490
Same (position, bed, and cut same as No. W31921).	343	W31922	A	1	2.4	35.8	51.6	10.2	.90						7,500	13,500
				2		36.7	52.9	10.4	.95						7,683	13,830
				3		41.0	59.0		1.06						8,578	15,440
Same (breast 16 at face of counter No. 1 north, off new slope, 3-foot 7¾-inch bed and cut).	343	W31923	A	1	2.6	35.4	51.7	10.3	.80						7,461	13,430
				2		36.3	53.1	10.6	.85						7,661	13,790
				3		40.6	59.4		.95						8,567	15,420
Same (position, bed, and cut same as No. W31923).	343	W31924	A	1	2.6	35.3	52.1	10.0	.85						7,511	13,520
				2		36.2	53.5	10.3	.90						7,711	13,880
				3		40.4	59.6		1.00						8,594	15,470
Same (pillar of breast 10, off north gangway 1, off new slope, 3-foot 9-inch bed, 3-foot 5½-inch cut).	343	W31925	A	1	2.9	35.7	51.8	9.6	.65						7,517	13,530
				2		36.8	53.3	9.9	.70						7,739	13,930
				3		40.8	59.2		.78						8,599	15,460
Same (position, bed, and cut same as No. W31925).	343	W31926	A	1	2.7	35.8	51.9	9.6	.70						7,528	13,550
				2		36.8	53.3	9.9	.70						7,733	13,920
				3		40.8	59.2		.78						8,583	15,450
Same (left air course 1 of new slope, below north gangway 1, 3-foot 8¼-inch bed, 3-foot ¾-inch cut).	343	W32106	A	1	2.4	34.9	52.4	10.3	1.00						7,411	13,340
				2		35.7	53.7	10.6	1.05						7,589	13,660
				3		39.9	60.1		1.17						8,489	15,280
Same (pillar of first crosscut on breast 14, off north gangway 1, off new slope, 3-foot 7¼-inch bed, 3-foot 4-inch cut).	343	W32107	A	1	2.6	35.5	53.1	8.8	.96						7,583	13,650
				2		36.4	54.6	9.0	1.00						7,783	14,010
				3		40.0	60.0		1.10						8,550	15,390
Same (face of crosscut on new slope opposite south gangway 2, 4-foot 8-inch bed, 3-foot 9-inch cut).	343	W32108	A	1	3.5	34.8	52.2	9.5	.90						7,422	13,360
				2		36.0	54.2	9.8	.95						7,689	13,840
				3		39.9	60.1		1.05						8,522	15,340
Same (composite of samples W32106–W32108 and W31921–W31926).	343	14390	A	1	1.91	35.13	53.08	9.88	.86	5.72	75.10	2.19	6.25		7,549	13,588
				2		35.82	54.11	10.07	.88	5.62	76.56	2.23	4.64		7,696	13,853
				3		39.83	60.17		.98	6.25	85.13	2.48	5.16		8,558	15,404
Same (from 4 cars, washed)	343	W32104	C	1	1.4	24.6	64.1	9.9	.85						7,594	13,670
				2		25.0	65.0	10.0	.85						7,706	13,870
				3		27.8	72.2		.94						8,561	15,410
Same (from same cars as No. W32104)	343	W32105	C	1	1.4	35.0	53.0	10.6	1.10						7,500	13,500
				2		35.5	53.7	10.8	1.10						7,606	13,690
				3		39.8	60.2		1.23						8,528	15,350
Same (from 4 cars, washed)	343	W32145	C	1	1.9	34.6	52.4	11.1	.65						7,433	13,380
				2		35.3	53.4	11.3	.65						7,578	13,640
				3		39.8	60.2		.73						8,544	15,380
Same (from same cars as No. W32145)	343	W32146	C	1	1.9	34.7	51.6	11.8	.90						7,339	13,210
				2		35.4	52.6	12.0	.90						7,478	13,460
				3		40.2	59.8		1.02						8,500	15,300

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WASHINGTON—Continued.																
PIERCE COUNTY—continued.																
Wilkeson; Gale Creek mine, Queen bed (from 4 cars, washed).	343	W32147	C	1	2.7	34.2	52.4	10.7	.85	......	......	......	......	......	7,389	13,300
				2	......	35.1	53.9	11.0	.90	......	......	......	......	......	7,594	13,670
				3	......	39.4	60.6	......	1.01	......	......	......	......	......	8,533	15,360
Same (from 4 cars, washed)	343	W32158	C	1	3.6	33.3	51.2	11.9	.90	......	......	......	......	......	7,239	13,030
				2	......	34.5	53.2	12.3	.95	......	......	......	......	......	7,506	13,510
				3	......	39.3	60.7	......	1.08	......	......	......	......	......	8,561	15,410
Same (from same cars as No. W32158)	343	W32159	C	1	3.6	33.7	51.1	11.6	.90	......	......	......	......	......	7,250	13,050
				2	......	35.0	53.0	12.0	.95	......	......	......	......	......	7,522	13,540
				3	......	39.8	60.2	......	1.08	......	......	......	......	......	8,550	15,390
Same (from 4 cars, washed)	343	W32160	C	1	3.5	34.6	50.9	11.0	.95	......	......	......	......	......	7,317	13,170
				2	......	35.9	52.7	11.4	1.00	......	......	......	......	......	7,583	13,650
				3	......	40.5	59.5	......	1.13	......	......	......	......	......	8,561	15,410
Same (from same cars as No. W32160)	343	W32161	C	1	3.5	34.7	50.8	11.0	.80	......	......	......	......	......	7,328	13,190
				2	......	36.0	52.6	11.4	.85	......	......	......	......	......	7,594	13,670
				3	......	40.6	59.4	......	.96	......	......	......	......	......	8,572	15,430
Same (from 4 cars, washed)	343	W32211	C	1	3.7	34.2	51.2	10.9	.90	......	......	......	......	......	7,311	13,160
				2	......	35.5	53.2	11.3	.95	......	......	......	......	......	7,594	13,670
				3	......	40.0	60.0	......	1.07	......	......	......	......	......	8,561	15,410
Same (from same cars as No. W32211)	343	W32212	C	1	3.4	34.7	51.3	10.6	.85	......	......	......	......	......	7,322	13,180
				2	......	35.9	53.1	11.0	.90	......	......	......	......	......	7,583	13,650
				3	......	40.3	59.7	......	1.01	......	......	......	......	......	8,522	15,340
Same (from 3 cars, washed)	343	W32213	C	1	3.0	34.7	51.7	10.6	.90	......	......	......	......	......	7,344	13,220
				2	......	35.8	53.3	10.9	.95	......	......	......	......	......	7,572	13,630
				3	......	40.2	59.8	......	1.07	......	......	......	......	......	8,500	15,300
Same (from same cars as No. W32213)	343	W32214	C	1	3.0	34.6	51.9	10.5	.85	......	......	......	......	......	7,383	13,290
				2	......	35.7	53.5	10.8	.90	......	......	......	......	......	7,611	13,700
				3	......	40.0	60.0	......	1.01	......	......	......	......	......	8,533	15,360
WEST VIRGINIA.																
BRAXTON COUNTY.																
Braxton; Braxton mine, Pittsburgh bed (in room, about 325 feet southeast of mine mouth, 5-foot 3-inch bed and cut).	344	13556	B	1	2.83	40.90	47.59	8.68	3.40	......	......	......	......	1.2	7,382	13,288
				2	......	42.09	48.98	8.93	3.50	......	......	......	......	......	7,597	13,675
				3	......	46.22	53.78	......	3.84	......	......	......	......	......	8,342	15,016

BROOKE COUNTY.																
Colliers, 1 mile west of; Lewis-Finley No. 1 mine, Pittsburgh bed (room 1, off butt entry 6, off main entry, 4-foot 7½-inch bed, 3-foot 6-inch cut).	345	15671	B	1	4.58	35.36	54.05	6.01	1.28	5.28	73.81	1.55	12.07	2.6	7,352	13,234
				2	....	37.06	56.64	6.30	1.34	5.00	77.35	1.62	8.39	....	7,705	13,869
				3	....	39.55	60.45	....	1.43	5.34	82.55	1.73	8.95	....	8,223	14,801
Same (room 77, off entry 10, side No. 2, 5,800 feet northwest of mine mouth, 4-foot 8-inch bed, 4-foot ½-inch cut).	345	15672	B	1	4.13	36.62	53.26	5.99	2.07	....	....	....	....	1.5	7,446	13,403
				2	....	38.20	55.55	6.25	2.16	....	....	....	....	....	7,767	13,981
				3	....	40.75	59.25	....	2.30	....	....	....	....	....	8,285	14,913
FAYETTE COUNTY.																
Caperton; Sewell mine, Sewell bed (room turned off at face of drainage entry of Sugar Camp No. 3 workings, 4-foot bed and cut).	345	14344	A	1	3.99	24.29	68.78	2.94	.56	....	....	....	....	3.0	8,050	14,490
				2	....	25.30	71.64	3.06	.58	....	....	....	....	....	8,385	15,093
				3	....	26.10	73.90	....	.60	....	....	....	....	....	8,650	15,570
Same (at last break-through in third right air course of Sugar Camp No. 1 workings, 2-foot 9-inch bed, 2-foot 5-inch cut).	345	14345	A	1	4.70	23.81	69.63	1.86	.50	....	....	....	....	3.7	8,114	14,606
				2	....	24.98	73.07	1.95	.52	....	....	....	....	....	8,514	15,325
				3	....	25.48	74.52	....	.53	....	....	....	....	....	8,683	15,629
Same (face of right entry 13, Old Hill workings, 5,000 feet northeast of mine mouth, 4-foot 2½-inch bed, 4-foot cut).	345	14346	A	1	3.28	24.37	70.44	1.91	.53	....	....	....	....	2.2	8,218	14,792
				2	....	25.20	72.83	1.97	.55	....	....	....	....	....	8,497	15,295
				3	....	25.71	74.29	....	.56	....	....	....	....	....	8,668	15,602
Same (composite of samples 14344, 14345, and 14346).	346	14347	A	1	3.97	23.84	69.97	2.22	.56	5.08	83.37	1.62	7.15	3.0	8,116	14,609
				2	....	24.82	72.87	2.31	.58	4.83	86.81	1.69	3.78	....	8,451	15,212
				3	....	25.41	74.59	....	.59	4.94	88.86	1.73	3.88	....	8,650	15,570
Same (pillar at last room off cross entry 11, Old Hill workings, 3-foot 8½-inch bed and cut).	345	14348	A	1	3.49	23.75	70.09	2.67	.57	....	....	....	....	2.5	8,137	14,647
				2	....	24.61	72.62	2.77	.59	....	....	....	....	....	8,432	15,178
				3	....	25.31	74.69	....	.61	....	....	....	....	....	8,672	15,610
Same (on pillar of right entry 2, Sugar Camp No. 2 workings, 4-foot 1-inch bed, 3-foot 11-inch cut).	345	14349	A	1	2.89	25.47	69.28	2.36	.53	....	....	....	....	2.0	8,235	14,823
				2	....	26.23	71.34	2.43	.55	....	....	....	....	....	8,480	15,264
				3	....	26.88	73.12	....	.56	....	....	....	....	....	8,691	15,644
Same (composite of samples 14348 and 14349).....	346	14350	A	1	3.22	24.54	69.70	2.54	.55	5.16	83.80	1.67	6.28	2.3	8,170	14,706
				2	....	25.36	72.02	2.62	.57	4.96	86.59	1.73	3.53	....	8,442	15,196
				3	....	26.04	73.96	....	.59	5.09	88.92	1.78	3.62	....	8,669	15,604
Dunloop; Dunn Loop No. 2 mine, Sewell bed (at last break-through to the right on main entry 5; 500 feet northeast of mouth of main entry 5; 5-foot 8½-inch bed and cut).	346	14383	A	1	3.75	22.11	71.95	2.19	.58	....	....	....	....	2.7	....	....
				2	....	22.97	74.75	2.28	.60	....	....	....	....	....	....	....
				3	....	23.51	76.49	....	.61	....	....	....	....	....	....	....
Same (face of room 20, off right entry 8, off main entry 4; 1,600 feet southeast of mouth of main entry 4; 5-foot 7-inch bed and cut.)	346	14384	A	1	3.62	21.01	72.16	3.21	.80	....	....	....	....	2.7	....	....
				2	....	21.80	74.87	3.33	.83	....	....	....	....	....	....	....
				3	....	22.55	77.45	....	.86	....	....	....	....	....	....	....
Same (face of room 2, first crosscut off first left air course off main entry 6; 1,200 feet east of mouth of main entry 6; 5-foot 8-inch bed and cut).	346	14385	A	1	3.31	21.33	72.87	2.49	.44	....	....	....	....	2.3	....	....
				2	....	22.06	75.36	2.58	.46	....	....	....	....	....	....	....
				3	....	22.64	77.36	....	.47	....	....	....	....	....	....	....
Same (composite of samples 14383, 14384, and 14385).	346	14386	A	1	3.58	21.07	72.75	2.60	.64	5.01	84.38	1.65	5.72	2.6	8,195	14,751
				2	....	21.85	75.45	2.70	.66	4.78	87.51	1.71	2.64	....	8,499	15,298
				3	....	22.46	77.54	....	.68	4.91	89.93	1.76	2.72	....	8,734	15,721
Same (face of room 5, off the air course of main entry 3; 500 feet southeast of mouth of main entry 3; 5-foot 2-inch bed, 61-inch cut).	346	14387	A	1	3.95	20.61	72.94	2.50	.53	....	....	....	....	2.8	....	....
				2	....	21.46	75.94	2.60	.55	....	....	....	....	....	....	....
				3	....	22.03	77.97	....	.56	....	....	....	....	....	....	....
Same (pillar 17 of main entry 6; 1,040 feet southeast of mouth of main entry 6; 5-foot 10-inch bed and cut).	346	14388	A	1	3.54	20.45	73.02	2.99	.55	....	....	....	....	2.6	....	....
				2	....	21.20	75.70	3.10	.57	....	....	....	....	....	....	....
				3	....	21.88	78.12	....	.59	....	....	....	....	....	....	....
Same (composite of samples 14387 and 14388).....	346	14389	A	1	3.77	21.00	72.46	2.77	.55	4.91	84.06	1.66	6.05	2.7	8,157	14,683
				2	....	21.82	75.80	2.88	.57	4.67	87.36	1.73	2.79	....	8,477	15,259
				3	....	22.47	77.53	....	.59	4.81	89.95	1.78	2.87	....	8,729	15,712

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
FAYETTE COUNTY—continued.																
Elverton, ¼ mile west of; Elverton mine, Sewell bed (face of old Bayne entry, 3,900 feet southwest of No. 2 entrance, 3-foot 4½-inch bed and cut).	347	14338	A	1	2.66	26.10	68.70	2.54	.53	......	......	......	......	1.5	......	......
				2	......	26.81	70.58	2.61	.54	......	......	......	......	......	......	......
				3	......	27.53	72.47	......	.55	......	......	......	......	......	......	......
Same (corner of break-through, 10 feet from face of main entry, 3,900 feet southwest of No. 2 entrance, 3-foot 2⅝-inch bed and cut).	347	14339	A	1	3.18	24.47	70.23	2.12	.54	......	......	......	......	1.9	......	......
				2	......	25.27	72.54	2.19	.56	......	......	......	......	......	......	......
				3	......	25.84	74.16	......	.57	......	......	......	......	......	......	......
Same (composite of samples 14338 and 14339).....	347	14340	A	1	2.89	25.61	69.18	2.32	.55	4.99	84.11	1.64	6.39	1.7	8,220	14,796
				2	......	26.37	71.24	2.39	.57	4.81	86.62	1.69	3.92	......	8,465	15,237
				3	......	27.02	72.98	......	.58	4.93	88.74	1.73	4.02	......	8,672	15,610
Same (pillar of room 2, off left entry 1, off left entry 5, 3,100 feet south of No. 2 entrance, 4-foot 2⅝-inch bed and cut).	347	14341	A	1	2.73	25.61	69.36	2.30	.66	......	......	......	......	1.4	......	......
				2	......	26.33	71.31	2.36	.68	......	......	......	......	......	......	......
				3	......	26.97	73.03	......	.70	......	......	......	......	......	......	......
Same (pillar of room 3, off right entry 5, 3,100 feet southwest of No. 2 entrance, 4-foot 7½-inch bed and cut).	347	14342	A	1	3.26	24.59	69.65	2.50	.57	......	......	......	......	1.9	......	......
				2	......	25.42	72.00	2.58	.59	......	......	......	......	......	......	......
				3	......	26.09	73.91	......	.61	......	......	......	......	......	......	......
Same (composite of samples 14341 and 14342).....	347	14343	A	1	2.99	25.27	69.36	2.38	.65	5.06	83.68	1.70	6.53	1.7	8,203	14,765
				2	......	26.05	71.50	2.45	.67	4.88	86.26	1.75	3.99	......	8,456	15,221
				3	......	26.70	73.30	......	.69	5.00	88.43	1.79	4.09	......	8,668	15,602
Hawks Nest, about ½ mile north of; Mill Creek mine, Sewell bed (face of left entry 1, at room 49, 3-foot 8½-inch bed and cut).	348	14358	A	1	3.14	28.10	65.30	3.46	.56	......	......	......	......	2.1	......	......
				2	......	29.01	67.42	3.57	.58	......	......	......	......	......	......	......
				3	......	30.08	69.92	......	.60	......	......	......	......	......	......	......
Same (face of left entry 2, at neck of room 29, 3-foot bed and cut).	348	14359	A	1	2.60	28.75	63.81	4.84	.78	......	......	......	......	1.6	......	......
				2	......	29.52	65.51	4.97	.80	......	......	......	......	......	......	......
				3	......	31.06	68.94	......	.84	......	......	......	......	......	......	......
Same (face of third left air course, 2-foot 5½-inch bed and cut).	348	14360	A	1	2.66	28.10	65.66	3.58	.55	......	......	......	......	1.6	......	......
				2	......	28.87	67.45	3.68	.57	......	......	......	......	......	......	......
				3	......	29.97	70.03	......	.59	......	......	......	......	......	......	......
Same (face of right entry 5, 2-foot 8-inch bed and cut).	348	14361	A	1	3.31	27.74	65.27	3.68	.56	......	......	......	......	2.2	......	......
				2	......	28.69	67.50	3.81	.58	......	......	......	......	......	......	......
				3	......	29.83	70.17	......	.60	......	......	......	......	......	......	......
Same (composite of samples 14358-14361).........	348	14362	A	1	3.00	28.00	65.18	3.82	.61	5.11	81.65	1.59	7.22	1.9	8,013	14,423
				2	......	28.87	67.19	3.94	.63	4.93	84.17	1.64	4.69	......	8,261	14,870
				3	......	30.05	69.95	......	.66	5.13	87.62	1.71	4.88	......	8,600	15 480

Kay Moor, ¾ mile south of; Kay Moor No. 1 mine, Sewell bed (face of room 1, off left entry 3, off third main entry, 4,700 feet south of mine mouth, 3-foot 8½-inch bed and cut).	349	14294	A	1	3.00	25.22	68.28	3.50	.55					2.1		
				2		26.00	70.39	3.61	.57							
				3		26.98	73.02		.59							
Same (face of right entry 1½, off left entry 1, off Buffalo entry, 3,300 feet southeast of mine mouth, 3-foot 10¾-inch bed and cut).	349	14295	A	1	3.41	25.09	69.17	2.33	.60					2.4		
				2		25.98	71.61	2.41	.62							
				3		26.62	73.38		.64							
Same (face of left entry 1, off right entry 9, 3,300 feet west of mine mouth, 3-foot 6½-inch bed and cut).	349	14296	A	1	3.11	26.03	67.32	3.54	.60					2.1		
				2		26.87	69.48	3.65	.62							
				3		27.89	72.11		.64							
Same (face of west entry 11, 3,600 feet southwest of mine mouth, 3-foot 3-inch bed and cut).	349	14297	A	1	3.12	25.58	69.02	2.28	.54					2.2		
				2		26.40	71.25	2.35	.56							
				3		27.04	72.96		.57							
Same (composite of samples 14294–14297).........	349	14298	A	1	3.17	25.11	68.81	2.91	.52	5.09	82.59	1.63	7.26	2.2	8,102	14,584
				2		25.93	71.06	3.01	.54	4.89	85.29	1.68	4.59		8,367	15,061
				3		26.73	73.27		.56	5.04	87.93	1.73	4.74		8,626	15.527
Same (pillar between rooms 3 and 4, off right entry 1, off main entry 7, 3-foot 4-inch bed and cut).	349	14299	A	1	3.17	25.24	67.89	3.70	.51	4.97	82.05	1.67	7.10	2.2	8,055	14,499
				2		26.07	70.11	3.82	.53	4.77	84.73	1.72	4.43		8,318	14,972
				3		27.10	72.90		.55	4.96	88.09	1.79	4.61		8,648	15,566
south of; Kay Moor No. 2 mine, Sewell bed (face of left entry 3, off main entry, 3-foot bed and cut).	350	14401	A	1	2.79	26.25	68.09	2.87	.50					1.8		
				2		27.00	70.05	2.95	.51							
				3		27.82	72.18		.53							
Same (face of room 9, off left entry 2, 2-foot 11½-inch bed and cut).	350	14402	A	1	3.02	26.00	68.60	2.38	.50					1.9		
				2		26.81	70.74	2.45	.52							
				3		27.48	72.52		.53							
Same (face of room 9, off left entry 1; 150 feet from the entry, 3-foot ½-inch bed and cut).	350	14403	A	1	3.11	25.67	67.05	4.17	.56					2.0		
				2		26.49	69.21	4.30	.58							
				3		27.68	72.32		.61							
Same (composite of samples 14401, 14402, and 14403).	350	14404	A	1	3.03	25.73	68.11	3.13	.57	5.14	82.73	1.65	6.78	1.9	8,114	14,605
				2		26.53	70.24	3.23	.59	4.95	85.31	1.70	4.22		8,367	15,061
				3		27.42	72.58		.61	5.12	88.16	1.76	4.35		8,646	15,563
South Nuttal; Brown mine, Sewell bed (face of room 2, off Hill entry, 3-foot 6-inch bed, 2-foot 10-inch cut).	350	10618	A	1	3.63	23.70	70.10	2.57	.70					2.8		
				2		24.59	72.74	2.67	.73							
				3		25.26	74.74		.75							
Same (rib of Lockett heading, between rooms 1 and 2, 2,000 feet from entry mouth, 3-foot 9¾-inch bed and cut).	350	10619	A	1	3.36	24.88	68.62	3.14	.68					2.5		
				2		25.75	71.00	3.25	.70							
				3		26.62	73.38		.72							
Same (neck of room 1, off right entry 6, off Wheeler entry, 3-foot 5-inch bed and cut).	350	10620	A	1	3.81	23.32	70.16	2.71	.52					3.0		
				2		24.24	72.94	2.82	.54							
				3		24.94	75.06		.56							
Same (neck of room 4, off left entry 2, off entry 18, 3-foot 6-inch bed and cut).	350	10621	A	1	3.45	23.91	70.39	2.25	.55					2.6		
				2		24.76	72.91	2.33	.57							
				3		25.35	74.65		.58							
Same (left rib near face of entry 18, 3-foot 5½-inch bed and cut).	350	10622	A	1	3.32	24.92	69.35	2.41	.51					2.5		
				2		25.77	71.74	2.49	.53							
				3		26.43	73.57		.54							
Same (composite of samples 10618–10622).........	351	10623	A	1	3.51	24.76	69.00	2.73	.58	5.17	83.06	1.57	6.89	2.7	8,146	14,663
				2		25.66	71.51	2.83	.60	4.95	86.08	1.63	3.91		8,443	15,197
				3		26.41	73.59		.62	5.09	88.58	1.68	4.03		8,689	15,640

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
FAYETTE COUNTY—continued.																
Thayer, Ephraims Creek (Buffalo) mine, Fire Creek bed (face of room 5, off right entry 5, off Slater entry, 3-foot 6-inch bed and cut).	351	10629	A	1	3.17	18.46	70.86	7.51	1.07	4.84	79.07	1.56	5.95	2.5	7,775	13,995
				2	....	19.06	73.18	7.76	1.10	4.64	81.66	1.61	3.23	....	8,029	14,452
				3	....	20.66	79.34	....	1.19	5.03	88.53	1.75	3.50	....	8,704	15,667
GREENBRIER COUNTY.																
Richwood, 9 miles southeast of; Spruce Knob mine, Sewell bed (in mine, 1,500 feet southwest of mine mouth; 4-foot 10-inch bed; 4-foot 1-inch cut).	352	13547	B	1	3.91	26.20	64.09	5.80	.62	5.00	79.33	1.59	7.66	3.1	7,810	14,058
				3	....	27.27	66.69	6.04	.65	4.76	82.56	1.65	4.34	....	8,128	14,630
				3	....	29.02	70.98	....	.69	5.07	87.87	1.76	4.61	....	8,651	15,572
M'DOWELL COUNTY.																
Algoma, 1 mile east of; Piney mine, Pocahontas No. 3 bed (face of main entry, 6-foot 1-inch bed, 5-foot 10-inch cut).	352	14154	A	1	2.50	17.84	74.30	5.36	.52	....	....	....	....	1.8	....	....
				2	....	18.30	76.20	5.50	.53	....	....	....	....	....	....	....
				3	....	19.37	80.63	....	.56	....	....	....	....	....	....	....
Same (face of room 14, off cross entry 10, 5-foot 3-inch bed, 5-foot 1½-inch cut).	352	14155	A	1	2.60	18.59	74.19	4.62	.47	....	....	....	....	1.9	....	....
				2	....	19.09	76.17	4.74	.48	....	....	....	....	....	....	....
				3	....	20.04	79.96	....	.50	....	....	....	....	....	....	....
Same (face of room 30, off cross entry 8, 5-foot 6-inch bed, 5-foot 3-inch cut).	352	14156	A	1	2.70	17.87	75.12	4.31	.44	....	....	....	....	2.0	....	....
				2	....	18.36	77.21	4.43	.45	....	....	....	....	....	....	....
				3	....	19.21	80.79	....	.47	....	....	....	....	....	....	....
Same (composite of samples 14154, 14155, and 14156)	352	14157	A	1	2.58	18.10	74.58	4.74	.47	4.52	84.25	1.17	4.85	1.9	8,086	14,555
				2	....	18.58	76.55	4.87	.48	4.34	86.48	1.20	2.63	....	8,300	14,940
				3	....	19.53	80.47	....	.50	4.56	90.91	1.26	2.77	....	8,725	15,705
Same (pillar of room 15, off cross entry 5, 6-foot 7-inch bed, 6-foot 4-inch cut).	352	14158	A	1	2.80	17.61	74.41	5.18	.64	....	....	....	....	2.1	....	....
				2	....	18.12	76.55	5.33	.66	....	....	....	....	....	....	....
				3	....	19.14	80.86	....	.70	....	....	....	....	....	....	....
Same (rib of room 33, off cross entry 7, 5-foot 6-inch bed, 5-foot 3-inch cut).	352	14159	A	1	2.63	18.09	74.74	4.54	.47	....	....	....	....	1.9	....	....
				2	....	18.58	76.76	4.66	.48	....	....	....	....	....	....	....
				3	....	19.49	80.51	....	.50	....	....	....	....	....	....	....
Same (pillar of room 15, off cross entry 6, 6-foot 4-inch bed, 6-foot ½-inch cut).	352	14160	A	1	2.39	17.71	75.61	4.29	.69	....	....	....	....	1.7	....	....
				2	....	18.14	77.46	4.40	.71	....	....	....	....	....	....	....
				3	....	18.97	81.03	....	.74	....	....	....	....	....	....	....

Same (pillar of room 8, off cross entry 3, 6-foot 7-inch bed, 6-foot 4-inch cut).	352	14161	A	1	2.75	17.37	75.51	4.37	.59					2.1		
				2		17.86	77.65	4.49	.61							
				3		18.70	81.30		.64							
Same (composite of samples 14158-14161)	353	14162	A	1	2.03	17.68	75.13	4.56	.63	4.44	84.72	1.22	4.43	2.0	8,096	14,573
				2		18.16	77.16	4.68	.65	4.26	87.01	1.25	2.15		8,315	14,967
				3		19.05	80.95		.68	4.47	91.28	1.31	2.26		8,723	15,701
Big Four; Cirrus No. 2 mine, Pocahontas No 3 bed (at the break-through near face of main heading, 2,850 feet south of slope mouth, 5-foot 7-inch bed, 5-foot ½-inch cut).	353	14218	A	1	2.31	17.05	76.18	4.46	.58					1.7		
				2		17.45	77.98	4.57	.59							
				3		18.29	81.71		.62							
Same (pocket 30 feet from face of south entry 6, 1,500 feet southwest of slope mouth, 5-foot 5½-inch bed, 5-foot 2½-inch cut).	353	14219	A	1	2.28	17.72	75.29	4.71	.61					1.6		
				2		18.13	77.05	4.82	.62							
				3		19.05	80.95		.65							
Same (face of room 4, off south heading 3; 2,850 feet south of slope mouth, 5-foot 8-inch bed, 5-foot 2½-inch cut).	353	14220	A	1	2.54	17.53	75.71	4.22	.70					1.9		
				2		17.99	77.68	4.33	.72							
				3		18.80	81.20		.75							
Same (face of east heading 6, 2,100 feet south of slope mouth, 5-foot 2½-inch bed, 4-foot 9-inch cut.)	353	14221	A	1	2.27	17.32	75.98	4.43	.58					1.6		
				2		17.72	77.75	4.53	.59							
				3		18.56	81.44		.62							
Same (composite of samples 14218-14221)	353	14222	A	1	2.30	16.98	76.21	4.51	.66	4.36	85.00	1.20	4.27	1.7	8,131	14,636
				2		17.38	78.00	4.62	.68	4.20	87.00	1.23	2.27		8,322	14,980
				3		18.22	81.78		.71	4.40	91.21	1.29	2.39		8,725	15,705
Same; Pocahontas No. 4 bed (face of north entry 250 feet northwest of slope mouth; 4-foot 7½-inch bed, 4-foot 2½-inch cut).	353	14223	A	1	3.10	17.91	75.26	3.73	.55	4.50	84.02	1.17	6.03	2.5	8,180	14,724
				2		18.48	77.67	3.85	.57	4.29	86.71	1.21	3.37		8,442	15,196
				3		19.22	80.78		.59	4.46	90.18	1.26	3.51		8,780	15,804
Huger; Middle States mine, Pocahontas No. 3 bed (pillar between rooms 3 and 6, off north entry; 4-foot 7½-inch bed, 4-foot 6-inch cut).	354	14421	A	1	2.89	17.42	71.16	8.53	.83					2.2		
				2		17.94	73.28	8.78	.85							
				3		19.67	80.33		.93							
Same, Pocahontas No. 4 bed (face of room 6, off second rock heading; 5-foot 1½-inch bed, 4-foot 10½-inch cut).	354	14416	A	1	1.28	18.13	73.57	7.02	.62					.5		
				2		18.37	74.52	7.11	.63							
				3		19.78	80.22		.68							
Same (face of second right air course; 5-foot 5-inch bed, 5-foot cut).	354	14417	A	1	1.96	18.09	74.76	5.19	.58					1.2		
				2		18.45	76.26	5.29	.59							
				3		19.48	80.52		.62							
Same (rib of last break-through on main diagonal entry; 5-foot 9½-inch bed, 5-foot 5-inch cut).	354	14418	A	1	1.64	17.65	75.89	4.82	.64					1.0		
				2		17.94	77.16	4.90	.65							
				3		18.86	81.14		.68							
Same (face of room 4 off first west parallel heading; 5-foot 6½-inch bed, 5-foot 5-inch cut).	354	14419	A	1	1.53	18.14	73.46	6.87	.59					.9		
				2		18.42	74.60	6.98	.60							
				3		19.80	80.20		.64							
Same (composite of samples 14416-14419)	354	14420	A	1	1.55	18.00	74.52	5.93	.61	4.38	83.70	1.32	4.06	.9	8,060	14,508
				2		18.28	75.70	6.02	.62	4.28	85.01	1.34	2.73		8,187	14,737
				3		19.45	80.55		.66	4.55	90.46	1.43	2.90		8,712	15,682
Same (pillar between rooms 8 and 9 off left entry 4 off main west heading; 4-foot 1-inch bed, 3-foot 11½-inch cut).	354	14422	A	1	1.64	17.87	75.02	5.47	.60					.9		
				2		18.17	76.27	5.56	.61							
				3		19.24	80.76		.65							
Same (pillar between rooms 20 and 21 off second parallel entry off main diagonal heading; 5-foot 7½-inch bed, 5-foot 4-inch cut).	354	14423	A	1	1.58	18.03	74.12	6.27	.70					.9		
				2		18.32	75.31	6.37	.71							
				3		19.57	80.43		.76							
Same (composite of samples 14421, 14422, and 14423).	354	14424	A	1	1.98	17.80	73.47	6.75	.71	4.49	82.64	1.20	4.21	1.3	7,958	14,324
				2		18.16	74.95	6.89	.72	4.36	84.31	1.22	2.50		8,119	14,614
				3		19.50	80.50		.77	4.68	90.55	1.31	2.69		8,720	15,696

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
M'DOWELL COUNTY—continued.																
Jed; Havaco (formerly Jed) mine, Pocahontas No. 3 bed (face of first butt left entry; 5-foot 1¼-inch bed, 4-foot 9½-inch cut).	355	13775	A	1	2.08	16.39	76.60	4.93	.90	4.20	84.87	1.16	3.94	1.5	8,100	14,580
				2		16.74	78.23	5.03	.92	4.05	86.67	1.18	2.15		8,272	14,890
				3		17.63	82.37		.97	4.26	91.26	1.24	2.27		8,710	15,678
Jenkinjones; Pocahontas No. 6 mine, Pocahontas No. 3 bed (face of main heading, 250 feet west of mine mouth; 9-foot 6-inch bed, 8-foot 11½-inch cut).	356	14185	A	1	3.01	22.04	69.46	5.49	.49					2.4		
				2		22.72	71.62	5.66	.51							
				3		24.08	75.92		.54							
Same (face of aircourse, 250 feet from mine mouth; 10-foot 2½-inch bed, 9-foot 2½-inch cut).	356	14186	A	1	3.66	21.78	70.64	3.92	.52					3.1		
				2		22.61	73.32	4.07	.54							
				3		23.57	76.43		.56							
Same, (composite of samples 14185, 14186, and 14187).	356	14188	A	1	3.36	22.50	68.86.	5.28	.52	4.83	82.51	1.22	5.64	2.7	7,963	14,369
				2		23.28	71.26	5.46	.54	4.62	85.38	1.26	2.74		8,261	14,870
				3		24.63	75.37		.57	4.89	90.31	1.33	2.90		8,738	15,729
¼-mile south of; Pocahontas No. 6 B mine, Pocahontas No. 3 bed (face of main heading, 500 feet from mine mouth; 10-foot 6½-inch bed, 9-foot 5-foot cut).	356	14187	A	1	3.23	22.45	67.83	6.49	.48					2.6		
				2		23.20	70.09	6.71	.50							
				3		24.87	75.13		.54							
Same (face of first right cross heading, 350 feet from mine mouth, 9-foot 4-inch bed, 8-foot 10-inch cut).	356	14189	A	1	3.27	22.88	69.36	4.49	.44					2.7	8,021	14,438
				2		23.65	71.71	4.64	.45						8,292	14,926
				3		24.80	75.20		.47						8,696	15,653
½-mile south of; Pocahontas No. 7 A mine, Pocahontas No. 3 bed (face of main heading, 120 feet from mine mouth, 7-foot 10-inch bed, 7-foot 1½-inch cut).	356	14190	A	1	4.09	22.07	69.05	4.79	.47					3.1	7,920	14,256
				2		23.01	72.00	4.99	.49						8,257	14,863
				3		24.22	75.78		.52						8,690	15,642
Leckie, ½ mile east of; Leckie No. 1 mine, Pocahontas No. 3 bed (face of room 1, off Monkey heading, 1,700 feet west of mine mouth, 8-foot 2¼-inch bed, 7-foot 10-inch cut).	357	14271	A	1	3.53	17.95	75.48	3.04	.53					2.9		
				2		18.61	78.24	3.15	.55							
				3		19.21	80.79		.57							
Same (face of Monkey heading, 1,900 feet west of mine mouth, 8-foot 1½-inch bed, 7-foot 6½-inch cut).	357	14272	A	1	3.39	18.56	74.81	3.24	.52					2.7		
				2		19.21	77.44	3.35	.54							
				3		19.88	80.12		.56							
Same (face of Kick-back heading, 1,250 feet northwest of mine mouth, 7-foot 10½-inch bed, 7-foot 7½-inch cut).	357	14273	A	1	4.38	18.12	73.92	3.58	.52					3.3		
				2		18.95	77.31	3.74	.54							
				3		19.69	80.31		.56							

Same (on chain pillar of right heading 1, 650 feet north of mine mouth, 7-foot 3-inch bed, 6-foot 10½-inch cut).	357	14274	A	1	4.03	18.50	74.46	3.01	.62	......	......	......	......	3.3	......	......
				2	......	19.28	77.58	3.14	.65	......	......	......	......	......	......	......
				3	......	19.90	80.10	......	.67	......	......	......	......	......	......	......
Same (pillar at mouth of room 1, off right heading 2, 950 feet northwest of mine mouth, 7-foot 4½-inch bed, 6-foot 11¼-inch cut).	357	14275	A	1	3.68	18.51	74.75	3.06	.55	......	......	......	......	2.7	8,159	14,686
				2	......	19.22	77.60	3.18	.57	......	......	......	......	......	8,471	15,248
				3	......	19.85	80.15	......	.59	......	......	......	......	......	8,749	15,748
Same (composite of samples 14271–14275)..........	357	14276	A	1	3.81	18.27	74.70	3.22	.54	4.73	84.48	1.22	5.81	3.0	8,161	14,690
				2	......	18.99	77.66	3.35	.56	4.48	87.83	1.27	2.51	......	8,484	15,271
				3	......	19.65	80.35	......	.58	4.64	90.88	1.31	2.59	......	8,778	15,800
¼-mile east of: Leckie No. 2 mine, Pocahontas No. 3 bed (face of room 17, off first right air course, 100 feet from outcrop, 7-foot 9-inch bed, 7-foot 4-inch cut).	357	14277	A	1	5.66	18.58	71.96	3.80	.48	......	......	......	......	4.4	......	......
				2	......	19.69	76.28	4.03	.51	......	......	......	......	......	......	......
				3	......	20.52	79.48	......	.53	......	......	......	......	......	......	......
Same (working face, 7-foot 6-inch bed, 7-foot 2-inch cut).	357	14278	A	1	3.17	18.83	75.05	2.95	.57	......	......	......	......	2.4	......	......
				2	......	19.45	77.50	3.05	.59	......	......	......	......	......	......	......
				3	......	20.06	79.94	......	.61	......	......	......	......	......	......	......
Same (face of main straight entry, 600 feet southeast of mine mouth, 8-foot bed, 7-foot 6-inch cut).	357	14279	A	1	4.16	18.65	74.52	2.67	.53	......	......	......	......	3.3	......	......
				2	......	19.46	77.75	2.79	.55	......	......	......	......	......	......	......
				3	......	20.02	79.98	......	.57	......	......	......	......	......	......	......
Same (face of room 5, off diagonal entry, 7-foot 11-inch bed, 7-foot 6-inch cut).	357	14280	A	1	3.75	18.10	75.03	3.12	.53	......	......	......	......	2.9	......	......
				2	......	18.81	77.95	3.24	.55	......	......	......	......	......	......	......
				3	......	19.44	80.56	......	.57	......	......	......	......	......	......	......
Same (face of left entry 1, 1,300 feet from mine mouth, 8-foot 4-inch bed, 7-foot 9½-inch cut).	357	14281	A	1	4.34	18.07	73.93	3.66	.53	......	......	......	......	3.5	......	......
				2	......	18.89	77.28	3.83	.55	......	......	......	......	......	......	......
				3	......	19.64	80.36	......	.57	......	......	......	......	......	......	......
Same (composite of samples 14277–14281)..........	358	14282	A	1	4.24	18.60	73.94	3.22	.52	4.76	83.32	1.22	6.96	3.3	8,127	14,629
				2	......	19.42	77.22	3.36	.54	4.48	87.01	1.27	3.34	......	8,487	15,277
				3	......	20.10	79.90	......	.56	4.64	90.04	1.31	3.45	......	8,782	15,808
Roderfield, 1¾ miles east of; Davy-Pocahontas No. 1 mine, War Creek bed (face of room 3, off entry 2, 6-foot 1-inch bed, 5-foot 5-inch cut).	358	14266	A	1	1.90	16.75	69.58	11.77	1.24	......	......	......	......	1.2	......	......
				2	......	17.07	70.93	12.00	1.26	......	......	......	......	......	......	......
				3	......	19.40	80.60	......	1.43	......	......	......	......	......	......	......
Same (face of entry 2, opposite room 9, 6-foot 10¾-inch bed, 6-foot 1¼-inch cut).	358	14267	A	1	2.06	17.72	68.95	11.27	2.32	......	......	......	......	1.4	......	......
				2	......	18.09	70.40	11.51	2.37	......	......	......	......	......	......	......
				3	......	20.44	79.56	......	2.68	......	......	......	......	......	......	......
Same (face of entry 7, at mouth of room 15, 5-foot 8¼-inch bed, 4-foot 10¾-inch cut).	358	14268	A	1	2.83	16.85	70.20	10.12	1.77	......	......	......	......	2.1	......	......
				2	......	17.34	72.25	10.41	1.82	......	......	......	......	......	......	......
				3	......	19.35	80.65	......	2.03	......	......	......	......	......	......	......
Same (composite of samples 14266, 14267, and 14268).	358	14269	A	1	2.32	16.76	69.80	11.12	1.78	4.35	77.46	1.27	4.02	1.6	7,508	13,514
				2	......	17.16	71.46	11.38	1.82	4.19	79.30	1.30	2.01	......	7,687	13,837
				3	......	19.36	80.64	......	2.05	4.73	89.48	1.47	2.27	......	8,674	15,613
2 miles east of; Davy-Pocahontas No. 2 mine, Davy bed (face of room 5, off left entry 1, 500 feet southwest of mine mouth, 3-foot 8½-inch bed, 3-foot ½-inch cut).	359	14262	A	1	2.81	18.77	73.60	4.82	1.07	......	......	......	......	1.9	......	......
				2	......	19.31	75.73	4.96	1.10	......	......	......	......	......	......	......
				3	......	20.32	79.68	......	1.16	......	......	......	......	......	......	......
Same (face of first left air course, 600 feet southwest of mine mouth, 3-foot 11¾-inch bed, 3-foot 4¼-inch cut).	359	14263	A	1	3.27	18.90	71.87	5.96	.69	......	......	......	......	2.4	......	......
				2	......	19.54	74.30	6.16	.71	......	......	......	......	......	......	......
				3	......	20.82	79.18	......	.76	......	......	......	......	......	......	......
Same (face of main heading, 600 feet west of mine mouth, 3-foot 6⅞-inch bed, 2-foot 10½-inch cut).	359	14264	A	1	2.86	18.79	70.41	7.94	.84	......	......	......	......	1.7	......	......
				2	......	19.34	72.49	8.17	.86	......	......	......	......	......	......	......
				3	......	21.06	78.94	......	.94	......	......	......	......	......	......	......
Same (composite of samples 14262, 14263, and 14264).	359	14265	A	1	2.95	19.08	71.76	6.21	.86	4.57	81.53	1.38	5.45	2.0	7,903	14,225
				2	......	19.66	73.94	6.40	.89	4.37	84.01	1.42	2.91	......	8,143	14,657
				3	......	21.00	79.00	......	.95	4.67	89.76	1.52	3.10	......	8,700	15,660

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
M'DOWELL COUNTY—continued.																
Rolfe, 3 miles from North Fork; Rolfe mine, Pocahontas No. 3 bed (face of B-2 butt entry off B entry, 5,800 feet northwest of mine mouth, 5-foot 3-inch bed, 4-foot 11-inch cut).	360	14394	A	1	2.63	18.08	73.07	6.22	.48					1.8		
				2		18.57	75.04	6.39	.49							
				3		19.84	80.16		.52							
Same (face of B-1 butt entry, off B entry, 6,250 feet northwest of mine mouth, 5-foot 1-inch bed, 4-foot 10-inch cut).	360	14395	A	1	2.77	17.77	74.55	4.91	.52					2.0		
				2		18.28	76.67	5.05	.53							
				3		19.25	80.75		.56							
Same (face of left boundary heading, 6,200 feet northwest of mine mouth, 5-foot 2-inch bed, 4-foot 10-inch cut).	360	14396	A	1	2.76	17.99	74.20	5.05	.48					2.0		
				2		18.50	76.31	5.19	.49							
				3		19.51	80.49		.52							
Same (face of room 15, off 3-A entry, off right entry 1, 5,100 feet northeast of mine mouth, 5-foot ½-inch bed, 4-foot 6½-inch cut).	360	14397	A	1	2.65	17.91	76.10	3.34	.50					1.9		
				2		18.40	78.17	3.43	.51							
				3		19.05	80.95		.53							
Same (face of room 18, off 2-A entry, off right entry 2, 5,000 feet northeast of mine mouth, 5-foot 1½-inch bed, 4-foot 7-inch cut).	360	14398	A	1	3.10	17.56	75.76	3.58	.48					2.2		
				2		18.12	78.19	3.69	.50							
				3		18.81	81.19		.52							
Same (composite of samples 14394-14398)	360	14399	A	1	2.82	17.67	74.95	4.56	.49	4.58	83.92	1.23	5.22	2.0	8,096	14,573
				2		18.18	77.13	4.69	.50	4.39	86.35	1.27	2.80		8,331	14,996
				3		19.07	80.93		.52	4.61	90.60	1.33	2.94		8,741	15,734
Same (pillar of room 4, off right entry 1, 4,200 feet east of mine mouth, 4-foot 10-inch bed, 4-foot 4-inch cut).	360	14400	A	1	3.03	18.06	76.22	2.69	.55					2.3	8,259	14,866
				2		18.62	78.61	2.77	.57						8,517	15,331
				3		19.15	80.85		.59						8,760	15,768
Susana station (Yukon post office), ¼ mile west of; Yukon-Pocahontas mine, War Creek bed (face of room 24, off main air course, 1,700 feet southwest of main opening, 4-foot ½-inch bed, 3-foot 8½-inch cut).	361	14426	A	1	1.84	16.89	72.87	8.40	.55					.8		
				2		17.21	74.23	8.56	.56							
				3		18.82	81.18		.61							
Same (face of main entry, 1,850 feet southwest of main opening, 5-foot 4½-inch bed, 3-foot 8-inch cut)	361	14427	A	1	2.11	17.01	73.39	7.49	.51					1.1		
				2		17.38	74.97	7.65	.52							
				3		18.82	81.18		.56							

Same (face of room 7, off main air course, 850 feet southwest of main opening, 5-foot 1-inch bed, 4-foot 1-inch cut).	361	14428	A	1	2.53	16.81	72.45	8.21	.63	......	......	......	......	1.5	......	......
				2	......	17.25	74.33	8.42	.65	......	......	......	......	......	......	......
				3	......	18.84	81.16	......	.71	......	......	......	......	......	......	......
Same (face of room 14, off main course, 1,100 feet southwest of main opening, 5-foot 3-inch bed, 3-foot 11½-inch cut).	361	14429	A	1	4.09	16.84	70.08	8.99	.51	......	......	......	......	3.2	......	......
				2	......	17.56	73.07	9.37	.53	......	......	......	......	......	......	......
				3	......	19.38	80.62	......	.58	......	......	......	......	......	......	......
Same (composite of samples 14426-14429)........	361	14430	A	1	2.63	17.19	71.98	8.20	.57	4.44	80.62	1.45	4.72	1.7	7,781	14,006
				2	......	17.65	73.93	8.42	.59	4.26	82.80	1.49	2.44	......	7,991	14,384
				3	......	19.27	80.73	......	.64	4.65	90.41	1.63	2.67	......	8,725	15,705
Same (pillar at last break-through, off third right crosscut, off main entry, 1,200 feet northwest of old opening, 5-foot 9-inch bed, 51½-inch cut).	361	14425	A	1	2.04	16.83	73.05	8.08	.58	......	......	......	......	1.3	7,790	14,022
				2	......	17.18	74.57	8.25	.59	......	......	......	......	......	7,952	14,314
				3	......	18.72	81.28	......	.64	......	......	......	......	......	8,667	15,601
Maybeury; Norfolk mine, Pocahontas No. 3 bed........	361	11247	C	1	2.21	15.78	71.65	10.36	.55	4.08	79.11	1.04	4.86	1.6	7 642	13,756
				2	......	16.14	73.27	10.59	.56	3.92	80.90	1.06	2.97	......	7,815	14,067
				3	......	18.05	81.95	......	.63	4.38	90.48	1.19	3.32	......	8,740	15,732
Same..	361	11780	C	1	2.97	15.87	71.39	9.77	.52	4.33	79.41	.99	4.98	2.5	7,649	13,768
				2	......	16.36	73.57	10.07	.54	4.12	81.84	1.02	2.41	......	7,883	14,189
				3	......	18.19	81.81	......	.60	4.58	91.01	1.13	2.68	......	8,766	15,779
War, 1 mile south of; John's Branch mine, War Creek bed (face of main entry, 800 feet from mine mouth, 3-foot 4-inch bed, 2-foot 9-inch cut).	362	14468	A	1	2.43	16.96	70.87	9.74	.65	......	......	......	......	1.6	......	......
				2	......	17.38	72.64	9.98	.67	......	......	......	......	......	......	......
				3	......	19.31	80.69	......	.74	......	......	......	......	......	......	......
Same (face of heading 3, off right entry, 3-foot 10-inch bed and cut).	362	14469	A	1	2.00	17.22	71.43	9.35	.57	......	......	......	......	1.2	......	......
				2	......	17.57	72.89	9.54	.58	......	......	......	......	......	......	......
				3	......	19.42	80.58	......	.64	......	......	......	......	......	......	......
Same (face of room 4, off first left air course, 4-foot 4-inch bed, 3-foot 8-inch cut).	362	14470	A	1	2.41	16.94	73.18	7.47	.58	......	......	......	......	1.6	......	......
				2	......	17.36	74.99	7.65	.59	......	......	......	......	......	......	......
				3	......	18.80	81.20	......	.64	......	......	......	......	......	......	......
Same (face of first right butt entry, off left entry 2, 4-foot 4-inch bed, 3-foot 7-inch cut).	362	14471	A	1	2.31	17.10	72.01	8.58	.66	......	......	......	......	1.5	......	......
				2	......	17.50	73.72	8.78	.68	......	......	......	......	......	......	......
				3	......	19.19	80.81	......	.75	......	......	......	......	......	......	......
Same (composite of samples 14468-14471)........	362	14472	A	1	2.24	17.29	71.59	8.88	.64	4.34	80.36	1.29	4.49	1.5	7,751	13,952
				2	......	17.69	73.23	9.08	.65	4.18	82.20	1.32	2.57	......	7,928	14,270
				3	......	19.46	80.54	......	.71	4.60	90.41	1.45	2.83	......	8,720	15,696
War Creek mine, War Creek bed (face of fourth left cross entry, off main entry, 3-foot 4-inch bed and cut).	363	14405	A	1	2.74	16.26	73.06	7.94	.69	......	......	......	......	1.9	......	......
				2	......	16.72	75.12	8.16	.71	......	......	......	......	......	......	......
				3	......	18.20	81.80	......	.77	......	......	......	......	......	......	......
Same (face of main entry, 1,200 feet from mine mouth, 3-foot 11-inch bed and cut).	363	14406	A	1	1.86	16.98	72.05	9.11	.76	......	......	......	......	1.1	......	......
				2	......	17.30	73.42	9.28	.77	......	......	......	......	......	......	......
				3	......	19.07	80.93	......	.85	......	......	......	......	......	......	......
Same (face of room 4. off main air course, about 1,000 feet from mine mouth, 3-foot bed and cut).	363	14407	A	1	1.97	17.03	72.74	8.26	.67	......	......	......	......	1.2	......	......
				2	......	17.37	74.20	8.43	.68	......	......	......	......	......	......	......
				3	......	18.97	81.03	......	.74	......	......	......	......	......	......	......
Same (composite of samples 14405, 14406, and 14407).	363	14408	A	1	2.18	16.36	73.02	8.44	.69	4.20	80.70	1.44	4.53	1.4	7,781	14,006
				2	......	16.72	74.65	8.63	.71	4.05	82.50	1.47	2.64	......	7,965	14,319
				3	......	18.30	81.70	......	.78	4.43	90.30	1.61	2.88	......	8,707	15,673

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
M'DOWELL COUNTY—continued.																
Welch, 2 miles west of; Hemphill Township; Oregon No. 2 mine, Pocahontas No. 4 bed (face of main entry 4,000 feet south of mine mouth, 3-foot 1-inch bed and cut).	363	14215	A	1	2.41	18.72	71.47	7.40	.52	......	......	......	......	1.7	......	......
				2	......	19.18	73.24	7.58	.53	......	......	......	......	......	......	......
				3	......	20.75	79.25	......	.57	......	......	......	......	......	......	......
Same (rib of room 11, off air course, 100 feet from face, and 3,500 feet southeast of mine mouth, 3-foot 4 inch bed and cut).	363	14216	A	1	2.22	18.08	72.41	7.29	.53	......	......	......	......	1.6	......	......
				2	......	18.49	74.05	7.46	.54	......	......	......	......	......	......	......
				3	......	19.98	80.02	......	.58	......	......	......	......	......	......	......
2 miles west of; Hemphill Township; Oregon No. 3 mine, Pocahontas No. 4 bed (face of main entry, 3,000 feet southwest from mine mouth; 3-foot 7-inch bed and cut).	363	14212	A	1	2.54	18.41	70.16	8.89	.63	......	......	......	......	1.8	......	......
				2	......	18.89	71.99	9.12	.65	......	......	......	......	......	......	......
				3	......	20.79	79.21	......	.72	......	......	......	......	......	......	......
Same (face of room 16, off left entry 4, 2,000 feet south of mine mouth, 3-foot 5-inch bed and cut).	363	14213	A	1	2.60	18.20	72.13	7.07	.67	......	......	......	......	1.9	......	......
				2	......	18.69	74.05	7.26	.69	......	......	......	......	......	......	......
				3	......	20.15	79.85	......	.74	......	......	......	......	......	......	......
Same (pillar of room 1, off left entry 1, 800 feet southeast of mine mouth, 4-foot 1-inch bed and cut).	363	14214	A	1	3.85	18.68	70.38	7.09	.52	......	......	......	......	2.8	......	......
				2	......	19.43	73.20	7.37	.54	......	......	......	......	......	......	......
				3	......	20.98	79.02	......	.58	......	......	......	......	......	......	......
Same (composite of samples 14212-14216—includes samples from No. 2 mine).	364	14217	A	1	2.76	18.54	71.19	7.51	.59	4.40	80.64	1.43	5.43	2.0	7,790	14,022
				2	......	19.07	73.21	7.72	.61	4.21	82.93	1.47	3.06	......	8,011	14,420
				3	......	20.67	79.33	......	.66	4.56	89.87	1.59	3.32	......	8,682	15,628
2 miles west of; Standard Pocahontas mine, Pocahontas No. 3 bed (face of main entry in No. 2 shaft, 85 feet from foot of shaft, 5-foot ¾-inch bed, 4-foot 8¾-inch cut).	364	12512	A	1	3.68	15.34	73.72	7.26	.89	......	......	......	......	3.0	7,726	13,907
				2	......	15.93	76.53	7.54	.92	......	......	......	......	......	8,021	14,438
				3	......	17.23	82.77	......	.99	......	......	......	......	......	8,675	15,615
Same (face of entry driven from No. 1 shaft to connect with No. 2 shaft, about 120 feet from bottom of shaft No. 2, 4-foot 11-inch bed, 4-foot 8¼-inch cut).	364	12513	A	1	2.36	15.48	75.16	7.00	1.00	......	......	......	......	1.8	7,894	14,209
				2	......	15.85	76.98	7.17	1.02	......	......	......	......	......	8,085	14,553
				3	......	17.07	82.93	......	1.10	......	......	......	......	......	8,709	15,676

MARION COUNTY.															
Monongah, No. 63 (formerly No. 8) mine, Pittsburgh bed (face of room 14, off left entry 3, off north entry 2, 2,700 feet northwest of mouth of main slope, 7-foot ¾-inch bed, 6-foot 9¾-inch cut).	365	11429	A	1	2.65	35.46	56.26	5.63	.70	......	......	......	......	1.3	......	......
				2	......	36.42	57.80	5.78	.72	......	......	......	......	......	......	......
				3	......	38.65	61.35	......	.76	......	......	......	......	......	......	......
Same (face of room 14, off left entry 4, off north entry 2, 3,200 feet northwest of mouth of main slope, 7-foot 7¾-inch bed, 7-foot 5-inch cut).	365	11430	A	1	2.67	35.44	55.97	5.92	.73	......	......	......	......	1.3	......	......
				2	......	36.41	57.51	6.08	.75	......	......	......	......	......	......	......
				3	......	38.77	61.23	......	.80	......	......	......	......	......	......	......
Same (face of room 18, off left entry 2, off north entry 2, 2,650 feet from mouth of main slope, 7-foot bed, 6-foot 10-inch cut).	365	11431	A	1	3.67	35.28	55.05	6.00	.65	......	......	......	......	2.4	......	......
				2	......	36.62	57.15	6.23	.67	......	......	......	......	......	......	......
				3	......	39.05	60.96	......	.71	......	......	......	......	......	......	......
Same (face of room 30, off left entry 1, off north entry 2, 2,900 feet northwest from mouth of main slope, 7-foot 5½-inch bed, 7-foot 3½-inch cut).	365	11432	A	1	2.43	35.76	56.85	4.96	.64	......	......	......	......	1.1	......	......
				2	......	36.65	58.27	5.08	.66	......	......	......	......	......	......	......
				3	......	38.61	61.39	......	.70	......	......	......	......	......	......	......
Same (composite of samples 11429–11432).........	365	11433	A	1	2.95	35.01	56.44	5.60	.67	5.33	77.89	1.38	9.13	1.5	7,701	13,862
				2	......	36.07	58.16	5.77	.69	5.15	80.26	1.42	6.71	......	7,935	14,283
				3	......	38.28	61.72	......	.73	5.47	85.17	1.51	7.12	......	8,421	15,158
Montana Station, ¼ mile northwest of; Parker Run mine Sewickley bed (face of room 8, off right entry 5, 6-foot 2-inch bed, 5-foot 11½-inch cut).	366	15068	A	1	1.77	37.86	50.47	9.90	4.02	......	......	......	......	.8	7,392	13,306
				2	......	38.54	51.38	10.08	4.09	......	......	......	......	......	7,525	13,545
				3	......	42.86	57.14	......	4.55	......	......	......	......	......	8,369	15,064
Same (face of room 21, off right entry 3, 6-foot 1-inch bed, 5-foot 11-inch cut).	366	15069	A	1	1.89	37.94	51.51	8.66	3.64	......	......	......	......	.9	7,504	13,507
				2	......	38.67	52.50	8.83	3.71	......	......	......	......	......	7,649	13,768
				3	......	42.42	57.58	......	4.07	......	......	......	......	......	8,390	15,102
Same (face of fourth left air course, 1,800 feet northeast of mine mouth, 6-foot 3½-inch bed, 5-foot 11½-inch cut).	366	15071	A	1	1.93	39.60	49.35	9.12	4.17	......	......	......	......	.9	7,485	13,473
				2	......	40.38	50.32	9.30	4.25	......	......	......	......	......	7,632	13,738
				3	......	44.52	55.48	......	4.69	......	......	......	......	......	8,414	15,145
Same (composite of samples 15068, 15069, and 15071).	366	15072	A	1	1.81	39.06	49.81	9.32	3.90	5.15	73.62	1.30	6.71	.9	7,453	13,415
				2	......	39.78	50.73	9.49	3.97	5.04	74.97	1.32	5.21	......	7,590	13,662
				3	......	43.95	56.05	......	4.39	5.57	82.83	1.46	5.75	......	8,386	15,095
Same (pillar on main entry, at mouth of right entry 7; 2,400 feet northeast of mine mouth; 5-foot 11½-inch bed, 65-inch cut).	366	15067	A	1	1.75	39.22	49.56	9.47	3.68	......	......	......	......	.8	7,471	13,448
				2	......	39.92	50.44	9.64	3.75	......	......	......	......	......	7.604	13,687
				3	......	44.18	55.82	......	4.15	......	......	......	......	......	8.415	15,147
Same (pillar on main air course, at mouth of left entry 2, 5-foot 5-inch bed, 62-inch cut).	366	15070	A	1	1.89	38.98	49.11	10.02	4.08	......	......	......	......	.9	7,372	13,270
				2	......	39.73	50.06	10.21	4.16	......	......	......	......	......	7.514	13,525
				3	......	44.25	55.75	......	4.63	......	......	......	......	......	8,368	15,062
Worthington; Hutchinson mine, Pittsburgh bed (on pillar 15 on left heading 3, 2,500 feet south of mouth of main slope, 8-foot ½-inch bed and cut).	366	11152	A	1	2.60	35.19	53.48	8.73	1.92	......	......	......	......	1.4	......	......
				2	......	36.13	54.91	8.96	1.97	......	......	......	......	......	......	......
				3	......	39.69	60.31	......	2.16	......	......	......	......	......	......	......
Same (on pillar 20 on left heading 1, 1,500 feet south of mouth of main slope, 7-foot 7¼-inch bed and cut).	366	11153	A	1	2.26	36.90	52.03	8.81	2.47	......	......	......	......	1.3	......	......
				2	......	37.75	53.24	9.01	2.53	......	......	......	......	......	......	......
				3	......	41.49	58.51	......	2.78	......	......	......	......	......	......	......
Same (face of room 8, off left butt entry 4, 3,000 feet south of mouth of main slope, 8-foot 3¼-inch bed, 8 foot 3¼-inch cut).	366	11154	A	1	2.67	35.09	54.24	8.00	2.06	......	......	......	......	1.3	......	......
				2	......	36.05	55.73	8.22	2.12	......	......	......	......	......	......	......
				3	......	39.28	60.72	......	2.31	......	......	......	......	......	......	......
Same (face of room 10, off left heading 2, 1,000 feet south of mouth of main slope, 7-foot 7¼-inch bed and cut).	366	11155	A	1	2.66	35.99	52.73	8.62	2.04	......	......	......	......	1.5	......	......
				2	......	36.97	54.17	8.86	2.10	......	......	......	......	......	......	......
				3	......	40.56	59.44	......	2.30	......	......	......	......	......	......	......
Same (composite of samples 11152–11155).........	366	11156	A	1	2.60	35.88	52.83	8.69	2.03	5.22	74.69	1.25	8.12	1.4	7,508	13.513
				2	......	36.84	54.24	8.92	2.08	5.06	76.68	1.28	5.98	......	7.708	13,874
				3	......	40.45	59.55	......	2.28	5.56	84.19	1.41	6.56	......	8.463	15.233

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
MARSHALL COUNTY.																
Moundsville, 1 mile south of: Panama mine, Pittsburgh bed (face of second south air course, 2,050 feet southeast of shaft, 4-foot 9¼-inch bed, 4-foot 8¼-inch cut).	367	14486	A	1	3.35	41.35	48.29	7.01	3.19	......	......	......	......	1.6	......	......
				2	......	42.78	49.97	7.25	3.30	......	......	......	......	......	......	......
				3	......	46.13	53.87	......	3.56	......	......	......	......	......	......	......
Same (face of room 3, off right entry 2, off north entry 2, 1,700 feet northeast of shaft, 4-foot 7⅞-inch bed, 4-foot 5⅝ inch cut).	367	14487	A	1	3.52	41.08	48.34	7.06	3.24	......	......	......	......	1.8	......	......
				2	......	42.58	50.10	7.32	3.36	......	......	......	......	......	......	......
				3	......	45.94	54.06	......	3.63	......	......	......	......	......	......	......
Same (composite of samples 14486 and 14487).....	367	14488	A	1	3.39	41.58	48.01	7.02	3.25	5.38	72.86	1.32	10.17	1.7	7,396	13,313
				2	......	43.04	49.69	7.27	3.36	5.18	75.42	1.37	7.40	......	7,656	13,781
				3	......	46.41	53.59	......	3.62	5.59	81.33	1.48	7.98	......	8,256	14,861
MERCER COUNTY.																
Crystal, ¼-mile southeast of; Godfrey mine, Pocahontas No. 3 bed (pillar of right entry 2, 1,100 feet northeast of mine mouth, 4-foot bed, 3-foot 7-inch cut).	368	14258	A	1	2.74	18.41	75.33	3.52	.71	......	......	......	......	2.0	......	......
				2	......	18.93	77.45	3.62	.73	......	......	......	......	......	......	......
				3	......	19.64	80.36	......	.76	......	......	......	......	......	......	......
Same (room 3, off left heading 2, off old main entry, 700 feet southeast of mine mouth, 4-foot 3-inch bed, 4-foot cut).	368	14259	A	1	3.36	17.68	74.76	4.20	.72	......	......	......	......	2.6	......	......
				2	......	18.30	77.35	4.35	.75	......	......	......	......	......	......	......
				3	......	19.13	80.87	......	.78	......	......	......	......	......	......	......
Same (composite of samples 14258 and 14259).....	368	14260	A	1	3.06	18.24	74.84	3.86	.71	4.74	84.32	1.12	5.25	2.3	8,160	14,688
				2	......	18.82	77.20	3.98	.73	4.54	86.98	1.16	2.61	......	8,418	15,152
				3	......	19.60	80.40	......	.76	4.73	90.58	1.21	2.72	......	8,767	15,781
Same (face of room 4, next to "Crystal," 4-foot 7-inch bed, 4-foot 3-inch cut).	368	14261	A	1	3.26	17.99	75.20	3.55	.68	......	......	......	......	2.5	8,163	14,693
				2	......	18.60	77.73	3.67	.70	......	......	......	......	......	8,438	15,188
				3	......	19.31	80.69	......	.73	......	......	......	......	......	8,759	15,766
Goodwill, ½ mile northeast of; Goodwill mine, Pocahontas No. 3 bed (face of room 16, off eighth left cross entry, 3,700 feet northeast of No. 3 drift mouth, 4-foot 9-inch bed, 4-foot 4-inch cut).	369	14163	A	1	3.14	18.91	73.92	4.03	.58	......	......	......	......	2.4	......	......
				2	......	19.52	76.32	4.16	.60	......	......	......	......	......	......	......
				3	......	20.37	79.63	......	.63	......	......	......	......	......	......	......

Same (face of room 3, off second crosscut, off Smith's entry, 3,000 feet northeast of No. 3 drift mouth, 4-foot 9-inch bed, 4-foot 4-inch cut).	369	14164	A	1	3.09	18.22	75.83	2.86	.57	......	......	......	......	2.4	......	......
				2	......	18.80	78.25	2.95	.59	......	......	......	......	......	......	......
				3	......	19.37	80.63	......	.61	......	......	......	......	......	......	......
Same (face of first crosscut, off Nichol's entry, 2,850 feet east of No. 2 drift mouth 4-foot 6-inch bed, 4-foot 1-inch cut).	369	14165	A	1	3.12	18.90	74.76	3.22	.56	......	......	......	......	2.5	......	......
				2	......	19.51	77.17	3.32	.58	......	......	......	......	......	......	......
				3	......	20.18	79.82	......	.60	......	......	......	......	......	......	......
Same (pillar 4, 2,400 feet east of No. 2 drift mouth, 4-foot 5½-inch bed, 4-foot 1-inch cut).	369	14166	A	1	3.49	18.59	74.98	2.94	.54	......	......	......	......	2.9	......	......
				2	......	19.26	77.69	3.05	.56	......	......	......	......	......	......	......
				3	......	19.87	80.13	......	.58	......	......	......	......	......	......	......
Same (composite of samples 14163–14166).........	369	14167	A	1	3.19	19.14	74.36	3.31	.56	4.60	85.16	1.10	5.27	2.6	8,175	14,715
				2	......	19.77	76.81	3.42	.58	4.39	87.97	1.14	2.50	......	8,445	15,201
				3	......	20.47	79.53	......	.60	4.55	91.08	1.18	2.59	......	8,744	15,739
½ mile northwest of; Louisville No. 1 mine (pillar of main entry, 900 feet northeast of mine mouth, 4-foot 10-inch bed, 4-foot 5-inch cut).	370	14207	A	1	3.11	19.30	74.27	3.32	.57	......	......	......	......	2.5	......	......
				2	......	19.92	76.65	3.43	.59	......	......	......	......	......	......	......
				3	......	20.63	79.37	......	.61	......	......	......	......	......	......	......
½ mile northwest of; Louisville No. 2 mine, Pocahontas No. 3 bed (face of main entry, 5,400 feet northeast of mine mouth, 4-foot 8-inch bed, 4-foot 3-inch cut).	370	14191	A	1	2.92	18.75	75.35	2.98	.56	......	......	......	......	2.3	......	......
				2	......	19.31	77.62	3.07	.58	......	......	......	......	......	......	......
				3	......	19.92	80.08	......	.60	......	......	......	......	......	......	......
Same (face of room 19, off left heading 9, 5,700 feet northeast of mine mouth, 4-foot 6¼-inch bed, 4-foot 2¼-inch cut).	370	14192	A	1	3.54	18.47	74.96	3.03	.48	......	......	......	......	2.8	......	......
				2	......	19.15	77.71	3.14	.50	......	......	......	......	......	......	......
				3	......	19.77	80.23	......	.52	......	......	......	......	......	......	......
Same (rib of room 14, off left heading 6, 4,500 feet northeast of mine mouth, 4-foot 6½-inch bed, 4-foot 2½-inch cut).	370	14193	A	1	3.12	18.64	74.37	3.87	.50	......	......	......	......	2.4	......	......
				2	......	19.24	76.77	3.99	.52	......	......	......	......	......	......	......
				3	......	20.04	79.96	......	.54	......	......	......	......	......	......	......
Same (rib of room 13, off left heading 4, 3,000 feet northeast of mine mouth, 4-foot 8-inch bed, 4-foot 5½-inch cut).	370	14194	A	1	2.61	18.85	73.59	4.95	.49	......	......	......	......	1.9	......	......
				2	......	19.36	75.56	5.08	.50	......	......	......	......	......	......	......
				3	......	20.40	79.60	......	.53	......	......	......	......	......	......	......
1 mile northwest of; Louisville No. 3 mine (barrier pillar between second left air course and second left pillar, 1,500 feet northwest of mine mouth, 5-foot 1-inch bed, 4-foot 9-inch cut).	370	14195	A	1	3.10	19.11	73.73	4.06	.47	......	......	......	......	2.5	......	......
				2	......	19.72	76.09	4.19	.49	......	......	......	......	......	......	......
				3	......	20.58	79.42	......	.51	......	......	......	......	......	......	......
Same (face, 1,950 feet northeast of mine mouth, 4-foot 9-inch bed, 4-foot 5-inch cut).	370	14206	A	1	4.10	19.18	73.60	3.12	.53	......	......	......	......	3.5	......	......
				2	......	20.00	76.75	3.25	.55	......	......	......	......	......	......	......
				3	......	20.67	79.33	......	.57	......	......	......	......	......	......	......
Same (composite of samples 14191, 14192, 14193, 14194, 14195, 14206, and 14207).	370	14196	A	1	3.30	18.58	74.45	3.67	.56	4.56	84.83	1.16	5.22	2.6	8,142	14,656
				2	......	19.21	76.99	3.80	.58	4.33	87.72	1.20	2.37	......	8,420	15,156
				3	......	19.97	80.03	......	.60	4.50	91.18	1.25	2.47	......	8,753	15,755
Matoaka; Pawama No. 1 mine (pillar of room 14, off Walker heading, 1,000 feet north of mine mouth, 4-foot 7½-inch bed, 4-foot ½-inch cut).	371	14235	A	1	3.55	17.17	75.19	4.09	.76	......	......	......	......	2.8	......	......
				2	......	17.80	77.96	4.24	.79	......	......	......	......	......	......	......
				3	......	18.59	81.41	......	.82	......	......	......	......	......	......	......
Same (pillar on Godfrey heading, 1,520 feet southwest of mine mouth, 4-foot 5-inch bed, 3-foot 11½-inch cut).	371	14236	A	1	2.58	17.56	76.50	3.36	.73	......	......	......	......	1.9	......	......
				2	......	18.03	78.52	3.45	.75	......	......	......	......	......	......	......
				3	......	18.67	81.33	......	.78	......	......	......	......	......	......	......
Same (composite of samples 14235 and 14236)	371	14237	A	1	3.16	16.98	76.21	3.65	.71	4.64	84.79	1.15	5.06	2.4	8,183	14,729
				2	......	17.53	78.70	3.77	.73	4.43	87.55	1.19	2.33	.....	8,450	15,210
				3	......	18.22	81.78	......	.76	4.60	90.98	1.24	2.42	......	8,781	15,806

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
MERCER COUNTY—continued.																
Matoaka; Pawama No. 2 mine, Pocahontas No. 3 bed (face of Roanoke heading, 2,060 feet southwest of mine mouth, 4-foot 4¾-inch bed, 4-foot 2-inch cut).	371	14231	A	1	4.35	17.33	74.16	4.16	.75	......	......	......	......	3.7	......	......
				2	......	18.12	77.53	4.35	.78	......	......	......	......	......	......	......
				3	......	18.94	81.06	......	.82	......	......	......	......	......	......	......
Same (face of main entry, 2,600 feet southwest of mine mouth, 4-foot 3½-inch bed, 4-foot 1-inch cut).	371	14232	A	1	3.28	17.14	75.57	4.01	.77	......	......	......	......	2.6	......	......
				2	......	17.72	78.13	4.15	.80	......	......	......	......	......	......	......
				3	......	18.49	81.51	......	.83	......	......	......	......	......	......	......
Same (face of room 20, off Norfolk entry, 2,300 feet southwest of mine mouth, 4-foot 5½-inch bed, 4-foot cut).	371	14233	A	1	3.02	17.59	75.00	4.39	.78	......	......	......	......	2.3	......	......
				2	......	18.14	77.33	4.53	.80	......	......	......	......	......	......	......
				3	......	19.00	81.00	......	.84	......	......	......	......	......	......	......
Same (composite of samples 14231, 14232, and 14233).	371	14234	A	1	3.61	17.41	74.84	4.14	.76	4.57	83.68	1.12	5.73	2.9	8,104	14,587
				2	......	18.06	77.64	4.30	.79	4.33	86.82	1.16	2.60	......	8,408	15,134
				3	......	18.87	81.13	......	.83	4.52	90.72	1.21	2.72	......	8,786	15,815
Mora, ½ mile west of; Crane Creek No. 1 mine, Pocahontas No. 3 bed (face of main heading, 4-foot 11-inch bed, 4-foot 6½-inch cut).	372	W32517	A	1	2.3	18.0	73.3	6.4	.60	......	......	......	......	......	7,978	14,360
				2	......	18.4	75.1	6.5	.60	......	......	......	......	......	8,167	14,700
				3	......	19.7	80.3	......	.64	......	......	......	......	......	8,733	15,720
Same (face of eighteenth cross air course, 4-foot 9-inch bed, 4-foot 4½-inch cut).	372	W32518	A	1	2.3	17.9	74.3	5.5	.55	......	......	......	......	......	8,072	14,530
				2	......	18.3	76.1	5.6	.55	......	......	......	......	......	8,267	14,880
				3	......	19.4	80.6	......	.58	......	......	......	......	......	8,756	15,760
Same (pillar beyond second break-through in room 11, off cross entry 16, 5-foot 7½-inch bed, 5-foot 3-inch cut).	372	W32519	A	1	2.2	17.0	75.1	5.7	.70	......	......	......	......	......	8,083	14,550
				2	......	17.4	76.8	5.8	.70	......	......	......	......	......	8,267	14,880
				3	......	18.5	81.5	......	.74	......	......	......	......	......	8,778	15,800
Same (pillar beyond second break-through in room 21, off cross entry 6, 4-foot 5-inch bed, 4-foot 3½-inch cut).	372	W32520	A	1	2.4	17.6	74.9	5.1	.65	......	......	......	......	......	8,128	14,630
				2	......	18.0	76.8	5.2	.65	......	......	......	......	......	8,322	14,980
				3	......	19.0	81.0	......	.69	......	......	......	......	......	8,778	15,800
¾ mile west of; Crane Creek No. 2 mine, Pocahontas No. 3 bed (cut from pocket beyond second break-through in room 4, off cross entry 3, 5-foot ½-inch bed, 4-foot 4½-inch cut).	372	W32600	A	1	3.3	17.5	73.7	5.5	.55	......	......	......	......	......	7,967	14,340
				2	......	18.1	76.2	5.7	.55	......	......	......	......	......	8,244	14,840
				3	......	19.2	80.8	......	.58	......	......	......	......	......	8,744	15,740
Same (face of room 14, off Allegheny entry, 5-foot 3-inch bed, 4-foot 10-inch cut).	372	W32601	A	1	2.0	18.5	75.5	4.0	.60	......	......	......	......	......	8,256	14,860
				2	......	18.9	77.0	4.1	.60	......	......	......	......	......	8,422	15,160
				3	......	19.7	80.3	......	.63	......	......	......	......	......	8,783	15,810

Same (chain pillar between rooms 21 and 22, off Blue Ridge entry, 5-foot ¾-inch bed, 4-foot 9-inch cut).	372	W32602	A	1	2.1	18.8	74.9	4.2	.55	....	....	....	....	....	8,200	14,760
				2	....	19.2	76.5	4.3	.55	....	....	....	....	....	8,378	15,080
				3	....	20.1	79.9	....	.57	....	....	....	....	....	8,756	15,760
Same (face of room 13, off Bryant entry, 4-foot 8½-inch bed, 4-foot 1-inch cut).	372	W32603	A	1	2.4	18.3	75.4	3.9	.55	....	....	....	....	....	8,211	14,780
				2	....	18.8	77.2	4.0	.55	....	....	....	....	....	8,411	15,140
				3	....	19.6	80.4	....	.57	....	....	....	....	....	8,761	15,770
1 mile north of; Pinnacle mine, Pocahontas No. 3 bed (face of cross entry 2, north mine, 4-foot 2¾-inch bed, 3-foot 11¾-inch cut).	373	W32604	A	1	2.4	18.1	76.3	3.2	.55	....	....	....	....	....	8,261	14,870
				2	....	18.6	78.1	3.3	.55	....	....	....	....	....	8,467	15,240
				3	....	19.2	80.8	....	.57	....	....	....	....	....	8,756	15,760
Same (face of north main heading, 4-foot 4½-inch bed, 4-foot cut).	373	W32605	A	1	2.7	17.3	76.4	3.6	.55	....	....	....	....	....	8,217	14,790
				2	....	17.8	78.5	3.7	.55	....	....	....	....	....	8,444	15,200
				3	....	18.5	81.5	....	.57	....	....	....	....	....	8,767	15,780
Same (face of pocket in first pillar of room 25, cross entry 2, off main heading, 4-foot 8½-inch bed, 4-foot 3-inch cut).	373	W32606	A	1	2.4	17.8	75.7	4.1	.60	....	....	....	....	....	8,183	14,730
				2	....	18.2	77.6	4.2	.60	....	....	....	....	....	8,383	15,090
				3	....	19.0	81.0	....	.63	....	....	....	....	....	8,750	15,750
Same (face of break-through off Cuba entry, off main heading 2, 4-foot 9¾-inch bed, 4-foot 4-inch cut).	373	W32607	A	1	2.4	18.3	75.5	3.8	.55	....	....	....	....	....	8,200	14,760
				2	....	18.7	77.4	3.9	.55	....	....	....	....	....	8,400	15,120
				3	....	19.5	80.5	....	.57	....	....	....	....	....	8,744	15,740
Same (face of main heading 2, 4-foot 8½-inch bed, 4-foot 4-inch cut).	373	W32608	A	1	2.6	17.8	75.9	3.7	.60	....	....	....	....	....	8,211	14,780
				2	....	18.3	77.9	3.8	.60	....	....	....	....	....	8,433	15,180
				3	....	19.0	81.0	....	.62	....	....	....	....	....	8,767	15,780
Same (pillar of room 16, off cross entry 4, off main heading 1, 4-foot 11-inch bed, 4-foot 6-inch cut).	373	W32609	A	1	2.8	17.1	76.3	3.8	.55	....	....	....	....	....	8,172	14,710
				2	....	17.6	78.5	3.9	.55	....	....	....	....	....	8,406	15,130
				3	....	18.3	81.7	....	.57	....	....	....	....	....	8,744	15,740
1 mile west of; Sagamore No. 1 mine, Pocahontas No. 3 bed (face of main straight entry H, 2,250 feet northwest of mine mouth, 5-foot bed, 3-foot 11-inch cut).	374	14250	A	1	2.53	18.56	74.89	4.02	.52	....	....	....	....	1.8	....	....
				2	....	19.04	76.84	4.12	.53	....	....	....	....	....	....	....
				3	....	19.86	80.14	....	.55	....	....	....	....	....	....	....
Same (face of G-1 heading, 2,250 feet west of mine mouth, 5-foot 6½-inch bed, 4-foot 9-inch cut).	374	14251	A	1	3.20	18.43	74.24	4.13	.57	....	....	....	....	2.4	....	....
				2	....	19.04	76.69	4.27	.59	....	....	....	....	....	....	....
				3	....	19.89	80.11	....	.62	....	....	....	....	....	....	....
Same (face of room 11, off G-2 heading, 1,900 feet southwest of mine mouth, 5-foot 11-inch bed, 4-foot 11-inch cut).	374	14252	A	1	3.53	18.20	74.99	3.28	.54	....	....	....	....	2.7	....	....
				2	....	18.87	77.73	3.40	.56	....	....	....	....	....	....	....
				3	....	19.53	80.47	....	.58	....	....	....	....	....	....	....
1 mile east of; Sagamore No. 2 mine (face of H-8 entry, 2,550 feet southeast of mine mouth, 4-foot 6-inch bed, 4-foot 2-inch cut).	375	14253	A	1	3.36	18.33	74.93	3.38	.57	....	....	....	....	2.6	....	....
				2	....	18.97	77.53	3.50	.59	....	....	....	....	....	....	....
				3	....	19.66	80.34	....	.61	....	....	....	....	....	....	....
Same (face of G-6 entry, 2,300 feet southwest of mine mouth, 4-foot 5½-inch bed, 4-foot 1½-inch cut).	375	14254	A	1	2.62	18.56	74.98	3.84	.55	....	....	....	....	1.8	....	....
				2	....	19.06	77.00	3.94	.56	....	....	....	....	....	....	....
				3	....	19.84	80.16	....	.58	....	....	....	....	....	....	....
Same (face of G-8 entry, 2,550 feet south of mine mouth, 4-foot 6-inch bed, 4-foot 2-inch cut).	375	14255	A	1	3.24	18.24	75.19	3.33	.61	....	....	....	....	2.4	....	....
				2	....	18.85	77.71	3.44	.63	....	....	....	....	....	....	....
				3	....	19.52	80.48	....	.65	....	....	....	....	....	....	....
Same (face of straight H air course, 2,350 feet southeast of mine mouth, 4-foot 6-inch bed and cut).	375	14256	A	1	3.11	18.18	73.38	5.33	.56	....	....	....	....	2.3	....	....
				2	....	18.76	75.74	5.50	.58	....	....	....	....	....	....	....
				3	....	19.85	80.15	....	.61	....	....	....	....	....	....	....
Same (composite of samples 14250-14256).........	375	14257	A	1	3.16	18.17	74.69	3.98	.56	4.60	84.13	1.17	5.56	2.3	8,133	14,639
				2	....	18.76	77.13	4.11	.58	4.39	86.87	1.21	2.84	....	8,398	15,116
				3	....	19.56	80.44	....	.60	4.58	90.60	1.26	2.96	....	8,758	15,764

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
MERCER COUNTY—continued.																
Weyanoke: Weyanoke No. 1 mine; Pocahontas No. 3 bed (face of room 11, off left entry 9, off main heading, 4-foot 1½-inch bed, 3-foot 11-inch cut).	376	14224	A	1	2.57	18.66	75.35	3.42	.56	......	......	......	......	3.0	......	......
				2	......	19.15	77.34	3.51	.57	......	......	......	......	......	......	......
				3	......	19.85	80.15	......	.59	......	......	......	......	......	......	......
Same (face of main air course; 4-foot ½-inch bed, 3-foot 10-inch cut).	376	14225	A	1	2.74	18.81	74.42	4.08	.60	......	......	......	......	2.1	......	......
				2	......	19.34	76.52	4.14	.62	......	......	......	......	......	......	......
				3	......	20.18	79.82	......	.65	......	......	......	......	......	......	......
Same (face of room 22, off left cross heading 8, 4-foot 6-inch bed, 4-foot 3-inch cut).	376	14226	A	1	3.25	17.33	75.40	4.02	.63	......	......	......	......	2.6	......	......
				2	......	17.91	77.93	4.16	.65	......	......	......	......	......	......	......
				3	......	18.69	81.31	......	.68	......	......	......	......	......	......	......
Same (face of room 6, off left entry 6, 4-foot 3½-inch bed, 4-foot cut).	376	14228	A	1	3.22	17.84	75.47	3.47	.59	......	......	......	......	2.5	......	......
				2	......	18.43	77.98	3.59	.61	......	......	......	......	......	......	......
				3	......	19.12	80.88	......	.63	......	......	......	......	......	......	......
Weyanoke No. 2 mine, Pocahontas No. 3 bed (face of right entry 3, off main entry, 4-foot bed, 3-foot 11-inch cut).	376	14227	A	1	3.10	18.06	75.63	3.21	.64	......	......	......	......	2.5	......	......
				2	......	18.64	78.05	3.31	.66	......	......	......	......	......	......	......
				3	......	19.28	80.72	......	.68	......	......	......	......	......	......	......
Same (pillar of last crosscut in room 7, off left butt entry 2; 4-foot 4-inch bed, 3-foot 11-inch cut).	376	14230	A	1	2.74	17.84	75.48	3.94	.63	......	......	......	......	2.0	8,186	14.735
				2	......	18.34	77.61	4.05	.65	......	......	......	......	......	8,417	15,151
				3	......	19.11	80.89	......	.68	......	......	......	......	......	8,772	15,790
Same (composite of samples 14224, 14225, 14226, 14227, and 14228).	376	14229	A	1	2.97	17.72	75.68	3.63	.65	4.52	85.26	1.16	4.78	2.3	8,167	14,701
				2	......	18.26	78.00	3.74	.67	4.32	87.87	1.20	2.20	......	8,417	15,151
				3	......	18.97	81.03	......	.70	4.49	91.29	1.25	2.27	......	8,744	15,739
MONONGALIA COUNTY.																
Richard, ¼ mile northeast of; Elkins No. 1 mine, Upper Freeport bed (face of fifth right "flat," 4,200 feet northeast of mine mouth, 4-foot ¾-inch bed, 3-foot 10½-inch cut).	377	W19475	A	1	2.2	28.7	62.1	7.0	.80	......	......	......	......	......	7,828	14,090
				2	......	29.4	63.4	7.2	.80	......	......	......	......	......	8,006	14,410
				3	......	31.7	68.3	......	.86	......	......	......	......	......	8,628	15,530
Same (face of butt entry 8 off right entry 5, 700 feet from main entry, 3-foot 7½-inch bed, 3-foot 6-inch cut).	377	W19476	A	1	2.6	27.2	61.9	8.3	.90	......	......	......	......	......	7,622	13,720
				2	......	27.9	63.6	8.5	.95	......	......	......	......	......	7,828	14,090
				3	......	30.5	69.5	......	1.04	......	......	......	......	......	8,556	15,400

Same (face of right heading 1, 4,200 feet from mine mouth, 4-foot 2¼-inch bed, 4-foot 1½-inch cut).	377	W19477	A	1	2.7	27.9	61.1	8.3	.90						7,672	13,810
				2		28.7	62.8	8.5	.95						7,883	14,190
				3		31.4	68.6		1.04						8,611	15,500
Same (face of butt entry 11, off right entry 1, 750 feet from right entry 1, 3-foot 11¼-inch bed, 3-foot 9-inch cut).	377	W19478	A	1	2.2	28.8	61.5	7.5	.90						7,778	14,000
				2		29.1	63.2	7.7	.95						7,950	14,310
				3		31.5	68.5		1.03						8,617	15,510
Same (butt entry 5, off fifth right "flat," 1,400 feet from fifth "flat," 4-foot ½-inch bed, 3-foot 10½-inch cut).	377	W19479	A	1	2.7	28.2	60.3	8.8	.70						7,617	13,710
				2		29.0	62.0	9.0	.75						7,822	14,080
				3		31.9	68.1		.82						8,594	15,470
Same (composite of samples 19475–19479)	377	12204	A	1	1.63	28.42	62.01	7.94	.96	5.00	78.24	1.28	6.58		7,743	13,937
				2		28.89	63.04	8.07	.98	4.90	79.54	1.30	5.21		7,872	14,170
				3		31.43	68.57		1.07	5.33	86.52	1.41	5.67		8,563	15,313
PRESTON COUNTY.																
Masontown, at; Elkins No. 6 mine, Upper Freeport bed (face of butt heading 1, 600 feet from third "flat," 150 feet from main entry, 4-foot ⅞-inch bed, 3-feet 11⅞-inch cut).	378	W19493	A	1	2.2	26.5	60.9	10.4	1.85						7,522	13,540
				2		27.1	62.3	10.6	1.90						7,689	13,840
				3		30.3	69.7		2.13						8,606	15,490
Same (face of right entry 3, off main entry, 50 feet from main entry, 4-foot 1⅜-inch bed, 3-foot 11⅜-inch cut).	378	W19494	A	1	2.7	24.2	64.7	8.4	1.15						7,606	13,690
				2		24.9	66.5	8.6	1.20						7,811	14,060
				3		27.2	72.8		1.31						8,544	15,380
Same (face of butt heading 2, 1,000 feet from main entry, 4-foot 1⅜-inch bed, 3-foot 11⅜-inch cut).	378	W19540	A	1	2.0	27.0	63.1	7.9	.75						7,739	13,930
				2		27.6	64.3	8.1	.75						7,894	14,210
				3		30.0	70.0		.82						8,594	15,470
Same (butt entry 2, 700 feet from second "flat," 300 feet from main entry, 4-foot 3½-inch bed, 4-foot ⅜-inch cut).	378	W19541	A	1	2.5	27.0	60.8	9.7	1.70						7,550	13,590
				2		27.7	62.4	9.9	1.75						7,739	13,930
				3		30.7	69.3		1.94						8,589	15,460
Same (composite of samples W19493, W19494, W19540, and W19541).	378	12225	A	1	1.40	26.40	62.92	9.28	1.50	4.83	77.92	1.43	5.04		7,671	13,808
				2		26.77	63.82	9.41	1.52	4.74	79.03	1.45	3.85		7,780	14,004
				3		29.55	70.45		1.68	5.23	87.24	1.60	4.25		8,588	15,458
RALEIGH COUNTY.																
Hotcoal; Big Stick mine, Beckley bed (face of left heading 2, 1,600 feet east of mouth of straight main entry, 5-foot 6-inch bed, 5-foot 3½-inch cut).	379	14375	A	1	2.82	16.64	75.57	4.97	.52					2.0		
				2		17.12	77.77	5.11	.54							
				3		18.04	81.96		.57							
Same (face of left entry 3, off straight main entry, 1,400 feet northeast of mine mouth, 5-foot 3½-inch bed and cut).	379	14376	A	1	2.37	17.14	74.95	5.54	.80					1.6		
				2		17.56	76.77	5.67	.82							
				3		18.62	81.38		.87							
Same (face of straight main entry, 1,900 feet northeast of mine mouth, 6-foot 3½-inch bed, 6-foot 1-inch cut).	379	14377	A	1	1.57	17.38	77.05	4.00	.51					.7		
				2		17.66	78.28	4.06	.52							
				3		18.41	81.59		.54							
Same (composite of samples 14375, 14376, and 14377).	379	14378	A	1	2.24	17.54	75.40	4.82	.63	4.52	83.73	1.62	4.68	1.4	8,096	14,573
				2		17.94	77.13	4.93	.64	4.37	85.65	1.66	2.75		8,281	14,906
				3		18.87	81.13		.67	4.60	90.09	1.75	2.80		8,710	15,678

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
RALEIGH COUNTY—continued.																
McQuade station, 2 miles southeast of Raleigh; Blue Jay No. 4 mine, Beckley bed (pillar of main heading, 600 feet east of mine mouth, 4-foot 11-inch bed, 4-foot 2-inch cut).	379	14288	A	1	3.53	19.01	74.38	3.08	.75	......	......	......	......	2.4	......	......
				2	......	19.71	77.10	3.19	.79	......	......	......	......	......	......	......
				3	......	20.36	79.64	......	.81	......	......	......	......	......	......	......
Same (face of right heading 2; 400 feet south of mine mouth; 4-foot 5-inch bed, 4-foot 2-inch cut).	379	14289	A	1	2.89	19.49	73.29	4.33	.78	......	......	......	......	1.8	......	......
				2	......	20.07	75.47	4.46	.80	......	......	......	......	......	......	......
				3	......	21.01	78.99	......	.84	......	......	......	......	......	......	......
Same (pillar of right heading 1; 250 feet south of mine mouth; 4-foot 5-inch bed and cut).	379	14290	A	1	3.40	18.74	72.56	5.30	.66	......	......	......	......	2.3	......	......
				2	......	19.40	75.11	5.49	.68	......	......	......	......	......	......	......
				3	......	20.53	79.47	......	.72	......	......	......	......	......	......	......
Same (composite of samples 14288, 14289, and 14290).	380	14291	A	1	3.29	18.96	73.46	4.29	.74	4.78	82.96	1.61	5.62	2.2	8,066	14,519
				2	......	19.60	75.96	4.44	.77	4.56	85.78	1.66	2.79	......	8,340	15,012
				3	......	20.51	79.49	......	.81	4.77	89.77	1.74	2.91	......	8,728	15,710
Raleigh; No. 1 mine, Beckley bed (neck of the last break-through off right entry 6, 5-foot 1⅜-inch bed and cut).	380	10630	A	1	2.80	17.99	75.61	3.60	.63	5.05	83.72	1.56	5.44	2.3	8,170	14,706
				2	......	18.51	77.79	3.70	.65	4.88	86.13	1.60	3.04	......	8,405	15,129
				3	......	19.22	80.78	......	.67	5.07	89.44	1.66	3.16	......	8,728	15,710
No. 3 mine, Beckley bed (left rib of left entry 7, opposite mouth of right heading 1, 5-foot 2⅜-inch bed and cut).	380	10624	A	1	3.47	18.07	74.87	3.59	.66	......	......	......	......	2.9	......	......
				2	......	18.72	77.56	3.72	.68	......	......	......	......	......	......	......
				3	......	19.44	80.56	......	.71	......	......	......	......	......	......	......
Same (face of right entry 15, 4-foot 2-inch bed and cut).	380	10625	A	1	1.58	18.87	76.37	3.18	.68	......	......	......	......	1.1	......	......
				2	......	19.17	77.60	3.23	.69	......	......	......	......	......	......	......
				3	......	19.81	80.19	......	.71	......	......	......	......	......	......	......
Same (last crosscut in main entry, 4-foot 1⅞-inch bed and cut).	380	10626	A	1	8.77	17.22	70.26	3.75	.73	......	......	......	......	8.2	......	......
				2	......	18.87	77.02	4.11	.80	......	......	......	......	......	......	......
				3	......	19.68	80.32	......	.83	......	......	......	......	......	......	......
Same (face of room 11, off right heading 12, 3-foot 7½-inch bed and cut).	380	10627	A	1	2.76	19.89	73.82	3.53	.81	......	......	......	......	2.3	......	......
				2	......	20.45	75.92	3.63	.83	......	......	......	......	......	......	......
				3	......	21.22	78.78	......	.86	......	......	......	......	......	......	......
Same (composite of samples 10624-10627).........	380	10628	A	1	4.37	18.06	74.08	3.49	.71	4.95	82.72	1.49	6.64	3.8	8,039	14,470
				2	......	18.89	77.46	3.65	.74	4.66	86.50	1.56	2.89	......	8,406	15,131
				3	......	19.61	80.39	......	.77	4.84	89.78	1.62	2.99	......	8,724	15,703

Slab Fork, about ¾ mile northeast of; Slab Fork No. 1 mine, Beckley bed (pillar of room 7 off right entry 2; 1,200 feet northeast of mine mouth 5-foot 4½-inch bed, 4-foot 10½-inch cut).	381	14337	A	1	4.93	19.96	72.38	2.78	.58	4.77	81.61	1.58	8.73	2.8	7,872	14,170
				2	....	21.00	76.13	2.87	.61	4.44	85.85	1.66	4.57	....	8,281	14,906
				3	....	21.62	78.38	....	.63	4.57	88.38	1.71	4.71	....	8,525	15,345
About ¾ mile northeast of; Slab Fork No. 2 mine, Beckley bed (pillar of left entry 3, between rooms 7 and 8, 1,800 feet southwest of mine mouth, 5-foot 6½-inch bed, 4-foot 10-inch cut).	381	14325	A	1	2.07	17.83	75.30	4.80	.53	....	....	....	....	1.3	....	....
				2	....	18.21	76.89	4.90	.54	....	....	....	....	....	....	....
				3	....	19.15	80.85	....	.57	....	....	....	....	....	....	....
Same (pillar between rooms 8 and 10, off left entry 7; 3,000 feet southwest of mine mouth, 4-foot 6-inch bed, 4-foot cut).	381	14326	A	1	2.38	17.52	77.33	2.77	.52	....	....	....	....	1.5	....	....
				2	....	17.95	79.21	2.84	.53	....	....	....	....	....	....	....
				3	....	18.47	81.53	....	.55	....	....	....	....	....	....	....
Same (composite of samples 14325 and 14326).....	381	14327	A	1	2.27	17.26	76.73	3.74	.54	4.62	84.90	1.57	4.63	1.4	8,196	14,753
				2	....	17.66	78.51	3.83	.55	4.47	86.87	1.61	2.67	....	8,386	15,095
				3	....	18.36	81.64	....	.57	4.65	90.33	1.67	2.78	....	8,720	15,696
Same (face of first panel off second drift, 4,200 feet south of mine mouth, 4-foot 2-inch bed, 3-foot 8-inch cut).	381	14328	A	1	2.57	17.60	77.16	2.67	.50	....	....	....	....	1.6	....	....
				2	....	18.06	79.20	2.74	.51	....	....	....	....	....	....	....
				3	....	18.57	81.43	....	.52	....	....	....	....	....	....	....
Same (face of right entry 1, off left entry 3; 1,200 feet southeast of mine mouth, 4-foot 4½-inch bed, 4-foot ¾-inch cut).	381	14329	A	1	2.09	17.83	75.93	4.15	.53	....	....	....	....	1.3	....	....
				2	....	18.21	77.55	4.24	.54	....	....	....	....	....	....	....
				3	....	19.02	80.98	....	.56	....	....	....	....	....	....	....
Same (face of left entry 14, 150 feet from straight main entry, 5,000 feet southwest of mine mouth, 4-foot 6-inch bed, 4-foot cut).	381	14330	A	1	2.58	17.88	76.41	3.13	.48	....	....	....	....	1.8	....	....
				2	....	18.35	78.44	3.21	.49	....	....	....	....	....	....	....
				3	....	18.96	81.04	....	.51	....	....	....	....	....	....	....
Same (composite of samples 14328, 14329, and 14330).	382	14331	A	1	2.45	17.52	76.67	3.36	.48	4.56	85.51	1.51	4.58	1.6	8,216	14,789
				2	....	17.96	78.60	3.44	.49	4.40	87.66	1.55	2.46	....	8,422	15,160
				3	....	18.60	81.40	....	.51	4.56	90.78	1.61	2.54	....	8,722	15,700
About ¾ mile northeast of; Slab Fork No. 3 mine, Beckley bed (face of main air course, 2,450 feet southeast of mine mouth, 5-foot 3½-inch bed, 4-foot 10½-inch cut).	382	14332	A	1	2.44	18.00	76.15	3.41	.60	....	....	....	....	1.6	....	....
				2	....	18.45	78.05	3.50	.62	....	....	....	....	....	....	....
				3	....	19.12	80.88	....	.64	....	....	....	....	....	....	....
Same (face of A entry, 2,350 feet southeast of mine mouth, 4-foot 2-inch bed, 3-foot 8½-inch cut).	382	14333	A	1	3.38	17.70	75.94	2.98	.48	....	....	....	....	2.4	....	....
				2	....	18.32	78.60	3.08	.50	....	....	....	....	....	....	....
				3	....	18.90	81.10	....	.52	....	....	....	....	....	....	....
Same (face of room 3, off right entry 1; 2,600 feet southeast of mine mouth, 4-foot 5-inch bed, 4-foot 1-inch cut).	382	14334	A	1	2.34	17.40	75.86	4.40	.53	....	....	....	....	1.4	....	....
				2	....	17.82	77.67	4.51	.54	....	....	....	....	....	....	....
				3	....	18.66	81.34	....	.57	....	....	....	....	....	....	....
Same (composite of samples 14332, 14333, and 14334).	382	14335	A	1	2.71	17.57	76.15	3.57	.56	4.56	84.52	1.51	5.28	1.8	8,176	14,717
				2	....	18.06	78.27	3.67	.58	4.38	86.88	1.55	2.94	....	8,404	15,127
				3	....	18.75	81.25	....	.60	4.55	90.19	1.61	3.05	....	8,724	15,703
About ¾ mile northeast of; Slab Fork No. 4 mine, Beckley bed (face of main entry, 1,450 feet northwest of mine mouth; 4-foot ¼-inch bed, 3-foot 9½-inch cut).	381	14336	A	1	2.22	18.21	77.23	2.34	.54	4.54	86.47	1.51	4.60	1.8	8,343	15,017
				2	....	18.62	78.99	2.39	.55	4.39	88.43	1.54	2.70	......	8,532	15,358
				3	....	19.08	80.92	....	.56	4.50	90.60	1.58	2.76	....	8,741	15,734
½ mile south of; Slab Fork No. 5 mine, Beckley bed (face of line heading (main), 3,200 feet southwest of mine mouth, 4-foot 5-inch bed, 3-foot 10-inch cut).	382	14313	A	1	3.50	17.71	75.35	3.44	.47	....	....	....	....	3.0	....	....
				2	....	18.35	78.09	3.56	.49	....	....	....	....	....	....	....
				3	....	19.03	80.97	....	.51	....	....	....	....	....	....	....
Same (face of room 15, off left entry 6; 2,000 feet southwest of mine mouth, 3-foot 7-inch bed, 3-foot 6½-inch cut).	392	14314	A	1	3.10	17.88	76.41	2.61	.48	....	....	....	....	2.5	....	....
				2	....	18.45	78.86	2.69	.50	....	....	....	....	....	....	....
				3	....	18.96	81.04	....	.51	....	....	....	....	....	....	....
Same (last cut-through off left entry 3; 1,900 feet southeast of mine mouth, 3-foot 7½-inch bed and cut).	382	14315	A	1	3.41	17.45	74.64	4.50	.46	....	....	....	....	2.8	....	....
				2	....	18.07	77.27	4.66	.48	....	....	....	....	....	....	....
				3	....	18.95	81.05	....	.50	....	....	....	....	....	....	....

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
RALEIGH COUNTY—continued.																
Slab Fork, ¼ mile south of; Slab Fork No. 5 mine, Beckley bed (pillar of room 10, off left entry 2; 800 feet south of mine mouth; 4-foot ½-inch bed, 3-foot 8½-inch cut).	382	14316	A	1	3.10	17.87	76.91	2.62	.49	......	......	......	......	2.8	8,234	14,821
				2	......	17.93	79.37	2.70	.51	......	......	......	......	......	8,497	15,295
				3	......	18.43	81.57	......	.52	......	......	......	......	......	8,732	15,718
Same (composite of samples 14313-14316)	382	14317	A	1	3.33	17.35	76.03	3.29	.47	4.80	84.37	1.58	5.49	2.7	8,169	14,704
				2	......	17.95	78.65	3.40	.49	4.58	87.27	1.63	2.63	......	8,450	15,210
				3	......	18.58	81.42	......	.51	4.74	90.34	1.69	2.72	......	8,747	15,745
Same (pillar of left entry 4, between rooms 13 and 14; 1,250 feet south of the mine mouth; 3-foot 9½-inch bed and cut).	382	14318	A	1	3.70	17.24	76.19	2.87	.48	4.74	84.59	1.53	5.79	2.9	8,149	14,668
				2	......	17.90	79.12	2.98	.50	4.50	87.84	1.59	2.59	......	8,462	15,232
				3	......	18.45	81.55	......	.52	4.64	90.54	1.64	2.66	......	8,722	15,700
Winding Gulf, ¼ mile south of; Lynwin mine, Beckley bed (break-through to air course at left entry 6; 2,600 feet from mine mouth; 5-foot 4-inch bed, 5-foot cut).	383	14319	A	1	3.16	17.28	75.96	3.60	.70	......	......	......	......	2.4	......	......
				2	......	17.84	78.44	3.72	.72	......	......	......	......	......	......	......
				3	......	18.53	81.47	......	.75	......	......	......	......	......	......	......
Same (face of right heading 6; 2,800 feet from mine mouth, 4-foot 9-inch bed; 4-foot 4-inch cut).	383	14320	A	1	2.73	17.98	75.75	3.54	.71	......	......	......	......	1.9	......	......
				2	......	18.49	77.87	3.64	.73	......	......	......	......	......	......	......
				3	......	19.19	80.81	......	.76	......	......	......	......	......	......	......
Same (face of room 2, off right heading 8; 3,300 feet from mine mouth; 5-foot 2-inch bed, 4-foot 9-inch cut).	383	14321	A	1	2.73	17.08	75.53	4.66	.96	......	......	......	......	1.9	......	......
				2	......	17.56	77.65	4.79	.99	......	......	......	......	......	......	......
				3	......	18.44	81.56	......	1.04	......	......	......	......	......	......	......
Same (face of right heading 8, about 3,400 feet from mine mouth, 5-foot 2-inch bed, 4-foot 7-inch cut).	383	14322	A	1	2.75	17.49	74.50	5.26	.66	......	......	......	......	2.0	......	......
				2	......	17.98	76.61	5.41	.68	......	......	......	......	......	......	......
				3	......	19.01	80.99	......	.72	......	......	......	......	......	......	......
Same (face of main heading; 3,200 feet from mine mouth, 5-foot 3-inch bed, 4-foot 8½-inch cut).	383	14323	A	1	3.15	17.27	75.58	4.00	.75	......	......	......	......	2.4	......	......
				2	......	17.83	78.04	4.13	.77	......	......	......	......	......	......	......
				3	......	18.60	81.40	......	.80	......	......	......	......	......	......	......
Same (composite of samples 14319-14323)	383	14324	A	1	2.83	17.73	75.23	4.21	.81	4.56	84.03	1.53	4.86	2.1	8,088	14,558
				2	......	18.25	77.42	4.33	.83	4.37	86.48	1.57	2.42	......	8,323	14,981
				3	......	19.07	80.93	......	.87	4.57	90.39	1.64	2.53	......	8,699	15,658

Winding Gulf No. 1 mine, Beckley bed (face of at; room 14, off left entry 1, off main right heading, 4-foot 4-inch bed, 4-foot 1-inch cut).	384	14368	A	1	2.68	17.71	76.62	2.99	.45	......	......	......	......	2.0	......	......
				2	......	18.20	78.78	3.07	.46	......	......	......	......	......	......	......
				3	......	18.78	81.22	......	.47	......	......	......	......	......	......	......
Same (face of air course of west entry, 4-foot 4-inch bed, 4-foot 1-inch cut).	384	14369	A	1	2.53	18.68	76.17	2.62	.46	......	......	......	......	1.9	......	......
				2	......	19.17	78.14	2.69	.47	......	......	......	......	......	......	......
				3	......	19.70	80.30	......	.48	......	......	......	......	......	......	......
Same (face of main right heading, 4-foot 9-inch bed, 4-foot 5-inch cut).	384	14370	A	1	3.38	16.79	76.56	3.27	.46	......	......	......	......	2.7	......	......
				2	......	17.38	79.24	3.38	.48	......	......	......	......	......	......	......
				3	......	17.99	82.01	......	.50	......	......	......	......	......	......	......
Same (composite of samples 14368, 14369, and 14370).	384	14371	A	1	2.87	17.77	76.44	2.92	.48	4.65	84.95	1.49	5.51	2.2	8,253	14,855
				2	......	18.29	78.70	3.01	.49	4.46	87.46	1.53	3.05	......	8,496	15,293
				3	......	18.86	81.14	......	.51	4.60	90.17	1.58	3.14	......	8,759	15,766
1 mile from; Winding Gulf No. 2 mine, Beckley bed (pillar at break-through to right heading, 4-foot 1-inch bed, 3-foot 9-inch cut).	384	14372	A	1	1.55	17.86	77.79	2.80	.50	......	......	......	......	.9	......	......
				2	......	18.14	79.02	2.84	.51	......	......	......	......	......	......	......
				3	......	18.67	81.33	......	.52	......	......	......	......	......	......	......
Same (face of left heading 2, 4-foot 11-inch bed, 4-foot 8-inch cut).	384	14373	A	1	2.61	17.27	77.47	2.65	.49	......	......	......	......	1.9	......	......
				2	......	17.73	79.55	2.72	.50	......	......	......	......	......	......	......
				3	......	18.23	81.77	......	.51	......	......	......	......	......	......	......
Same (composite of samples 14372 and 14373).....	384	14374	A	1	2.17	17.39	77.77	2.67	.50	4.69	86.28	1.51	4.35	1.4	8,326	14,987
				2	......	17.78	79.49	2.73	.51	4.55	88.20	1.54	2.47	......	8,511	15,320
				3	......	18.28	81.72	......	.52	4.68	90.68	1.58	2.54	......	8,750	15,750
Wood Bay; McAlpin No. 1 mine, Beckley bed (face of second panel, off left heading 3, off second main entry, 5-foot 4½-inch bed and cut).	385	14301	A	1	1.91	17.51	75.17	5.41	.75	......	......	......	......	1.3	......	......
				2	......	17.85	76.63	5.52	.76	......	......	......	......	......	......	......
				3	......	18.89	81.11	......	.80	......	......	......	......	......	......	......
Same (face of second main air course, 5-foot 3-inch bed and cut).	385	14302	A	1	2.58	17.78	74.64	5.00	.53	......	......	......	......	1.8	......	......
				2	......	18.25	76.62	5.13	.54	......	......	......	......	......	......	......
				3	......	19.24	80.76	......	.57	......	......	......	......	......	......	......
Same (face of room 10, second panel, left entry 2, 6-foot 3-inch bed and cut).	385	14303	A	1	2.35	17.75	75.55	4.35	.60	......	......	......	......	1.7	......	......
				2	......	18.18	77.37	4.45	.61	......	......	......	......	......	......	......
				3	......	19.03	80.97	......	.64	......	......	......	......	......	......	......
Same (face of room 3, first panel, left entry 3, 5-foot 3½-inch bed and cut).	385	14304	A	1	2.39	18.40	75.66	3.55	.62	......	......	......	......	1.8	......	......
				2	......	18.85	77.51	3.64	.64	......	......	......	......	......	......	......
				3	......	19.56	80.44	......	.66	......	......	......	......	......	......	......
Same (pillar of main heading 1 between right entries 3 and 4, 4-foot 6½-inch bed, 4-foot 2½-inch cut).	385	14305	A	1	2.05	17.96	77.23	2.76	.52	......	......	......	......	1.4	......	......
				2	......	18.34	78.84	2.82	.53	......	......	......	......	......	......	......
				3	......	18.87	81.13	......	.55	......	......	......	......	......	......	......
Same (face of second panel, off right heading 2, 4-foot 3½-inch bed, 3-foot 11-inch cut).	385	14306	A	1	1.65	17.80	77.72	2.83	.52	......	......	......	......	1.1	......	......
				2	......	18.10	79.02	2.88	.53	......	......	......	......	......	......	......
				3	......	18.64	81.36	......	.55	......	......	......	......	......	......	......
Same (face of fourth panel, off right heading 2, 4-foot 3½-inch bed, 4-foot cut).	385	14307	A	1	1.87	17.73	78.06	2.34	.62	......	......	......	......	1.3	......	......
				2	......	18.07	79.55	2.38	.63	......	......	......	......	......	......	......
				3	......	18.51	81.49	......	.65	......	......	......	......	......	......	......
Same (pillar of air course, near right entry 2, 4-foot 4½-inch bed, 4-foot cut).	385	14308	A	1	2.26	17.92	77.40	2.42	.56	......	......	......	......	1.6	......	......
				2	......	18.33	79.19	2.48	.57	......	......	......	......	......	......	......
				3	......	18.80	81.20	......	.58	......	......	......	......	......	......	......
Same (composite of samples 14301-14308).........	386	14309	A	1	2.20	17.23	77.05	3.52	.59	4.61	84.87	1.58	4.83	1.5	8,188	14,738
				2	......	17.62	78.78	3.60	.60	4.47	86.78	1.62	2.93	......	8,372	15,070
				3	......	18.28	81.72	......	.62	4.64	90.02	1.68	3.04	......	8,684	15,631

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WEST VIRGINIA—Continued.																
RANDOLPH COUNTY.																
Mill Creek, 6 miles south of; Tolbert and Spiker mine, Sewell (?) bed (in entry, about 100 feet north of mine mouth, about 40 feet from outcrop, 4-foot bed, 3-foot 7-inch cut).	386	13561	B	1	5.03	25.45	58.67	10.85	.49	......	......	......	......	3.4	......	......
				2	......	26.80	61.77	11.43	.52	......	......	......	......	......	......	......
				3	......	30.26	69.74	......	.59	......	......	......	......	......	......	......
Spruce, 5 miles from, and about 9 miles southwest of Cheat Bridge; Hopkins mine, Sewell (?) bed (in mine, 1,075 feet southwest of mine mouth, 4-foot 7-inch bed).	386	13601	B	1	2.72	28.71	55.10	13.47	.96	4.74	73.22	1.09	6.52	1.9	7,192	12,946
				2	......	29.51	56.64	13.85	.99	4.56	75.27	1.12	4.21	......	7,393	13,307
				3	......	34.26	65.74	......	1.15	5.29	87.37	1.30	4.89	......	8,582	15,448
WYOMING.																
CONVERSE COUNTY.																
Douglas, 50 miles north of; sec. 33, T. 40 N., R. 72 W., outcrop, Lower Burned bed (6-foot 5-inch bed, 5-foot 7-inch cut).	387	*11442	B	1	27.44	30.91	33.20	8.45	.36	......	......	......	......	19.6	4,372	7,870
				2	......	42.60	45.75	11.65	.50	......	......	......	......	......	6,025	10,845
				3	......	48.22	51.78	......	.57	......	......	......	......	......	6,820	12,276
Glenrock, 20 miles north of; sec. 5, T. 37 N., R. 75 W., country bank (face of pit, 12-foot 8-inch bed, 5-foot cut).	387	*11048	B	1	28.08	31.63	35.67	4.62	.45	6.55	48.32	.69	39.37	14.3	4,639	8,350
				2	......	43.98	49.60	6.42	.63	4.77	67.18	.96	20.04	......	6,450	11,610
				3	......	47.00	53.00	......	.67	5.10	71.79	1.03	21.41	......	6,892	12,406
Inez, 1½ miles south of; sec. 16, T. 33 N., R. 73 W.; Diamond mine (face of entry, 450 feet from mine mouth, 4-foot 8-inch bed and cut).	387	*14781	B	1	25.55	32.73	30.33	11.39	1.25	......	......	......	......	11.2	4,370	7,866
				2	......	43.96	40.74	15.30	1.68	......	......	......	......	......	5,870	10,566
				3	......	51.90	48.10	......	1.98	......	......	......	......	......	6,930	12,474
1 mile south of; NW. ¼ sec. 16, T. 33 N. (?), R. 73 W.; Inez mine (at end of entry, 400 feet northwest of end of slope which is 120 feet northeast of mine mouth, 7-foot 8-inch bed, 5-foot 10-inch cut).	388	*14754	B	1	27.88	32.10	31.70	8.32	.71	6.35	45.34	.73	38.55	15.7	4,328	7,790
				2	......	44.51	43.95	11.54	.98	4.51	62.87	1.01	19.09	......	6,001	10,802
				3	......	50.32	49.68	......	1.11	5.10	71.07	1.14	21.58	......	6,784	12,211
Lost Spring, 9 miles west of; sec. 11, T. 32 N., R. 69 W.; Fred Haynes prospect (face of main entry, 50 feet from entry mouth).	388	*10711	B	1	24.80	32.90	36.46	5.84	1.18	......	......	......	......	7.6	4,612	8,302
				2	......	43.75	48.48	7.77	1.57	......	......	......	......	......	6,133	11,039
				3	......	47.43	52.57	......	1.70	......	......	......	......	......	6,649	11,968
17 miles northeast of; sec. 13, T. 34 N., R. 67 W.; Harney Creek mine (at end of main entry, 50 feet from mouth of entry, 5½-foot bed, 4-foot 11-inch cut).	389	*10011	B	1	29.87	28.06	34.71	7.36	.79	......	......	......	......	16.6	4,258	7,664
				2	......	40.01	49.50	10.49	1.13	......	......	......	......	......	6,071	10,928
				3	......	44.70	55.30	......	1.26	......	......	......	......	......	6,782	12,208

2 miles northwest of; sec. 36, T. 33 N., R. 68 W.; Onyon mine (room 15, off north entry 1, 1,800 feet from mine mouth).	389	*10740	B	1	27.91	27.07	36.73	8.29	.89	6.25	47.47	.80	36.30	10.0	4,403	7,927
				2		37.55	50.95	11.50	1.23	4.37	65.85	1.11	15.94		6,109	10,996
				3		42.43	57.57		1.39	4.94	74.40	1.25	18.02		6,903	12,425
35 miles north of; sec. 8, T. 37 N., R. 68 W.; prospect, D bed (face of open cut, 4-foot 4-inch bed).	389	*11447	B	1	27.78	29.38	38.56	4.28	.27					16.8	4,673	8,411
				2		40.68	53.39	5.93	.37						6,471	11,648
				3		43.24	56.76		.39						6,879	12,382
8 miles north of; sec. 35, T. 34 N., R. 68 W.; Rosin mine (face of left entry 1, about 1,000 feet from the mine mouth, 5-foot 1-inch bed, 4-foot 9-inch cut).	390	*10775	B	1	27.65	26.70	35.66	9.99	1.03	6.26	45.75	.68	36.29	15.7	4,338	7,808
				2		36.90	49.29	13.81	1.42	4.42	63.24	.94	16.17		5,996	10,793
				3		42.81	57.19		1.65	5.13	73.37	1.09	18.76		6,957	12,523
8 miles northwest of; Sunset mine (face of main entry, 500 feet from mine mouth, 5-foot 3½-inch bed, 4-foot 8-inch cut).	390	*10835	B	1	24.04	28.44	34.87	12.65	.61					9.7	4,268	7,682
				2		37.44	45.91	16.65	.80						5,619	10,114
				3		44.92	55.08		.96						6,742	12,136
CROOK COUNTY.																
Sundance, 7½ miles west of; SE. ¼ NW. ¼ sec. 24, T. 51 N., R. 64 W.; Belshe mine (end of room 150 feet long turned east from back entry, about 275 feet from mine mouth; 5-foot 3½-inch bed, 4-foot 11¾-inch cut).	391	15713	B	1	20.24	36.35	34.08	9.33	5.25					15.7	5,192	9,346
				2		45.58	42.72	11.70	6.58						6,510	11,718
				3		51.62	48.38		7.45						7,373	13,271
Same (in last crosscut, 48 feet from front entry, crosscut turn 336 feet from mouth of mine, 4-foot 3¼-inch bed, 3-foot 6-inch cut).	391	15714	B	1	19.34	35.35	34.24	11.07	4.38	5.90	51.02	.75	26.88	13.7	5,110	9,198
				2		43.83	42.45	13.72	5.43	4.65	63.25	.93	12.02		6,335	11,403
				3		50.80	49.20		6.29	5.39	73.31	1.08	13.93		7,342	13,216
Same (position and bed same as No. 15714, 4-foot 3¼-inch cut).	391	15715	B	1	17.16	34.08	32.57	16.19	5.18					12.4	4,891	8,804
				2		41.14	39.32	19.54	6.25						5,904	10,627
				3		51.13	48.87		7.77						7,338	13,208
FREEMONT COUNTY.																
Hudson, 1 mile west of; sec. 11, T. 1 S., R. 2 E.; Hickey mine (room 3, off south entry, 300 feet from mine mouth, 3-foot 4-inch bed and cut).	391	*17247	A	1	23.07	33.07	39.66	4.20	.69	6.22	55.13	1.40	32.36	5.6	5,283	9,509
				2		42.99	51.55	5.46	.90	4.76	71.66	1.82	15.40		6,867	12,361
				3		45.47	54.53		.95	5.04	75.80	1.93	16.28		7,264	13,075
1½ miles north of; sec. 2, T. 2 S., R. 2 E.; Indian mine (fifth west back entry, one-fourth mile southwest of mine mouth, 6-foot 6-inch bed, 3-foot 6-inch cut).	392	*13178	B	1	20.88	31.90	39.16	8.06	.51					14.0	5,134	9,241
				2		40.32	49.49	10.19	.64						6,489	11,680
				3		44.90	55.10		.71						7,226	13,007
Same (in main entry, about 900 feet from mouth of mine, 7-foot bed, 5-foot 6-inch cut).	392	*13179	B	1	20.39	33.07	40.69	5.85	.58					11.2	5,307	9,553
				2		41.54	51.11	7.35	.73						6,666	11,999
				3		44.83	55.17		.79						7,195	12,951
Same (face of room 17, off 00 entry, 7-foot 7½-inch bed and cut).	392	*10838	A	1	22.18	31.85	42.53	3.44	.51					9.2		
				2		40.93	54.65	4.42	.66							
				3		42.82	57.18		.69							
Same (face of west entry 4, 7-foot 7½-inch bed, 6-foot 3-inch cut).	392	*10839	A	1	20.98	30.93	43.14	4.95	.44					8.2		
				2		39.14	54.60	6.26	.56							
				3		41.75	58.25		.60							
Same (in room (about No. 30) off west entry 2, 7-foot 10-inch bed, 7-foot 2-inch cut).	392	*10840	A	1	21.28	31.15	40.69	6.88	.52					8.0		
				2		39.57	51.69	8.74	.66							
				3		43.36	56.64		.72							
Same (face of room 4, off east entry 3, 7-foot 2-inch bed, 6-foot 6-inch cut).	392	*10841	A	1	21.20	29.87	41.72	7.21	.52					8.3		
				2		37.91	52.94	9.15	.66							
				3		41.73	58.27		.73							

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WYOMING—Continued.																
FREEMONT COUNTY—continued.																
Hudson, 1½ miles north of; sec. 2, T. 2 S., R. 2, E.; Indian mine (face of room 14½, off west entry 2, 7-foot 6-inch bed and cut).	392	*10843	A	1	20.87	31.97	43.43	3.73	.47	6.21	56.74	1.28	31.62	7.8	5,434	9,781
				2	....	40.40	54.89	4.71	.59	4.92	71.70	1.55	16.53	....	6,867	12,361
				3	....	42.40	57.60	....	.62	5.16	75.24	1.68	17.35	....	7,206	12,971
Same (composite of samples 10838-10841).........	392	*10842	A	1	21.10	31.35	41.75	5.80	.49	6.21	54.33	1.21	31.96	8.4	5,257	9,463
				2	....	39.73	52.92	7.35	.62	4.90	68.86	1.53	16.74	....	6,663	11,993
				3	....	42.88	57.12	....	.67	5.29	74.32	1.65	18.07	....	7,191	12,944
5 miles from; SW. ¼ sec. 22, T. 1 S., R. 2 E ; Mitchell mine (in last room off main slope, 40 feet from slope and 500 feet from mouth of slope, 3-foot 4-inch bed, 3-foot 2½-inch cut).	393	*17248	A	1	19.53	34.69	39.57	6.21	.71	6.05	56.08	1.35	29.65	3.8	5,413	9,743
				2	....	43.11	49.17	7.72	.88	4.82	69.63	1.68	15.27	....	6,727	12,109
				3	....	46.72	53.28	....	.95	5.22	75.46	1.82	16.55	....	7,290	13,122
4 miles from; sec.3, T. 33 N., R.98 W.; Proposia mine, Lander bed (face of north entry 2, 1,800 feet from mine mouth, 17-foot bed, 9-foot 6-inch cut).	393	*17244	A	1	23.38	33.55	38.54	4.53	.66	....	....	....	....	6.7	5,233	9,419
				2	....	43.79	50.30	5.91	.86	....	....	....	....	....	6,830	12,294
				3	....	46.54	53.46	....	.91	....	....	....	....	....	7,259	13,066
Same (face of south entry 5, 1,200 feet from mine mouth, 8-foot bed and cut).	393	*17245	A	1	24.70	31.78	39.52	4.00	.54	....	....	....	....	10.1	5,166	9,299
				2	....	42.20	52.49	5.31	.72	....	....	....	....	....	6,860	12,348
				3	....	44.57	55.43	....	.76	....	....	....	....	....	7,245	13,041
Same (composite of samples 17244 and 17245).....	394	*17246	A	1	23.95	31.67	40.14	4.24	.65	6.59	54.44	1.21	32.87	8.4	5,209	9,376
				2	....	41.64	52.78	5.58	.85	5.17	71.58	1.59	15.23	....	6,849	12,328
				3	....	44.10	55.90	....	.90	5.48	75.81	1.68	16.13	....	7,254	13,057
HOT SPRINGS COUNTY.																
Crosby, 2 miles from Kirby; sec. 13, T. 44 N., R. 95 W.; Big Horn mine, Gebo bed. (face of room 5 near east rib, off butt entry 3, off west entry 1, 7-foot 6-inch bed and cut).	394	*15202	A	1	16.59	33.64	46.37	3.40	.58	....	....	....	....	8.6	6,044	10,879
				2	....	40.33	55.59	4.08	.70	....	....	....	....	....	7,246	13,043
				3	....	42.04	57.96	....	.73	....	....	....	....	....	7,554	13,597
Same (east rib of room 5, off second butt dip entry, off west entry 2, 50 feet in from entry, 8-foot 5-inch bed, 8-foot 2-inch cut).	394	*15209	A	1	15.98	32.91	43.78	7.33	.62	....	....	....	....	9.7	5,771	10,388
				2	....	39.17	52.11	8.72	.74	....	....	....	....	....	6,869	12,364
				3	....	42.91	57.09	....	.81	....	....	....	....	....	7,525	13,545
Same (face of room 6, off first dip butt entry, off west entry 2, 8-foot 6-inch bed, 8-foot 3-inch cut).	394	*15217	A	1	17.24	31.09	46.70	4.97	.52	....	....	....	....	8.7	5,889	10,606
				2	....	37.57	56.42	6.01	.63	....	....	....	....	....	7,116	12,800
				3	....	39.97	60.03	....	.67	....	....	....	....	....	7,571	13,628

Same (face of room 4, off east entry 2, 45 feet from entry, 5-foot 3-inch bed and cut).	394	*15219	A	1	16.40	33.78	46.14	3.68	.62	......	......	......	......	9.0	6,029	10,852
				2	......	40.41	55.19	4.40	.74	......	......	......	......	......	7,212	12,982
				3	......	42.27	57.73	......	.77	......	......	......	......	......	7,544	13,579
Same (composite of samples 15202, 15209, 15217, and 15219).	394	*15237	A	1	16.56	33.11	45.49	4.84	.59	6.04	60.64	1.33	26.56	9.0	5,926	10,667
				2	......	39.68	54.52	5.80	.71	5.03	72.68	1.59	14.19	......	7,102	12,784
				3	......	42.12	57.88	......	.75	5.34	77.16	1.69	15.06	......	7,539	13,570
Same (position, bed, and cut same as sample 15202).	394	*15201	A	1	16.34	33.29	46.89	3.48	.58	......	......	......	......	8.6	6,074	10,933
				2	......	39.79	56.05	4.16	.69	......	......	......	......	......	7,260	13,068
				3	......	41.52	58.48	......	.72	......	......	......	......	......	7,575	13,635
Same (position, bed, and cut same as sample 15209).	394	*15210	A	1	16.08	33.00	43.86	7.06	.57	......	......	......	......	10.2	5,789	10,420
				2	......	39.32	52.27	8.41	.68	......	......	......	......	......	6,898	12,416
				3	......	42.93	57.07	......	.74	......	......	......	......	......	7,531	13,556
Same (position, bed, and cut same as sample 15217).	394	*15216	A	1	17.07	32.54	45.66	4.73	.53	......	......	......	......	8.5	5,904	10,627
				2	......	39.24	55.06	5.70	.64	......	......	......	......	......	7,119	12,814
				3	......	41.61	58.39	......	.68	......	......	......	......	......	7,549	13,588
Same (position, bed, and cut same as sample 15219).	394	*15218	A	1	16.46	33.87	45.97	3.70	.60	......	......	......	......	9.2	6,036	10,865
				2	......	40.54	55.03	4.43	.72	......	......	......	......	......	7,225	13,005
				3	......	42.42	57.58	......	.75	......	......	......	......	......	7,560	13,608
Same (composite of samples 15201, 15210, 15216, and 15218).	395	*15236	A	1	16.48	32.91	45.83	4.78	.58	5.97	61.08	1.35	26.24	9.1	5,972	10,750
				2	......	39.40	54.88	5.72	.69	4.96	73.13	1.62	13.88	......	7,150	12,870
				3	......	41.79	58.21	......	.73	5.26	77.57	1.72	14.72	......	7,584	13,651
Same (pillar between rooms 6 and 7, off butt entry 2, off west entry 1, 8-foot 1-inch bed and cut).	394	*15214	A	1	17.20	32.31	46.61	3.88	.61	......	......	......	......	10.2	5,971	10,748
				2	......	39.02	56.29	4.69	.74	......	......	......	......	......	7,211	12,980
				3	......	40.94	59.06	......	.78	......	......	......	......	......	7,566	13,619
Same (position, bed, and cut same as sample 15214).	394	*15215	A	1	17.01	32.54	46.70	3.75	.61	......	......	......	......	9.7	5,970	10,746
				2	......	39.21	56.27	4.52	.74	......	......	......	......	......	7,194	12,949
				3	......	41.06	58.94	......	.78	......	......	......	......	......	7,534	13,561
Same (slack coal, sampled at the mine)	395	*15203	C	1	17.47	32.08	43.41	7.04	.72	......	......	......	......	11.5	5,635	10,143
				2	......	38.87	52.60	8.53	.87	......	......	......	......	......	6,828	12,290
				3	......	42.50	57.50	......	.95	......	......	......	......	......	7,465	13,437
Same (coal from same pile as sample 15203)	395	*15204	C	1	17.30	32.63	43.24	6.83	.70	......	......	......	......	10.9	5,643	10,157
				2	......	39.46	52.28	8.26	.85	......	......	......	......	......	6,824	12,283
				3	......	43.01	56.99	......	.93	......	......	......	......	......	7,438	13,388
Same (slack taken from 17 wagonloads)	395	*15205	C	1	17.60	31.82	44.03	6.55	.69	......	......	......	......	11.5	5,648	10,166
				2	......	38.62	53.43	7.95	.84	......	......	......	......	......	6,854	12,337
				3	......	41.96	58.04	......	.91	......	......	......	......	......	7,446	13,403
Same (slack taken from 20 wagonloads)	395	*15207	C	1	17.88	32.15	42.46	7.51	.67	......	......	......	......	12.1	5,580	10,044
				2	......	39.15	51.71	9.14	.82	......	......	......	......	......	6,795	12,231
				3	......	43.09	56.91	......	.90	......	......	......	......	......	7,479	13,462
Same (slack taken from 18 wagonloads)	395	*15211	C.	1	17.80	31.36	43.18	7.66	.68	......	......	......	......	11.8	5,589	10,060
				2	......	38.15	52.53	9.32	.83	......	......	......	......	......	6,799	12,238
				3	......	42.07	57.93	......	.92	......	......	......	......	......	7,498	13,496
Same (composite of samples 15205, 15207, and 15211).	395	*15234	C.	1	17.85	31.57	43.37	7.21	.64	6.06	57.70	1.26	27.13	11.8	5,610	10,098
				2	......	38.43	52.79	8.78	.78	4.97	70.24	1.53	13.70	......	6,829	12,292
				3	......	42.13	57.87	......	.86	5.45	77.00	1.68	15.01	......	7,487	13,477
Same (slack taken from same wagons as sample 15205).	395	*15206	C.	1	17.75	32.06	44.07	6.12	.70	......	......	......	......	12.0	5,673	10,211
				2	......	38.98	53.58	7.44	.85	......	......	......	......	......	6,897	12,415
				3	......	42.11	57.89	......	.92	......	......	......	......	......	7,452	13,414
Same (slack taken from same wagons as sample 15207).	395	*15208	C.	1	17.81	32.30	41.86	8.03	.70	......	......	......	......	12.0	5,541	9,974
				2	......	39.30	50.93	9.77	.85	......	......	......	......	......	6,742	12,136
				3	......	43.56	56.44	......	.94	......	......	......	......	......	7,472	13,450

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WYOMING—Continued.																
HOT SPRINGS COUNTY—continued.																
Crosby, 2 miles from Kirby; sec. 13, T. 44 N., R. 95 W.; Big Horn mine, Gebo bed (slack taken from same wagons as sample 15211).	395	*15212	C.	1	17.88	31.52	42.38	8.22	.65					11.7	5,557	10,003
				2		38.38	51.61	10.01	.79						6,767	12,181
				3		42.65	57.35		.88						7,519	13,534
Same (composite of samples 15206, 15208, and 15212).	395	*15235	C.	1	17.87	31.26	43.48	7.39	.66	5.99	57.41	1.28	27.27	11.9	5,590	10,062
				2		38.06	52.94	9.00	.80	4.87	69.90	1.56	13.87		6,806	12,251
				3		41.82	58.18		.88	5.35	76.81	1.71	15.25		7,479	13,462
Kirby, 2 miles from; sec. 11, T. 44 N.; R. 95 W.; Gebo mine, Gebo bed (face of east entry 4; 225 feet from butt entry 2; 7-foot 10-inch bed and cut).	395	*15277	A.	1	16.66	31.88	47.03	4.43	.62					10.5	6,005	10,809
				2		38.25	56.43	5.32	.74						7,205	12,969
				3		40.40	59.60		.78						7,610	13,698
Same (face of room 8, off second east butt entry, off entry 5, 8-foot 9-inch bed and cut).	396	*15278	A.	1	15.46	33.50	47.58	3.46	.66					7.6	6,189	11,140
				2		39.63	56.28	4.09	.78						7,321	13,178
				3		41.32	58.68		.81						7,633	13,739
Same (face of room 7, off first east butt entry, off fifth level, 9-foot 8-inch bed and cut).	396	*15279	A.	1	15.27	33.64	46.73	4.36	.56					9.1	6,106	10,991
				2		39.70	55,15	5.15	.66						7,206	12,971
				3		41.86	58.14		.70						7,597	13,675
Same (face of east entry 3; 400 feet from butt entry 2; 9-foot 2-inch bed and cut).	396	*15280	A.	1	16.30	33.57	46.95	3.18	.52					9.6	6,098	10,976
				2		40.11	56.09	3.80	.62						7,285	13,113
				3		41.69	58.31		.64						7,573	13,631
Same (composite of samples 15277, 15278, 15279, and 15280).	396	*15283	A.	1	15.86	33.01	47.39	3.74	.59	6.06	62.03	1.29	26.29	9.2	6,102	10,984
				2		39.23	56.33	4.44	.70	5.11	73.72	1.53	14.50		7,252	13,054
				3		41.05	58.95		.73	5.35	77.15	1.60	15.17		7,589	13,660
Same (face of west entry 3; 300 feet from main slope; 7-foot 11-inch bed and cut).	396	*15281	A.	1	16.35	32.93	47.72	3.00	.43					8.2	6,120	11,016
				2		39.37	57.04	3.59	.51						7,316	13,169
				3		40.83	59.17		.53						7,588	13,658
Same (barrier pillar between room 1 west side and back entry, off second level, 6-foot 9-inch bed and cut).	396	*15282	A.	1	17.06	32.10	47.75	3.09	.43					10.0	6,043	10,877
				2		38.70	57.57	3.73	.52						7,286	13,115
				3		40.20	59.80		.54						7,568	13,622
Same (pillar between west entry 2 and back entry, 375 feet beyond butt entry 3, about 1,750 feet from main slope, 5-foot 9-inch bed and cut).	396	*15342	A	1	16.64	32.92	46.68	3.76	.46					8.3	6,005	10,809
				2		39.49	56.00	4.51	.55						7,204	12,967
				3		41.35	58.65		.58						7,544	13,579

Same (position, bed, and cut same as No. 15342).	396	*15343	A	1	16.68	33.23	46.34	3.75	.43					8.2	5,995	10,791
				2		39.88	55.62	4.50	.52						7.195	12,951
				3		41.76	58.24		.54						7,534	13,561
Same (pillar between rooms 1 and 2, off first west butt entry, off east entry 2, about 400 feet from main slope; 6-foot 1-inch bed and cut).	396	*15344	A	1	17.36	33.91	45.90	2.83	.47					8.7	6,019	10,834
				2		41.03	55.55	3.42	.57						7,284	13,111
				3		42.48	57.52		.59						7,542	13,576
Same (position, bed, and cut same as No. 15344).	396	*15345	A	1	17.27	33.11	46.87	2.75	.46					9.1	6,035	10,863
				2		40.02	56.66	3.32	.56						7,295	13,131
				3		41.39	58.61		.58						7,545	13,581
Same (pillar between pitch rooms 10 and 11, off west entry 1, off butt entry 2; 6-foot 5-inch bed and cut).	396	*15346	A	1	16.55	33.40	46.82	3.23	.42					6.9	6,012	10,822
				2		40.02	56.11	3.87	.50						7,204	12,967
				3		41.63	58.37		.52						7,494	13,489
Same (position, bed, and cut same as No. 15346).	396	*15347	A	1	16.63	33.27	46.96	3.14	.42					6.9	6,029	10,852
				2		39.91	56.32	3.77	.50						7,232	13,018
				3		41.47	58.53		.52						7,515	13,527
Same (near face on pillar between west entry 1 and back entry, 2,100 feet from main slope, 6-foot 7-inch bed and cut).	396	*15348	A	1	16.87	32.83	46.77	3.53	.52					6.5	5,950	10,710
				2		39.49	56.26	4.25	.63						7,157	12,883
				3		41.24	58.76		.66						7,475	13,455
Same (position, bed, and cut same as No. 15348).	396	*15349	A	1	16.91	33.01	46.71	3.37	.51					7.3	5,960	10,728
				2		39.73	56.21	4.06	.61						7,173	12,911
				3		41.41	58.59		.64						7,476	13,457
Same (composite of 6 samples of slack taken from cars while being loaded).	396	*15475	C	1	17.47	31.99	45.29	5.25	.66	6.10	60.06	1.32	26.61	10.3	5,870	10,566
				2		38.76	54.88	6.36	.80	5.04	72.77	1.60	13.43		7,113	12,803
				3		41.39	58.61		.85	5.38	77.71	1.71	14.35		7,596	13,673
Same (composite of 6 samples of slack taken from cars while being loaded)	396	*15482	C	1	17.08	32.51	45.36	5.05	.62	6.07	60.71	1.29	26.26	10.7	5,876	10,577
				2		39.21	54.70	6.09	.75	5.03	73.22	1.56	13.35		7,086	12,755
				3		41.75	58.25		.80	5.36	77.96	1.66	14.22		7,545	13,581
Same (slack from a car shipped to Bureau of Mines station at Pittsburgh for testing purposes).	396	*15687	C	1	16.58	31.97	46.05	5.40	.64	5.93	60.18	1.33	26.52	9.3	5,894	10,609
				2		38.33	55.20	6.47	.77	4.90	72.14	1.59	14.13		7,066	12,719
				3		40.98	59.02		.82	5.24	77.13	1.70	15.11		7,555	13,599
Same (slack from a car shipped to Bureau of Mines station at Pittsburgh for testing purposes).	396	*16102	C	1	16.67	33.30	45.24	4.79	.69	6.01	60.62	1.36	26.53	11.3	5,883	10,589
				2		39.96	54.29	5.75	.83	4.99	72.74	1.63	14.06		7,060	12,708
				3		42.40	57.60		.88	5.29	77.18	1.73	14.92		7,491	13,484
Meeteetse, 18 miles southeast of; sec. 2, T. 46 N., R. 99 W.; Dickie No. 1 prospect (east rib of west entry, 25 feet from mouth, 4-foot bed and cut).	397	14679	B	1	14.75	37.77	38.08	9.40	.70					0.9	5,587	10,057
				2		44.30	44.67	11.03	.82						6,554	11,797
				3		49.79	50.21		.92						7,367	13,261
19 miles southeast of; E. ½ sec. 26, T. 46 N., R. 99 W.; Mayfield No. 2 prospect (face of new drift entry, 30 feet from entry mouth, 30-foot bed, 6-foot-cut).	397	*14756	B	1	10.67	37.99	43.99	7.35	.35					3.4	6,151	11,072
				2		42.53	49.24	8.23	.39						6,885	12,393
				3		46.34	53.66		.42						7,503	13,505
JOHNSON COUNTY.																
Buffalo, 18 miles southeast of; prospect, B bed (10-foot 9½-inch bed, 3-foot 4-inch cut).	398	10683	B	1	28.81			6.67	.56					19.6	4,406	7,931
				2				9.37	.79						6,189	11,140
				3					.87						6,829	12,292
Casper, 75 miles north of; sec. 23, T. 44 N., R. 81 W.; prospect (6-foot 3-inch bed, 42-inch cut).	398	10728	B	1	23.61			7.76	.68					11.8	4,692	8,446
				2				10.16	.89						6,141	11,054
				3					.99						6,836	12,305
60 miles north of; sec. 36, T. 43 N., R. 79 W.; prospect (13-foot 11-inch bed, 43-inch cut)	399	*10804	B	1	18.82	35.75	37.88	7.55	.57					8.1	5,087	9,157
				2		44.04	46.66	9.80	.70						6,266	11,279
				3		48.55	51.45		.77						6,908	12,434

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WYOMING—Continued.																
JOHNSON COUNTY—continued.																
Casper, 50 miles north of; prospect (3-foot 10-inch bed, 26-inch cut).	399	*11059	B	1	16.83	31.40	36.78	14.99	1.54	5.53	48.93	1.00	28.01	3.7	4,711	8,480
				2	......	37.76	44.22	18.02	1.85	4.40	58.83	1.20	15.70	......	5,664	10,195
				3	......	46.06	53.94	......	2.26	5.37	71.76	1.46	19.15	......	6,908	12,434
65 miles north of; sec. 20, T. 42 N., R. 77 W., Puggsley mine (20-foot 5-inch bed, 60-inch cut).	400	*10827	B	1	23.46	35.64	35.73	5.17	.49	6.50	51.24	.70	35.90	9.4	5,028	9,050
				2	......	46.56	46.69	6.75	.64	5.08	66.95	.91	19.67	......	6,569	11,824
				3	......	49.93	50.07	......	.69	5.45	71.79	.98	21.09	......	7,044	12,679
Kaycee, 6 miles northeast of; abandoned mine (exposed face in mine, entire section of bed).	400	10694	B	1	......	......	......	14.59	.56	......	......	......	......	10.4	4,431	7,976
				2	......	......	......	......	......	......	......	......	......	......	......	......
				3	......	......	......	......	......	......	......	......	......	......	......	......
NATRONA COUNTY.																
Casper, 25 miles southwest of; sec. 14, T. 31 N., R. 82 W., Red Ash mine (face of room 2, off main entry, 300 feet from entry mouth, 6-foot 3-inch bed).	400	*10671	B	1	29.50	27.48	37.66	5.36	.47	......	......	......	......	19.7	4,388	7,898
				2	......	38.98	53.42	7.60	.67	......	......	......	......	......	6,224	11,203
				3	......	42.19	57.81	......	.73	......	......	......	......	......	6,736	12,125
PARK COUNTY.																
Cody, 9 miles southeast of; NE. ¼ sec. 17, T. 52 N., R. 100 W.; McGuffey mine (east wall of room, 20 feet above entry 3 and 70 feet east of slope, 3-foot 1-inch bed, 2-foot 8-inch cut).	401	* 12886	B	1	15.52	35.95	40.51	8.02	.91	......	......	......	......	5.8	5,668	10,202
				2	......	42.55	47.96	9.49	1.08	......	......	......	......	......	6,710	12,078
				3	......	47.01	52.99	......	1.19	......	......	......	......	......	7,414	13,345
Meeteetse, 4 miles north of, and 32 miles southeast of Cody; sec. 21, T. 49 N., R. 100 W.; Black Diamond mine (face of room, 30 feet above entry 6 and 200 feet west of slope, 3-foot 6-inch bed and cut).	401	* 12887	B	1	17.80	30.89	43.50	7.81	.20	......	......	......	......	7.3	5,328	9,590
				2	......	37.58	52.92	9.50	.24	......	......	......	......	......	6,482	11,668
				3	......	41.53	58.47	......	.27	......	......	......	......	......	7,168	12,898
3½ miles southwest of, and 40 miles south of Cody; SE. ¼ sec. 13, T. 48 N., R. 101 W.; Greybull or Wilson mine; breast of room, 200 feet west of slope	401	* 12888	B	1	14.21	35.81	37.65	12.33	.92	......	......	......	......	5.5	5,335	9,603
				2	......	41.74	43.89	14.37	1.07	......	......	......	......	......	6,218	11,192
				3	......	48.74	51.26	......	1.25	......	......	......	......	......	7,261	13,070

and 400 feet from portal, 5-foot 7-inch bed, 4-foot 6-inch cut.																
Wiley, 13 miles southeast of Cody; SW. ¼ sec. 2, T. 51 N., R. 100 W.; Eagle mine (west wall of entry driven north from bottom of shaft, 6 feet from shaft, 5-foot 8-inch bed, 5-foot 5-inch cut).	402	* 12885	B	1	15.25	32.08	41.07	11.60	.53	......	......	......	......	6.8	5,264	9,475
				2	......	37.85	48.46	13.69	.63	......	......	......	......	......	6,211	11,180
				3	......	43.85	56.15	......	.73	......	......	......	......	......	7,196	12,953
14 miles southeast of Cody; SE. ¼ sec. 10, T. 51 N., R. 100 W.; East Wiley mine (south wall near bottom of slope, 180 feet from mine mouth, 4-foot bed, 3-foot 7-inch cut).	402	* 12884	B	1	15.82	35.40	41.01	7.77	.90	......	......	......	......	6.4	5,674	10,214
				2	......	42.05	48.72	9.23	1.07	......	......	......	......	......	6,740	12,132
				3	......	46.33	53.67	......	1.18	......	......	......	......	......	7,425	13,365
15 miles southeast of Cody; sec. 34, T. 51 N., R. 101 W.; West Wiley (Thompson) mine (face of room, off drift entry, 300 feet from entry mouth; 5-foot 7-inch bed, 3-foot 3-inch cut).	403	* 12883	B	1	14.80	35.16	38.46	11.58	.68	......	......	......	......	4.3	5,285	9,513
				2	......	41.27	45.14	13.59	.80	......	......	......	......	......	6,203	11,165
				3	......	47.76	52.24	......	.93	......	......	......	......	......	7,179	12,922
SHERIDAN COUNTY.																
Acme; sec. 13, T. 57 N., R. 85 W.; Acme No. 1 mine, Carney bed (8-foot cut).	403	* 12687	B	1	21.16	34.00	39.66	5.18	.35	6.01	54.87	1.11	32.48	8.0	5,211	9,380
				2	......	43.13	50.30	6.57	.44	4.64	69.60	1.41	17.34	......	6,610	11,898
				3	......	46.16	53.84	......	.47	4.97	74.49	1.51	18.56	......	7,075	12,735
Same (pillar of room 9, off east entry 5, off main entry, 6-foot 2-inch cut).	403	* 10823	A	1	22.48	33.19	39.75	4.58	.26	......	......	......	......	10.0	......	......
				2	......	42.82	51.27	5.91	.34	......	......	......	......	......	......	......
				3	......	45.51	54.49	......	.36	......	......	......	......	......	......	......
Same (pillar of room 1, off west entry 3, off main north entry, 6-foot 10-inch cut).	403	* 10824	A	1	22.84	31.93	40.64	4.59	.30	......	......	......	......	10.9	......	......
				2	......	41.38	52.67	5.95	.39	......	......	......	......	......	......	......
				3	......	44.00	56.00	......	.41	......	......	......	......	......	......	......
Same (composite of samples 10823 and 10824).....	403	* 10825	A	1	22.57	32.53	40.36	4.55	.30	6.30	53.43	1.09	34.33	10.5	5,121	9,218
				2	......	42.01	52.11	5.88	.39	4.89	69.00	1.41	18.43	......	6,614	11,905
				3	......	44.64	55.36	......	.41	5.20	73.31	1.50	19.58	......	7,027	12,649
Same (on pillar near face of west entry 3, off main north entry, 7-foot cut).	403	* 10826	A	1	22.08	33.43	40.11	4.38	.44	6.32	53.17	.96	34.73	9.4	5,142	9,256
				2	......	42.90	51.48	5.62	.56	4.97	68.24	1.23	19.38	......	6,599	11,878
				3	......	45.45	54.55	......	.59	5.27	72.30	1.30	20.54	......	6,992	12,586
Sec. 13, T. 57 N., R. 85 W.; Acme No. 2 mine; Carney and Masters bed (in mine 2,000 feet northwest of mine mouth, 8-foot 6-inch bed, 7-foot cut).	404	* 12683	B	1	23.88	34.11	38.20	3.81	.33	6.28	53.68	1.19	34.71	9.5	5,168	9,302
				2	......	44.81	50.18	5.01	.43	4.77	70.52	1.56	17.71	......	6,789	12,220
				3	......	47.17	52.83	......	.45	5.02	74.24	1.64	18.65	......	7,147	12,865
Same (face of room 13, off south entry 2, off west entry 3).	404	* 10807	A	1	23.04	33.39	39.99	3.58	.37	......	......	......	......	9.4	......	......
				2	......	43.39	51.96	4.65	.48	......	......	......	......	......	......	......
				3	......	45.51	54.49	......	.50	......	......	......	......	......	......	......
Same (face of south entry 5, off west entry 3).....	404	* 10808	A	1	22.21	32.63	41.66	3.50	.37	......	......	......	......	8.2	......	......
				2	......	41.95	53.55	4.50	.48	......	......	......	......	......	......	......
				3	......	43.93	56.07	......	.52	......	......	......	......	......	......	......
Same (face of room 1, off north entry 1, off west entry 5).	404	* 10809	A	1	23.14	32.48	41.33	3.05	.33	......	......	......	......	9.1	......	......
				2	......	42.27	53.76	3.97	.43	......	......	......	......	......	......	......
				3	......	44.02	55.98	......	.45	......	......	......	......	......	......	......
Same (composite of samples 10807, 10808, and 10809).	404	* 10810	A	1	22.72	32.17	41.74	3.37	.36	6.47	54.19	.99	34.62	8.9	5,191	9,344
				2	......	41.63	54.01	4.36	.47	5.11	70.12	1.28	18.66	......	6,717	12,091
				3	......	43.53	56.47	......	.49	5.34	73.32	1.34	19.51	......	7,023	12,641
Same (pillar of west entry 1)......................	404	* 10811	A	1	24.66	31.75	39.94	3.65	.38	6.48	52.31	1.20	35.98	11.3	5,004	9,007
				2	......	42.14	53.01	4.85	.50	4.96	69.43	1.59	18.67	......	6,642	11,956
				3	......	44.29	55.71	......	.53	5.21	72.97	1.67	19.62	......	6,981	12,566

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WYOMING—Continued.																
SHERIDAN COUNTY—continued.																
Carneyville, just north of; T. 57 N., R. 84 W.; Carney mine, Carney bed (in mine, 1,000 feet northwest of mine mouth, 16-foot bed, 10-foot cut).	405	* 12684	B	1	25.25	32.94	39.06	2.75	.32	6.28	53.49	1.11	36.05	11.6	5,089	9,160
				2	......	44.07	52.25	3.68	.43	4.64	71.56	1.48	18.21	......	6,808	12,254
				3	......	45.75	54.25	......	.45	4.82	74.29	1.54	18.90	......	7,068	12,722
½ mile from Alger; secs. 16 and 17, T. 57 N., R. 84 W.; Carney No. 1 mine, Carney bed (face of room 10, off north entry 4, off east entry 2, 10-foot 6-inch bed and cut).	405	* 15677	A	1	25.62	30.15	40.82	3.41	.36	......	......	......	......	14.7	5,035	9,063
				2	......	40.53	54.89	4.58	.48	......	......	......	......	......	6,769	12,184
				3	......	42.48	57.52	......	.50	......	......	......	......	......	7,094	12,769
Same (in room 9, off north entry 2, off west entry 4, 25 feet in from track, 11-foot 2-inch bed and cut).	405	* 15678	A	1	25.36	31.12	40.12	3.40	.40	......	......	......	......	14.8	5,080	9,144
				2	......	41.69	53.75	4.56	.54	......	......	......	......	......	6,806	12,251
				3	......	43.68	56.32	......	.57	......	......	......	......	......	7,131	12,836
Same (face of main entry, 10-foot 10-inch bed and cut).	405	* 15679	A	1	24.93	30.89	40.35	3.83	.40	......	......	......	......	14.0	5,052	9,094
				2	......	41.15	53.75	5.10	.53	......	......	......	......	......	6,730	12,114
				3	......	43.36	56.64	......	.56	......	......	......	......	......	7,091	12,764
Same (face of room 12, off north entry 4, off west entry 3, 75 feet from north entry 4, 10-foot 1-inch bed and cut).	405	* 15680	A	1	24.48	31.60	40.12	3.80	.41	......	......	......	......	13.2	5,116	9,209
				2	......	41.84	53.13	5.03	.54	......	......	......	......	......	6,775	12,195
				3	......	44.06	55.94	......	.57	......	......	......	......	......	7,134	12,841
½ mile from Alger; secs. 16 and 17, T. 57 N., R. 84 W.; Carney No. 2 mine, Carney bed (face of room 5, off north entry 2, off west entry 2, 11-foot 10-inch bed and cut).	405	* 15681	A	1	24.21	31.02	40.94	3.83	.53	......	......	......	......	13.0	5,147	9,265
				2	......	40.93	54.02	5.05	.70	......	......	......	......	......	6,791	12,224
				3	......	43.11	56.89	......	.74	......	......	......	......	......	7,152	12,874
Same (face of room 16, off north entry 3, off east entry 1, 85 feet from north entry 3, 11-foot 1-inch bed and cut).	405	* 15682	A	1	24.51	31.47	40.31	3.71	.40	......	......	......	......	13.2	5,143	9,257
				2	......	41.69	53.40	4.91	.53	......	......	......	......	......	6,813	12,263
				3	......	43.84	56.16	......	.56	......	......	......	......	......	7,165	12,897
Same (face of room 15, off north entry 2, off east entry 2, 125 feet in room, 10-foot 11-inch bed and cut).	405	*15683	A	1	24.20	30.91	41.42	3.47	.35	......	......	......	......	14.2	5,183	9,329
				2	......	40.78	54.64	4.58	.46	......	......	......	......	......	6,838	12,308
				3	......	42.74	57.26	......	.48	......	......	......	......	......	7,166	12,899
Same (face of main north entry, 11-foot 7-inch bed and cut).	405	*15684	A	1	24.61	31.31	39.87	4.21	.39	......	......	......	......	13.1	5,106	9,191
				2	......	41.53	52.89	5.58	.52	......	......	......	......	......	6,773	12,191
				3	......	43.98	56.02	......	.55	......	......	......	......	......	7,173	12,911

Same (face of room 9, off north entry 4, off west entry 1, 65 feet from north entry 4, 10-foot 5-inch bed and cut).	405	*15685	A	1	25.41	29.82	40.60	4.17	.36					16.0	5,032	9,058
				2		39.98	54.43	5.59	.48						6,746	12,143
				3		42.35	57.65		.51						7,145	12,861
Same (composite of samples 15677–15685; includes samples from No. 1 mine).	406	*15686	A	1	24.66	30.81	40.65	3.88	.37	6.26	58.23	1.07	35.19	14.0	5,101	9,182
				2		40.89	53.96	5.15	.49	4.67	70.65	1.42	17.62		6,771	12,188
				3		43.11	56.89		.52	4.92	74.49	1.50	18,57		7,139	12,850
Dietz; sec. 34, T. 57 N., R. 84 W.; Dietz No. 4 mine, Dietz No. 2 (?) bed (face of room 3, off west entry 4, 8-foot 5½-inch bed and cut).	406	*13498	A	1	23.87	30.31	37.60	8.22	1.23					14.9		
				2		39.81	49.39	10.80	1.62							
				3		44.63	55.37		1.82							
Same (face of room 1, off west entry 12, off main north entry, 7-foot 8-inch bed and cut).	406	*13499	A	1	21.79	31.24	39.46	7.51	.98					12.6		
				2		39.94	50.46	9.60	1.25							
				3		44.18	55.82		1.38							
Same (composite of samples 13498 and 13499).....	406	*13537	A	1	22.88	32.51	36.74	7.87	1.11	6.12	50.59	1.25	33.06	13.8	4,998	8,996
				2		42.16	47.63	10.21	1.44	4.64	65.60	1.62	16.49		6,481	11,666
				3		46.95	53.05		1.60	5.17	73.06	1.80	18.37		7,218	12,992
Same (face of room 9, off west entry 6, off north entry 6, 7-foot 4-inch bed and cut).	407	*16221	A	1	21.88	34.07	35.54	8.51	1.30					14.9	5,049	9,088
				2		43.61	45.50	10.89	1.66						6,463	11,633
				3		48.94	51.06		1.86						7,253	13,055
Same (face of room 2, off west entry 12, off north entry 4, 7-foot 7-inch bed and cut).	407	*16222	A	1	22.36	33.37	37.70	6.57	.95					14.4	5,131	9,236
				2		42.98	48.56	8.46	1.22						6,609	11,896
				3		46.95	53.05		1.33						7,220	12,996
Same (face of north entry 4, off west entry 4, 7-foot 3-inch bed and cut).	407	*16223	A	1	21.88	34.09	36.78	7.25	.95					14.5	5,136	9,245
				2		43.64	47.08	9.28	1.22						6,575	11,835
				3		48.10	51.90		1.34						7,248	13,046
Same (face of room 10, off east entry 5, off main north entry, 7-foot 5-inch bed and cut).	407	*16224	A	1	21.63	33.64	38.29	6.44	.92					13.9	5,200	9,360
				2		42.92	48.86	8.22	1.17						6,635	11,943
				3		46.77	53.23		1.27						7,229	13,012
Same (face of main north entry, 7-foot 1-inch bed and cut.)	407	*16225	A	1	23.56	32.32	38.17	5.95	.73					16.6	5,050	9,090
				2		42.28	49.94	7.78	.95						6,606	11,891
				3		45.85	54.15		1.03						7,164	12,895
Same (on pillar in room 9, off east entry 14, 8-foot 7-inch bed and cut).	407	*16226	A	1	23.38	32.75	37.77	6.10	.99					16.4	5,089	9,160
				2		42.74	49.30	7.96	1.29						6,642	11,956
				3		46.44	53.56		1.40						7,217	12,991
Same (face of main entry, 150 feet from mine mouth, 6-foot 1-inch bed and cut).	407	*16227	A	1	21.74	33.87	37.89	6.50	1.00					14.3	5,119	9,214
				2		43.28	48.41	8.31	1.28						6,541	11,774
				3		47.20	52.80		1.40						7,134	12,841
Same (composite of samples 16221–16227)	407	*16228	A	1	22.52	33.28	37.48	6.72	.97	6.32	52.33	1.21	32.45	15.0	5,100	9,180
				2		42.95	48.38	8.67	1.25	4.93	67.54	1.56	16.05		6,583	11,849
				3		47.03	52.97		1.37	5.40	73.95	1.71	17.57		7,208	12,974
Kool, 1½ miles south of; sec. 27, T. 57 N., R. 85 W.; Hughey prospect, Monarch (?) bed, 4-foot bed.	407	12642	B	1	24.64	35.17	35.31	4.88	.31					15.2	4,820	8,676
				2		46.67	46.85	6.48	.41						6,396	11,513
				3		49.90	50.10		.44						6,839	12,310
NW. ¼ sec. 23, T. 57 N., R. 85 W.; Kool mine; Carney(?) (Monarch) bed (in east entry, 2,100 feet from mine mouth, 23-foot bed, 9-foot cut).	408	*12641	B	1	22.39	35.05	37.50	5.06	.55					11.9	5,104	9,187
				2		45.16	48.32	6.52	.71						6,577	11,839
				3		48.31	51.69		.76						7,035	12,663
Same (face of room 20, off fifth east entry, first panel, 7-foot 11-inch cut).	408	*16060	A	1	23.96	32.72	37.94	5.38	.65					17.0	5,065	9.117
				2		43.03	49.89	7.08	.85						6,661	11,990
				3		46.31	53.69		.91						7,169	12,904
Same (face of room 9, off fifth east entry, fourth panel, 9-foot cut).	408	*16061	A	1	24.02	33.12	38.40	4.46	.54					9.0	5,099	9,178
				2		43.59	50.54	5.87	.71						6,711	12,080
				3		46.31	53.69		.75						7,130	12,834

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WYOMING—Continued.																
SHERIDAN COUNTY—continued.																
Kool, NW. ¼ sec. 23, T. 57 N., R. 85 W.; Kool mine; Carney (?) (Monarch) bed (face of fifth east entry, 1,900 feet from the main slope, 7-foot 11-inch cut).	408	*16062	A	1	23.07	33.50	38.78	4.66	.60	......	......	......	......	11.9	5,182	9,328
				2	......	43.55	50.41	6.04	.78	......	......	......	......	......	6,736	12,125
				3	......	46.35	53.65	......	.83	......	......	......	......	......	7,169	12,904
Same (last crosscut off fourteenth east entry; 165 feet from main slope; 7-foot 9-inch cut).	408	*16063	A	1	23.02	33.76	38.88	4.34	.64	......	......	......	......	7.2	5,192	9,346
				2	......	43.85	50.51	5.64	.83	......	......	......	......	......	6,744	12,139
				3	......	46.47	53.53	......	.88	......	......	......	......	......	7,147	12,865
Same (face of room 4, off tenth east entry, fourth panel, 8-foot 6-inch cut).	408	*16064	A	1	23.66	32.49	39.62	4.23	.48	......	......	......	......	10.0	5,099	9,178
				2	......	42.56	51.90	5.54	.63	......	......	......	......	......	6,679	12,022
				3	......	45.05	54.95	......	.67	......	......	......	......	......	7,070	12,726
Same (face of room 7, off tenth east entry, second panel, 8-foot 4-inch cut).	408	*16065	A	1	22.14	34.29	37.82	5.75	1.00	......	......	......	......	8.4	5,133	9,239
				2	......	44.04	48.57	7.39	1.28	......	......	......	......	......	6,593	11,867
				3	......	47.55	52.45	......	1.38	......	......	......	......	......	7,119	12,814
Same (face of twelfth west entry; 185 feet from main slope; 8-foot 2-inch cut).	408	*16066	A	1	23.64	32.98	38.64	4.74	.66	......	......	......	......	8.9	5,084	9,151
				2	......	43.19	50.60	6.21	.86	......	......	......	......	......	6,658	11.984
				3	......	46.05	53.95	......	.92	......	......	......	......	......	7,099	12,778
Same (composite of samples 16060–16066)........	408	*16067	A	1	23.35	33.86	38.08	4.71	.64	6.30	53.10	1.25	34.00	10.3	5,116	9,209
				2	......	44.17	49.69	6.14	.83	4.84	69.27	1.63	17.29	......	6,674	12,013
				3	......	47.06	52.94	......	.88	5.16	73.80	1.74	18.42	......	7,110	12,798
Same (face of room 9, off fifth west entry, off main entry, 8-foot 5-inch cut).	408	*10818	A	1	23.60	32.12	39.19	5.09	.57	......	......	......	......	10.8	......	......
				2	......	42.04	51.30	6.66	.75	......	......	......	......	......	......	......
				3	......	45.04	54.96	......	.80	......	......	......	......	......	......	......
Same (face of main air course, which is also a manway, 8-foot 6-inch cut).	408	*10819	A	1	24.60	30.63	39.99	4.78	.53	......	......	......	......	12.4	......	......
				2	......	40.62	53.04	6.34	.70	......	......	......	......	......	......	......
				3	......	43.37	56.63	......	.75	......	......	......	......	......	......	......
Same (face of room 3, off seventh west entry, 8-foot 9½-inch cut).	408	*10820	A	1	23.84	31.99	39.16	5.01	.48	......	......	......	......	11.4	......	......
				2	......	42.00	51.42	6.58	.63	......	......	......	......	......	......	......
				3	......	44.96	55.04	......	.67	......	......	......	......	......	......	......
Same (face of room 6, off fifth east entry, off main entry, 8-foot ½-inch cut).	408	*10821	A	1	22.71	32.27	39.96	5.06	.59	......	......	......	......	9.6	......	......
				2	......	41.75	51.70	6.55	.76	......	......	......	......	......	......	......
				3	......	44.68	55.32	......	.81	......	......	......	......	......	......	......
Same (composite of samples 10818–10821)........	409	*10822	A	1	23.49	31.22	40.40	4.89	.50	6.57	52.00	.96	35.08	11.1	5,067	9,121
				2	......	40.80	52.81	6.39	.65	5.18	67.96	1.25	18.57	......	6,623	11,921
				3	......	43.59	56.41	......	.69	5.53	72.60	1.34	19.84	......	7,075	12,735

Monarch; Monarch mine, Monarch bed (pillar of butt entry 3, off west entry 9, near room 70, 7-foot 10-inch cut).	409	12005	A	1	22.25	35.01	39.01	3.73	.37	6.27	55.28	1.07	33.28	11.1	5,343	9,617
				2	....	45.03	50.17	4.80	.48	4.89	71.10	1.38	17.35	....	6,872	12,370
				3	....	47.30	52.70	....	.50	5.14	74.68	1.45	18.23	....	7,218	12,992
Same (pillar of west entry 9, near room 70, 8-foot 2-inch cut).	409	*12008	A	1	23.05	31.89	41.83	3.23	.37	6.44	54.65	1.10	34.21	9.5	5,269	9,484
				2	....	41.44	54.36	4.20	.48	5.04	71.02	1.43	17.83	....	6,847	12,325
				3	....	43.26	56.74	....	.50	5.26	74.13	1.49	18.62	....	7,147	12,865
Same (face of room 70, near west entry 7, 8-foot 2-inch cut).	409	*12006	A	1	23.41	33.55	39.44	3.60	.39	6.42	54.38	1.03	34.18	10.0	5,218	9,392
				2	....	43.81	51.49	4.70	.51	4.99	71.00	1.34	17.46	....	6,813	12,263
				3	....	45.97	54.03	....	.54	5.24	74.50	1.41	18.31	....	7,149	12,868
West of; SE. ¼ sec. 24, T. 57 N., R. 85 W.; New Monarch mine, Monarch bed (800 feet southwest of mine mouth, 10-foot cut).	409	*12685	B	1	23.88	34.33	38.44	3.35	.38	6.29	54.07	1.14	34.77	8.1	5,186	9,335
				2	....	45.10	50.50	4.40	.50	4.78	71.03	1.50	17.79	....	6,813	12,263
				3	....	47.17	52.83	....	.52	5.00	74.30	1.57	18.61	....	7,126	12,827
Same (face of east entry 6, 250 feet from main haulage slope, 9-foot 2-inch cut).	409	*15830	A	1	24.67	32.48	39.40	3.45	.44	....	....	....	....	15.9	5,139	9,250
				2	....	43.12	52.30	4.58	.58	....	....	....	....	....	6,822	12,280
				3	....	45.19	54.81	....	.61	....	....	....	....	....	7,149	12,868
Same (face of west entry 8, 9-foot 4-inch cut).....	409	*15831	A	1	22.56	34.36	38.74	4.34	.58	....	....	....	....	14.0	5,214	9,385
				2	....	44.37	50.03	5.60	.75	....	....	....	....	....	6,733	12,119
				3	....	47.00	53.00	....	.79	....	....	....	....	....	7,132	12,838
Same (face of west entry 13, off old slope, 8-foot 11-inch cut).	410	*15832	A	1	22.78	33.21	40.64	3.37	.32	....	....	....	....	13.6	5,282	9,508
				2	....	43.01	52.63	4.36	.41	....	....	....	....	....	6,840	12,312
				3	....	44.97	55.03	....	.43	....	....	....	....	....	7,152	12,874
Same (in west entry 4, opposite fifth and sixth butts, 800 feet from haulage way, 10 foot 1-inch cut).	410	*15833	A	1	22.26	34.79	38.81	4.14	.44	....	....	....	....	12.8	5,285	9,513
				2	....	44.75	49.92	5.33	.57	....	....	....	....	....	6,798	12,236
				3	....	47.27	52.73	....	.60	....	....	....	....	....	7,181	12,926
Same (face of room 8, off butt entry 4, off west entry 4, 10-foot 4-inch cut).	410	*15834	A	1	24.31	32.99	38.76	3.94	.49	....	....	....	....	14.3	5,143	9,257
				2	....	43.59	51.20	5.21	.65	....	....	....	....	....	6,795	12,231
				3	....	45.99	54.01	....	.69	....	....	....	....	....	7,169	12,904
Same (pillar between rooms 24 and 25, off west entry 12, 9-foot cut).	410	*15835	A	1	22.46	34.16	39.54	3.84	.49	....	....	....	....	12.8	5,280	9,504
				2	....	44.06	50.99	4.95	.63	....	....	....	....	....	6,810	12,258
				3	....	46.36	53.64	....	.66	....	....	....	....	....	7,165	12,897
Same (pillar between rooms 5 and 6, off butt entry 2, off west entry 3, 9-foot 9-inch cut).	410	*15836	A	1	22.42	34.17	39.58	3.83	.49	....	....	....	....	12.6	5,296	9,533
				2	....	44.05	51.01	4.94	.63	....	....	....	....	....	6,827	12,289
				3	....	46.34	53.66	....	.66	....	....	....	....	....	7,182	12,928
Same (composite of samples 15830–15836).........	410	*15837	A	1	23.24	33.01	39.88	3.87	.46	6.29	54.33	1.09	33.96	13.7	5,219	9,394
				2	....	43.01	51.95	5.04	.60	4.83	70.78	1.42	17.33	....	6,799	12,238
				3	....	45.29	54.71	....	.63	5.09	74.54	1.50	18.24	....	7,160	12,888
New Acme; sec. 22, T. 57 N., R. 84 W.; New Acme mine, Monarch bed (face of west entry 7, 150 feet from main slope, 6-foot 8-inch cut).	410	*16055	A	1	23.66	31.75	41.66	2.93	.41	....	....	....	....	9.3	5,257	9,463
				2	....	41.59	54.57	3.84	.54	....	....	....	....	....	6,886	12,395
				3	....	43.25	56.75	....	.56	....	....	....	....	....	7,161	12,890
Same (face of room 4, off east entry 5, 7-foot 2-inch cut).	410	*16056	A	1	21.71	33.51	40.80	3.98	.63	....	....	....	....	7.1	5,407	9,733
				2	....	42.80	52.12	5.08	.80	....	....	....	....	....	6,906	12,431
				3	....	45.09	54.91	....	.84	....	....	....	....	....	7,275	13,095
Same (face of room 13, off east entry 3, 7-foot 7-inch cut).	410	*16057	A	1	24.29	33.49	38.68	3.54	.58	....	....	....	....	10.3	5,244	9,439
				2	....	44.23	51.09	4.68	.77	....	....	....	....	....	6,926	12,467
				3	....	46.40	53.60	....	.81	....	....	....	....	....	7,266	13,079
Same (face of room 11, off east entry 2, 6-foot 9-inch cut).	410	*16058	A	1	24.24	31.44	41.55	2.77	.43	....	....	....	....	10.1	5,218	9,392
				2	....	41.50	54.84	3.66	.57	....	....	....	....	....	6,888	12,398
				3	....	43.08	56.92	....	.59	....	....	....	....	....	7,150	12,870
Same (composite of samples 16055–16058).........	410	*16059	A	1	23.30	32.05	41.34	3.31	.53	6.35	54.96	1.01	33.84	9.2	5,296	9,533
				2	....	41.79	53.89	4.32	.69	4.90	71.66	1.32	17.11	....	6,905	12,429
				3	....	43.67	56.33	....	.72	5.12	74.89	1.38	17.89	....	7,216	12,989

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WYOMING—Continued.																
SWEETWATER COUNTY.																
Gunn, 4 miles from Baxter Junction; sec. 8, T. 19 N., R. 104 W.; B mine, No. 11 bed (face of room 15, off entry 5, 6-foot 2-inch bed).	411	11507	A	1	17.37	33.89	46.76	1.98	.92	....	....	....	....	13.3	....	....
				2	....	41.01	56.59	2.40	1.11	....	....	....	....	....	....	....
				3	....	42.02	57.08	....	1.14	....	....	....	....	....	....	....
Same (face of room 25, off entry 4, 6-foot 1-inch bed).	411	11508	A	1	15.79	33.96	48.27	1.98	.79	....	....	....	....	11.5	....	....
				2	....	40.33	57.32	2.35	.94	....	....	....	....	....	....	....
				3	....	41.30	58.70	....	.96	....	....	....	....	....	....	....
Same (face of room 8, off entry 8, 6-foot 2-inch bed).	411	11509	A	1	16.03	34.65	47.16	2.16	.95	....	....	....	....	11.8	....	....
				2	....	41.26	56.17	2.57	1.13	....	....	....	....	....	....	....
				3	....	42.35	57.65	....	1.16	....	....	....	....	....	....	....
Same (composite of samples 11507, 11508, 11509)..	411	11510	A	1	16.32	34.86	46.77	2.05	.87	6.15	63.61	1.33	25.99	12.2	6,196	11,153
				2	....	41.66	55.89	2.45	1.04	5.19	76.01	1.59	13.72	....	7,404	13,327
				3	....	42.71	57.29	....	1.07	5.32	77.92	1.63	14.06	....	7,590	13,662
Superior; sec. 29, T. 21 N., R. 102 W.; B mine, No. 7 bed (room 27, off inside dip entry, 7-foot 2-inch bed and cut).	412	11461	A	1	11.07	37.49	47.93	3.51	1.03	....	....	....	....	7.4	....	....
				2	....	42.16	53.89	3.95	1.16	....	....	....	....	....	....	....
				3	....	43.89	56.11	....	1.21	....	....	....	....	....	....	....
Same (face of south entry 2, 7-foot 4¾-inch bed and cut).	412	11462	A	1	12.65	36.61	47.29	3.45	1.01	....	....	....	....	8.6	....	....
				2	....	41.91	54.14	3.95	1.16	....	....	....	....	....	....	....
				3	....	43.63	56.37	....	1.21	....	....	....	....	....	....	....
Same (face of room 20, off south entry 3, 7-foot 3¾-inch bed and cut).	412	11463	A	1	11.74	37.15	47.52	3.59	.80	....	....	....	....	7.1	....	....
				2	....	42.09	53.84	4.07	.91	....	....	....	....	....	....	....
				3	....	43.87	56.13	....	.95	....	....	....	....	....	....	....
Same (room 11, off north entry 3, 6-foot 3½-inch bed and cut).	412	11464	A	1	11.64	37.86	47.24	3.26	.86	....	....	....	....	7.0	....	....
				2	....	42.85	53.46	3.69	.97	....	....	....	....	....	....	....
				3	....	44.49	55.51	....	1.01	....	....	....	....	....	....	....
Same (composite of samples 11461–11464).........	412	11465	A	1	11.79	36.96	47.88	3.37	.99	5.90	67.87	1.44	20.43	7.5	6,666	11,999
				2	....	41.90	54.28	3.82	1.12	5.20	76.94	1.63	11.29	....	7,557	13,603
				3	....	43.56	56.44	....	1.16	5.41	79.99	1.69	11.75	....	7,857	14,143
Sec. 21, T. 21 N., R. 101 W., Superior C mine, No. 1 bed (room 15, off south entry 6, 8-foot 7½-inch bed, 8-foot 6-inch cut).	412	11456	A	1	12.53	36.22	47.36	3.89	1.05	....	....	....	....	9.0	....	....
				2	....	41.41	54.14	4.45	1.20	....	....	....	....	....	....	....
				3	....	43.34	56.66	....	1.26	....	....	....	....	....	....	....

Same (room 25, off south entry 5, 8-foot 7½-inch bed, 8-foot 6-inch cut).	412	11457	A	1	13.26	36.54	48.33	1.87	1.00	......	......	......	......	9.7	......	......
				2	......	42.13	55.71	2.16	1.15	......	......	......	......	......	......	......
				3	......	43.06	56.94	......	1.18	......	......	......	......	......	......	......
Same (room 1, off north entry 7, 6-foot 7-inch bed and cut).	412	11458	A	1	13.34	36.07	47.70	2.89	.83	......	......	......	......	9.5	......	......
				2	......	41.62	55.05	3.33	.96	......	......	......	......	......	......	......
				3	......	43.05	56.95	......	.99	......	......	......	......	......	......	......
Same (room 38, south entry 4, 8-foot 6½-inch bed, 8-foot 5-inch cut).	412	11459	A	1	13.79	35.15	46.68	4.38	1.00	......	......	......	......	10.1	......	......
				2	......	40.77	54.15	5.08	1.16	......	......	......	......	......	......	......
				3	......	42.95	57.05	......	1.22	......	......	......	......	......	......	......
Same (composite of samples 11456, 11457, 11458, and 11459).	412	11460	A	1	13.11	35.75	47.74	3.40	1.00	5.87	66.17	1.25	22.31	9.6	6,455	11,619
				2	......	41.14	54.95	3.91	1.15	5.08	76.16	1.44	12.26	......	7,429	13,372
				3	......	42.81	57.19	......	1.20	5.29	79.26	1.50	12.75	......	7,731	13,916
UINTA COUNTY.																
Elkol; sec. 11, T. 20 N., R. 117 W.; Elkol mine, No. 1 bed (face of north entry 3, 1,900 feet from slope mouth, 51-foot bed, 8-foot 6-inch cut).	413	11730	A	1	21.45	35.29	40.42	2.84	.61	6.40	58.35	.77	31.03	15.2	5,698	10,256
				2	......	44.93	51.45	3.62	.78	5.12	74.29	.98	15.21	......	7,254	13,057
				3	......	46.62	53.38	......	.81	5.31	77.08	1.02	15.78	......	7,527	13,549
Same (face of room 1, off north entry 3, 1,450 feet from slope mouth, 51-foot bed, 8-foot 6-inch cut).	413	11731	A	1	20.60	35.09	39.86	4.45	.57	6.28	57.82	.78	30.60	14.9	5,610	10,098
				2	......	44.19	50.21	5.60	.72	5.03	72.19	.98	15.48	......	7,065	12,717
				3	......	46.81	53.19	......	.76	5.33	76.47	1.04	16.40	......	7,484	13,471
Frontier; sec. 12, T. 21 N., R. 116 W.; Kemmerer No. 1 mine, No. 1 bed (face of room 16, off south entry 13, 8-foot 4-inch cut).	413	11565	A	1	5.65	37.70	51.34	5.31	1.41	5.46	70.58	1.02	16.22	2.9	6,989	12,580
				2	......	39.96	54.41	5.63	1.49	5.12	74.81	1.08	11.87	......	7,408	13,334
				3	......	42.35	57.65	......	1.58	5.43	79.28	1.14	12.57	......	7,850	14,130
Same (pillar of room 18, off north entry 10, 8-foot 5-inch bed, 8-foot 1-inch cut).	413	11566	A	1	6.29	39.04	51.14	3.53	.90	5.54	71.95	.98	17.10	3.5	7,100	12,780
				2	......	41.66	54.57	3.77	.96	5.16	76.78	1.05	12.28	......	7,576	13,637
				3	......	43.29	56.71	......	1.00	5.36	79.79	1.09	12.76	......	7,873	14,171
Same (face of room 3, off north entry 7, 5-foot 10-inch bed, 5-foot 4-inch cut).	413	11567	A	1	5.57	37.44	47.46	9.53	1.40	5.50	67.43	.93	15.21	3.0	6,669	12,004
				2	......	39.65	50.26	10.09	1.48	5.17	71.41	.98	10.87	......	7,062	12,712
				3	......	44.10	55.90	......	1.65	5.75	79.42	1.09	12.09	......	7,854	14,137
Same (stub pillar of room 62, off south entry 7, 6-foot bed, 5-foot 2-inch cut).	413	11568	A	1	5.18	38.73	47.54	8.55	1.40	5.53	68.08	1.10	15.34	2.7	6,783	12,209
				2	......	40.84	50.14	9.02	1.48	5.22	71.80	1.16	11.32	......	7,153	12,875
				3	......	44.89	55.11	......	1.63	5.74	78.92	1.27	12.44	......	7,862	14,152
Kemmerer, 1½ miles west of: sec. 33, T. 22 N., R. 116 W.; prospect (face of first entry, 200 feet from mouth of slope, 16-foot bed).	414	11740	A	1	18.07	37.09	42.34	2.50	1.95	6.27	60.01	1.05	28.22	9.6	5,905	10,629
				2	......	45.27	51.68	3.05	2.38	5.20	73.25	1.28	14.84	......	7,208	12,974
				3	......	46.70	53.30	......	2.45	5.36	75.56	1.32	15.31	......	7,435	13,383
Susie, 4 miles north of Kemmerer; secs. 25 and 26, T. 22 N., R. 116 W.; Kemmerer No. 4 mine, Kemmerer No. 1 bed (in mine, 1,000 feet from mouth of slope, 5-foot 2 inch bed and cut).	414	13517	A	1	6.19	39.63	48.42	5.76	1.31	......	......	......	......	3.0	......	......
				2	......	42.25	51.61	6.14	1.40	......	......	......	......	......	......	......
				3	......	45.01	54.99	......	1.49	......	......	......	......	......	......	......
Same (face of north entry 2, 1,300 feet from mouth of slope, 4-foot 3-inch bed and cut).	414	13518	A	1	7.00	38.93	47.09	6.98	1.34	......	......	......	......	3.8	......	......
				2	......	41.86	50.63	7.51	1.44	......	......	......	......	......	......	......
				3	......	45.26	54.74	......	1.56	......	......	......	......	......	......	......
Same (composite of samples 13517 and 13518).....	414	13519	A	1	6.56	39.20	47.78	6.46	1.37	5.29	69.56	1.25	16.07	3.4	6,866	12,359
				2	......	41.95	51.14	6.91	1.47	4.88	74.44	1.34	10.96	......	7,348	13,226
				3	......	45.06	54.94	......	1.58	5.24	79.96	1.44	11.78	......	7,893	14,207

Table of chemical analyses—Continued.

Locality, bed, etc.	Mine notes (page).	Sample.			Proximate.				Ultimate.					Air-drying loss.	Calorific value.	
		Laboratory No.	Kind.	Condition.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.		Calories.	British thermal units.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
WYOMING—Continued.																
WESTON COUNTY.																
Cambria, 6½ miles from Newcastle; sec. 20, T. 46 N., R. 61 W.; Antelope No. 4 mine (face of east entry 1, 3-foot 11-inch bed, 2-foot 11-inch cut).	415	16,412	A	1	10.97	38.25	33.30	17.48	4.80	......	......	......	......	8.5	5,508	9,914
				2	......	42.96	37.41	19.63	5.39	......	......	......	......	......	6,187	11,137
				3	......	53.45	46.55	......	6.71	......	......	......	......	......	7,698	13,856
Same (face of north entry 1, off east entry 1; 6-foot 1-inch bed, 5-foot 3-inch cut).	415	16413	A	1	10.35	39.75	34.65	15.25	4.99	......	......	......	......	7.6	5,725	10,305
				2	......	44.34	38.65	17.01	5.57	......	......	......	......	......	6,386	11,495
				3	......	53.43	46.57	......	6.71	......	......	......	......	......	7,695	13,851
Same (face of room 29 off east entry 3, 8-foot 11-inch bed, 6-foot 2-inch cut).	415	16414	A	1	10.84	38.99	37.73	12.44	5.07	......	......	......	......	8.2	5,969	10,744
				2	......	43.73	42.32	13.95	5.60	......	......	......	......	......	6,695	12,051
				3	......	50.82	49.18	......	6.66	......	......	......	......	......	7,780	14,004
Same (composite of samples 16412, 16413, and 16414).	415	16415	A	1	10.77	39.10	35.10	15.03	4.87	5.46	56.09	.74	17.81	8.1	5,745	10,341
				2	......	43.82	39.34	16.84	5.46	4.77	62.86	.83	9.24	......	6,438	11,588
				3	......	52.69	47.31	......	6.57	5.74	75.59	1.00	11.10	......	7,742	13,936
Sec. 19, T. 46 N., R. 61 W.; Antelope No. 3 mine (face of room 35, off east entry 5; 3-foot 9-inch bed and cut).	415	16428	A	1	11.88	37.32	34.64	16.16	4.67	......	......	......	......	9.2	5,573	10,031
				2	......	42.35	39.31	18.34	5.30	......	......	......	......	......	6,324	11,383
				3	......	51.86	48.14	......	6.49	......	......	......	......	......	7,744	13,939
Same (face of west entry 8, 7-foot 3-inch bed, 6-foot 6-inch cut).	415	16429	A	1	9.52	40.79	34.24	15.45	4.98	......	......	......	......	7.2	5,837	10,507
				2	......	45.08	37.84	17.08	5.50	......	......	......	......	......	6,451	11,612
				3	......	54.37	45.63	......	6.63	......	......	......	......	......	7,780	14,004
Same (face of west entry7, 7-foot 5-inch bed, 6-foot 11-inch cut).	415	16430	A	1	8.92	38.41	34.71	17.96	4.92	......	......	......	......	6.2	5,676	10,217
				2	......	42.17	38.11	19.72	5.40	......	......	......	......	......	6,232	11,218
				3	......	52.53	47.47	......	6.73	......	......	......	......	......	7,763	13,973
Same (composite of samples 16428, 16429, and 16430).	415	16431	A	1	10.00	39.11	34.33	16.56	4.90	5.36	55.21	.68	17.29	7.5	5,693	10,247
				2	......	43.46	38.14	18.40	5.44	4.72	61.34	.76	9.34	......	6,325	11,385
				3	......	53.26	46.74	......	6.67	5.78	75.17	.93	11.45	......	7,751	13,952
Moorcroft, 12 miles south of, 15 miles northwest of Upton; SE. ¼ sec. 29, T. 48 N., R. 67 W.; prospect (face of entry, 100 feet north of mine mouth; 3-foot bed, 2-foot 11-inch cut).	416	*12446	B	1	26.08	31.05	36.86	6.01	.56	6.26	49.36	.91	36.90	14.0	4,647	8,365
				2	......	42.00	49.87	8.13	.76	4.55	66.77	1.23	18.56	......	6,286	11,315
				3	......	45.72	54.28	......	.83	4.95	72.68	1.34	20.20	......	6,842	12,316
12 miles southwest of; sec. 29, T. 48 N., R. 68 W.; prospect (28-inch bed and cut).	416	*12392	B	1	29.97	28.77	34.52	6.74	.92	......	......	......	......	17.2	4,279	7,702
				2	......	41.08	49.30	9.62	1.31	......	......	......	......	......	6,110	10,998
				3	......	45.45	54.55	......	1.45	......	......	......	......	......	6,760	12,168

DESCRIPTIONS OF SAMPLES.

ALABAMA.

JEFFERSON COUNTY.

LITTLETON. BANNER MINE.

Sample.—Bituminous coal; Warrior field; analyses Nos. 12083, 12084, 12085, and 12086 (p. 17).

Mine.—Banner, a shaft (originally a slope) mine near Littleton, about 20 miles northwest of Birmingham, on the Cane Creek division of the Louisville & Nashville Railroad.

Coal bed.—Big or Mary Lee. Carboniferous age, Pottsville group. This same bed is worked at Palos, Ala. Thickness about 10 feet. The bed is practically horizontal, the dip being 1 to 2 per cent. The bed is overlain by sandstone and the floor is shale. Near the center of the bed is a stratum of hard gray shale 18 to 22 inches thick, known as "middleman." The bed was measured and sampled at three points in the mine by J. J. Rutledge, on April 26, 1911, as described below:

Sections of coal bed in Banner mine.

Section	A		B		C	
Laboratory No.	12083		12084		12085	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	2	1	4	1	2
Shale, gray	[a]0	1	[a]0	1	[a]0	1
Coal	0	8	0	8	0	7
Shale, hard, gray	..	..	[a]0	8	[a]0	10
Coal	1	8	1	4	1	9
"Sulphur"	..	..	[a]0	1	..	..
Coal	..	..	0	4	..	..
"Sulphur" ("slate")	[a]0	1	[a]0	1	[a]0	1
Coal	0	6	0	6	0	6
Shale, hard, gray	[a]1	6	[a]1	9	[a]1	9
Coal	2	3	2	7	2	7
Shale, gray	0	1	[a]0	2	0	1
Coal	0	7½	0	9	0	7
Floor, shale.						
Thickness of bed	8	7½	10	4	10	0
Thickness of coal sampled	6	11½	7	6	7	4

[a] Not included in sample.

Section A (sample 12083) was cut from the rib near the mouth of room 4, off right entry 5, 2,000 feet east of shaft No. 1.

Section B (sample 12084) was cut from the rib near the mouth of room 2, off left entry 8, 2,400 feet east of shaft No. 1.

Section C (sample 12085) was cut from the right rib of the main slope, at the mouth of left entry 9, 2,700 feet east of shaft No. 1.

A composite sample was made by combining face samples 12083, 12084, and 12085. The results of an ultimate analysis of this sample are given under Laboratory No. 12086.

Notes.—The room-and-pillar system of mining is used. At the time of sampling the coal was undercut by electric chain machines and shot down with a permissible explosive. The output was about 1,000 tons a day, all of which was from advance workings. In advance workings, about 40 per cent of the coal was taken out.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 285, 1906, p. 216.

MORRIS. INDIO MINE.

Sample.—Bituminous coal; Warrior field; analyses Nos. 16683, 16684, 16685, 16686, and 16687 (p. 17).

Mine.—The Indio mine is opened by a shaft 186 feet deep, and is 4 miles east of Morris, on the Louisville & Nashville Railroad.

Coal bed.—Jefferson. Carboniferous age, Black Creek formation. Thickness, 4 feet to 4 feet 8 inches. The bed lies practically horizontal and has two characteristic shale partings, one near the bottom and one near the top. Roof, shale, with a sandstone cap rock. The shale occasionally came down with the coal and particles of the roof got mixed with the coal in loading. The floor is a hard, smooth shale, particles of which did not get mixed with the coal in mining.

The bed was measured and sampled at four points in the mine by E. B. Sutton and J. W. Webb, on March 28, 1913, as described below:

Sections of coal bed in Indio mine.

Sections	A		B		C		D	
Laboratory No.	16683		16684		16685		16686	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	7	0	8	0	8½	0	8½
Shale	[a]0	2	[a]0	3	[a]0	5	[a]0	3½
Coal	1	6	1	7	1	2	1	3½
Shale	[a]1	8	[a]1	8	[a]1	3	[a]1	2½
Coal	0	6	0	6	0	6	0	7
Floor, shale.								
Thickness of bed	4	5	4	8	4	½	4	1
Thickness of coal sampled	2	7	2	9	2	4½	2	7

[a] Not included in sample.

Section A (sample 16683) was measured at the face of room 7, off the fifth left south entry.

Section B (sample 16684) was measured at the face of room 4, off the seventh right north entry.

Section C (sample 16685) was measured at the neck of room 4, off the sixth right south entry.

Section D (sample 16686) was measured at the intersection of the main south air course and the sixth left south entry.

A composite sample was made by combining face samples 16683 to 16686, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 16687.

Notes.—At the time the mine was visited the coal was shot off the solid with permissible explosives. Black powder was used for brushing the floor and roof. The coal was all shipped as run-of-mine. The average daily output was 275 tons. There was about 100 acres to be mined from this opening.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 285, 1906, p. 219.

ST. CLAIR COUNTY.

COAL CITY. COAL BRANCH MINE.

Sample.—Bituminous coal; Coosa Basin field; analyses Nos. 12440, 12441, 12442, and 12443 (p. 18).

Mine.—Coal Branch, a slope mine at Coal City, on the Seaboard Air Line Railway.

Coal bed.—Coal City. Carboniferous age, Pottsville group. Thickness, 2 feet 10 inches to 5 feet; roof, weak calcareous shale; floor, hard shale.

The bed was measured and sampled at three points in the mine by Charles Butts on July 21, 1911, as described below:

Section of coal bed in Coal Branch mine.

Section	A	B	C
Laboratory No	12440	12441	12442
Roof, shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	4 10	3 10	2 10
"Rash"	[a]0 2		
Floor, shale, hard.			
Thickness of bed	5 0	3 10	2 10
Thickness of coal sampled	4 10	3 10	2 10

[a] Not included in sample.

Section A (sample 12440) was measured in the first left room off twenty-sixth entry, 3,000 feet north of the slope mouth.

Section B (sample 12441) was measured at the end of thirty-fourth entry, 2,800 feet north of the slope mouth.

Section C (sample 12442) was measured at the end of room 32, 3,000 feet northeast of the slope mouth.

A composite sample was made by combining face samples 12440, 12441, and 12442. The results of an ultimate analysis of this sample are shown under laboratory No. 12443.

COAL CITY. VULCAN NO. 4 MINE.

Sample.—Bituminous coal; Coosa Basin field; analyses Nos. 12436, 12437, 12438, 12439 (p. 18).

Mine.—Vulcan No. 4; a slope mine at Coal City, on the Seaboard Air Line Railway.

Coal bed.—Broken Arrow. Carboniferous age, Pottsville group. Thickness, 3 feet 6 inches to 4 feet 9 inches.

The bed was measured and sampled at three points in the mine by Charles Butts on July 21, 1911, as described below:

Section of coal bed in Vulcan No. 4 mine.

Section	A	B	C
Laboratory No	12436	12437	12438
Roof, shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal, bony	[a]0 3	[a]0 5	[a]0 4
Clay	[a]0 3½	[a]0 5	[a]0 3½
Coal, bony	[a]0 3½	[a]0 3	[a]0 3
Coal, hard	1 ..	0 10	0 9
Coal, soft	1 6	1 10	1 10
Coal, rashy, "mining"	0 9½	1 0	0 10
Floor, shale.			
Thickness of bed	4 1½	4 9	4 3½
Thickness of coal sampled	3 3½	3 8	3 5

[a] Not included in sample.

Section A (sample 12436) was measured in room 21, off second left heading, 750 feet southeast of the mine mouth.

Section B (sample 12437) was measured 1,100 feet east of the mine mouth.

Section C (sample 12438) was measured in room 15, off second left heading, 600 feet south of the mine mouth.

A composite sample was made by combining face samples 12436, 12437, and 12438. The results of an ultimate analysis of this sample are shown under laboratory No. 12439.

SHELBY COUNTY.

STRAVEN. STRAVEN MINE.

Sample.—Bituminous coal; Cahaba field; analyses Nos. 14391 and 14392 (p. 18).

Mine.—Straven, a slope mine in sec. 7, T. 21S., R. 3 W., at Straven, on the Mobile branch of the Southern Railway.

Coal bed.—Upper Straven. Carboniferous age, Pottsville group. Thickness, 5 feet 6½ inches.

The bed was measured and sampled at one point in the mine by Charles Butts on June 14, 1912, as described below:

Section of coal bed in Straven mine.

Laboratory No	14391 14392	
Roof, shale.	*Ft.*	*in.*
Coal, rash, and shale	[a]0	8
Coal, hard	[b]0	10½
Parting (knife edge to 4 inches of rash or soft coal)		
Coal	[b]2	2
Clay	[a]0	8¼
Coal, softer than upper bench	[c]1	1½
Floor, clay:		
Thickness of bed	4	4½

[a] Excluded from samples. [b] Included in sample 14391. [c] Included in sample 14392.

The section was measured near the end of the slope, 2,000 feet north of the slope mouth.

Notes.—The coal was hard and bright. At the time of sampling, the coal was sold as run-of-mine for domestic and steam purposes.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 39, 351.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 431, 1909, p. 140.

TUSCALOOSA COUNTY.

ABERNANT. ABERNANT MINE.

Sample.—Bituminous coal; Warrior field; analysis No. 14717 (p. 19).

Mine.—Abernant, a slope mine in sec. 18, T. 20 S, R. 6 W., at Abernant, seven-eighths of a mile south of Yolande.

Coal bed.—Jagger. Carboniferous age, Pottsville group. Thickness of coal mined, 6 to 7 feet; dip, 14° NW. The coal has no cleat, but has several shale partings. Roof-shale, which sometimes comes down and gets mixed with the coal in mining; above the shale is a sandstone cap rock. Floor, hard blue sandstone, which did not mix with the coal in loading.

The bed was measured and sampled at one point in the mine by E. B. Sutton on August 15, 1912, as described below:

Section of coal bed in Abernant mine.

Laboratory No	14717	
Roof, sandstone.	*Ft.*	*in.*
Bone	[a]0	1½
Coal	0	8
Mother coal	[a]0	1
Coal	0	5
Shale	[a]0	3
Coal	1	0
Shale	[a]0	1
Coal	0	11
Shale	[a]0	1
Coal	1	8
Shale	[a]0	4
Coal	0	2
Shale	[a]0	2
Coal	0	8
Floor, sandstone.		
Thickness of bed	6	7½
Thickness of coal sampled	5	6

[a] Not included in sample.

Section A (sample 14717) was measured at the face of right entry 13.

Notes.—At the time the mine was sampled the coal was undercut by hand and blasted down with a permissible explosive. The tipple was a wooden structure. All coal was shipped as run-of-mine. Two men were employed to trim the coal and pick "slate" from the cars. The average daily output was 300 tons a day, with a capacity of 500 tons.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 39, 352.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 260, 1905, p. 375, and Bull. 400, 1910, p. 170.

ALASKA.

ALASKA PENINSULA.

CHIGNIK BAY. WARNER CLAIM.

Sample.—Bituminous coal; Alaska Peninsula field; analysis No. 12742 (p. 19).

Mine.—Warner claim, Chignik Bay.

Coal bed.—Not named. Cretaceous age, Chignik (?) formation.

Notes.—Sample 12742 was taken at the Warner claim on November 1, 1911. No further information available.

The coal beds of the Chignik Bay region are described in U. S. Geol. Survey Bull. 467, 1911, p. 109.

BERING RIVER.

BARRETT CREEK. CUNNINGHAM CLAIMS.

Sample.—Semibituminous coal; Bering River field; analysis No. 12716 (p. 19).

Locality.—Cunningham claims, on Barrett Creek.

Coal bed.—Unnamed. Tertiary age, Kushtaka formation. Thickness, about 5 feet. A number of beds outcrop on this creek.

Notes.—The bed was sampled on November 4, 1911. The sample was taken at an outcrop on Barrett Creek, three-fourths of a mile above its junction with Clear Creek. The bed was about 5 feet thick at this point.

For a description of the geologic relations of this coal bed, see Report on Coal in Alaska for Use in United States Navy: House Document No. 876, 1914, p. 38.

BARRETT CREEK. CUNNINGHAM CLAIMS.

Sample.—Semibituminous coal; Bering River field; analysis No. 12709 (p. 19).

Locality.—Cunningham claims, Barrett Creek.

Coal bed.—Unnamed. Tertiary age, Kushtaka formation. Thickness about 2 feet. This bed lies above the bed represented by sample 12716.

Notes.—The bed was sampled at one point on November 3, 1911. The sample was taken at a point about 100 feet up the creek, measured across the dip, from the place where sample 12716 was taken. The bed was about 2 feet thick at the point sampled.

For a description of the geologic relations of this coal bed, see Report on Coal in Alaska for Use in United States Navy: House Document No. 876, 1914, p. 38.

BARRETT CREEK. CUNNINGHAM CLAIMS.

Sample.—Semibituminous coal; Bering River field; analysis No. 12707 (p. 19).

Locality.—Cunningham claims on Barrett Creek.

Coal bed.—Unnamed bed. Tertiary age, Kushtaka formation. The bed outcrops above the one represented by sample 12709.

Notes.—The bed was sampled at one point on November 3, 1911. The sample was taken at a point 15 to 20 feet, measured across the strike, above the bed represented by sample 12709.

For a description of the geologic relations of this coal bed see Report on Coal in Alaska for Use in United States Navy: House Document No. 876, 1914, p. 38.

BARRETT CREEK. CUNNINGHAM CLAIMS.

Sample.—Semibituminous coal; Bering River field; analysis No. 12708 (p. 19).

Location.—Cunningham claims on Barrett Creek.

Coal bed.—Unnamed. Tertiary age, Kushtaka formation. The bed lies between the beds represented by samples 12707 and 12714.

Notes.—The bed was sampled at one point on November 3, 1911. The sample was taken from an outcrop on the creek, at a point between the third and fifth seams, about 350 feet, measured along the creek, or between 100 and 125 feet, measured across strike, below fifth bed, and probably between 100 and 150 feet, measured across strike, above the third bed (noted from memory).

For a description of the geologic relations of this coal bed see Report on Coal in Alaska for Use in United States Navy: House Document No. 876, 1914, p. 38.

BARRETT CREEK. CUNNINGHAM CLAIMS.

Sample.—Semibituminous coal; Bering River field; analysis Nos. 12710 and 12714 (p. 19).

Locality.—Cunningham claims on Barrett Creek.

Coal bed.—Unnamed. Tertiary age, Kushtaka formation.

Notes.—The bed was sampled at two points on November 4, 1911. Sample 12714 was taken from an outcrop east of Barrett Creek, about 350 feet up the creek, and about 125 to 150 feet, measured across the strike, from the fourth bed on this creek, and represents 3 to 4 feet of coal next the roof or hanging wall opposite the part represented by sample 12710. There was a band of shale in the middle of the bed 1 to 4 feet thick.

Sample 12710 was taken between the foot wall or floor and the band of shale in the middle of the bed, and represents a bench of coal about 3 feet thick.

For a description of the geologic relations of this coal bed see Report on Coal in Alaska for Use in United States Navy: House Document No. 876, 1914, p. 38.

BARRETT CREEK. CUNNINGHAM CLAIMS.

Sample.—Semibituminous coal; Bering River field; analysis No. 12718 (p. 19).

Locality.—Cunningham claims, on Barrett Creek.

Coal bed.—Unnamed. Tertiary age, Kushtaka formation.

Notes.—The bed was sampled at one point on November 4, 1911. Sample 12718 was taken 20 to 30 feet, measured across strike, above points where samples 12710 and 12714 were taken, about three-quarters mile above mouth of creek.

For a description of the geologic relations of this coal bed see Report on Coal in Alaska for Use in United States Navy: House Document No. 876, 1914, p. 38.

BARRETT CREEK. CUNNINGHAM CLAIMS.

Sample.—Semibituminous coal; Bering River field; analysis No. 12712 (p. 19).

Locality.—Cunningham claims, on Barrett Creek.

Coal bed.—Unnamed. Tertiary age, Kushtaka formation.

Notes.—The coal bed was sampled on November 3, 1911. The sample was taken from a point about 150 feet, measured upstream, or 50 feet, measured across the strike, from points where samples 12710 and 12718 were taken.

For a description of the geologic relations of this coal bed see Report on Coal in Alaska for Use in United States Navy: House Document No. 876, 1914, p. 38.

BRANCH OF CLEAR CREEK. CUNNINGHAM CLAIMS.

Sample.—Semibituminous coal; Bering River field; analysis No. 12711 (p. 19).

Locality.—Cunningham claims, on creek which empties into Clear Creek about 1,000 feet above mouth of Barrett Creek.

Coal bed.—Unnamed. Tertiary age, Kushtaka formation. Thickness at outcrop, 15 feet. The coal in the bed was crushed.

Notes.—The bed was sampled on November 3, 1911. The sample was taken at a point about 3,000 feet northeast of the point on Barrett Creek where sample 12712 was taken.

For a description of the geologic relations of this coal bed see Report on Coal in Alaska for Use in United States Navy: House Document No. 876, 1914, p. 38.

CARBON MOUNTAIN. NO. 1 OPENING.

Sample.—Semibituminous coal; Bering River field; analysis No. 12732 (p. 19).

Locality.—No. 1 opening, on Carbon Mountain.

Coal bed.—Unnamed. Tertiary age, Kushtaka (?) formation.

Notes.—The bed was sampled on November 7, 1911, at No. 1 opening. The bed or lens was 10 or 12 feet thick at this point.

For a description of the geologic relations of this coal bed see Report on Coal in Alaska for Use in United States Navy: House Document No. 876, 1914, p. 45.

CLEAR CREEK. OUTCROP.

Sample.—Semianthracite coal; Bering River field; analyses Nos. 12713, 12715, and 12717 (p. 20).

Locality.—Clear Creek, 4½ miles from mouth of creek.

Coal bed.—Unnamed. Tertiary age, Kushtaka formation.

Thickness, 10 to 13 feet. The bed outcrops just below the main (lower) falls on Clear Creek, about 4½ miles above the mouth of the creek, in the east wall of the gorge, and had not been found in the west wall. The hanging wall or roof is 3 to 4 feet of diabase sill.

Just below the diabase sill or roof is a layer of natural coke, 6 to 10 inches thick.

Notes.—Three samples were taken from this bed on November 4, 1911. Sample 12713 represents the layer of natural coke. Sample 12715 represents 6 to 8 inches of the coal next to the coke. Sample 12717 represents coal from the main part of the bed.

For a description of the geologic relations of this coal bed see Report on Coal in Alaska for Use in United States Navy: House Document No. 876, 1914, p. 41.

CUNNINGHAM CLAIM.

Sample.—Semibituminous coal; Bering River field; analysis No. 12739 (p. 20).

Locality.—Cunningham claim.

Coal bed.—Unnamed.

Notes.—The bed was sampled on November 8, 1911. The sample was taken at the first crosscut off the entry.

No further information available.

GLACIER CREEK.

Sample.—Anthracite; Bering River field; analysis No. 12734 (p. 20).

Locality.—Glacier Creek.

Notes.—This sample was taken on November 8, 1911, from a bowlder of anthracite

HARTLINE No. 1 MINE.

Sample.—Semibituminous coal; Bering River field; analysis No. 12724 (p. 20).

Mine.—Hartline No. 1.

Notes.—The bed was sampled at one point on November 6, 1911. No further information available.

HARTLINE No. 2 MINE.

Sample.—Semibituminous coal; Bering River field; analysis No. 12726 (p. 20).

Mine.—Hartline No. 2.

Notes.—The bed was sampled at one point on November 6, 1911. No further information available.

HARTLINE No. 3 MINE.

Sample.—Semibituminous coal; Bering River field; analysis No. 12725 (p. 20).

Mine.—Hartline No. 3.

Notes.—The bed was sampled at one point on November 6, 1911. No further information available.

HARTLINE No. 4 MINE.

Sample.—Semibituminous coal; Bering River field; analysis No. 12727 (p. 20).

Mine.—Hartline No. 4.

Notes.—The bed was sampled at one point on November 6, 1911. No further information available.

HARTLINE No. 5 MINE.

Sample.—Semibituminous coal; Bering River field; analysis No. 12728 (p. 20).

Mine.—Hartline No. 5.

Notes.—The bed was sampled at one point on November 7, 1911. No further information available.

HARTLINE No. 6 MINE.

Sample.—Semibituminous coal; Bering River field; analysis No. 12723 (p. 20).

Mine.—Hartline No. 6.

Notes.—The bed was sampled at one point on November 6, 1911. No further information available.

HUNTS MINE.

Sample.—Semibituminous coal; Bering River field; analysis No. 12729 (p. 20).

Notes.—A sample was taken at the mine on November 7, 1911. No further information available.

MOUNT ANN. TOP OF MOUNTAIN.

Sample.—Anthracite; Bering River field; analyses Nos. 12719 and 12733 (p. 20).

Locality.—Mount Ann.

Notes.—Sample 12719 was taken near the top, and sample 12733 at the top, of Mount Ann on November 7, 1911.

MOUNT ANN. SIDE OF MOUNTAIN.

Sample.—Semianthracite coal; Bering River field; analysis No. 12720 (p. 20).

Locality.—Mount Ann.

Notes.—This sample was taken on the side of Mount Ann on November 4, 1911.

MOUNT HAMILTON. McDONALD MINE.

Sample.—Semibituminous coal; Bering River field; analysis Nos. 12722, 12730, and 12731 (p. 21).

Mine.—McDonald, one-half mile south of U. S. L. M., Christopher group, near south line, top of ridge, on Mount Hamilton.

Coal bed.—Unnamed. Tertiary age, Kushtaka formation. Thickness about 15 feet. Nearly vertical.

The bed was sampled at three points in the mine on November 6, 7, and 8, 1911. Sample 12722 was taken at the face of the entry. No further information available.

ST. MARYS CREEK. HARTLINE NO. 7 MINE.

Sample.—Semibituminous coal; Bering River field; analysis No. 12721 (p. 21).

Mine.—Hartline No. 7.

Coal bed.—Unnamed. Tertiary age.

Notes.—The bed was sampled at one point on November 6, 1911. No further information available.

TROUT CREEK. CUNNINGHAM PROSPECT TUNNELS NO. 4 AND NO. 5.

Sample.—Semibitumious coal, analyses Nos. 15355, 15356, 15357, 15358, 15359, and 15360 (p. 21).

Locality.—Prospect tunnels No. 4 and No. 5, on the Cunningham claims, about one-fourth of a mile down the creek from the Cunningham cabin on Trout Creek. No. 4 tunnel is about 100 feet below No. 5 tunnel.

Coal bed.—Unnamed bed. Tertiary age, Kushtaka formation. The thickness of the bed as mined in these tunnels average 6 feet 9 inches. The roof was a treacherous shale; the floor a sandstone.

The bed was measured and sampled at four points in tunnel No. 4 and at two points in tunnel No. 5 by W. A. Selvig on November 12 to 22, 1912, as described below:

Section of coal bed in prospect tunnel No. 4, Cunningham claim, Trout Creek.

Section	A		B		C		D	
Laboratory No.	15355		15356		15357		15358	
Roof; shale, sandstone, and coal.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	8	2	5	...	..	1	0
Charcoal, dirt, and shale	[a]0	2½	[a]0	5	[a]2	3	[a]0	5
Coal	1	2	1	8	5	8½	1	11
Charcoal, shale, and "sulphur"	[a]0	1½	[a]0	4	[a]0	½	[a]0	3
Coal	1	6	...	..	2	7¼	3	8
Floor, sandstone and coal.								
Thickness of bed	5	8	4	10	10	7¼	7	3
Thickness of coal sampled	5	4	4	1	8	3¾	6	7

[a] Not included in sample.

Section A (sample 15355) was measured in the chute between timber sets 19 and 20, 76 feet from the drift mouth and 10 feet from the entry.

Section B (sample 15356) was measured on the rib of the right crosscut between timber sets 18 and 19, 72 feet from the drift mouth and 17 feet from the entry.

Section C (sample 15357) was measured on the rib near the face of the right crosscut between timber sets 11 and 12, 44 feet from the drift mouth.

Section D (sample 15358) was measured in the right crosscut between timber sets 25 and 26, 100 feet from the drift mouth and 11 feet from the entry.

Section of coal bed in prospect tunnel No. 5, Cunningham claim, Trout Creek.

Section	A		B	
Laboratory No.	15359		15360	
Roof, shale and coal.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	8	...	..
Shale and dirt	[a]0	8	...	..
Coal	2	7	4	0
Charcoal and shale	[a]1	7	[a]1	7
Coal	...	..	1	3
Floor; coal.				
Thickness of bed	7	6	6	10
Thickness of coal sampled	5	3	5	3

[a] Not included in sample.

Section A (sample 15359) was measured in the chute to the left of the entry between timber sets 21 and 22, 84 feet from the drift mouth and 17 feet up chute.

Section B (sample 15360) was measured in the left crosscut between timber sets 13 and 14, 52 feet from the drift mouth and 5 feet from the entry.

The coal beds of the Trout Creek region are described in U. S. Geol. Survey Bull. 335, 1908, p. 72.

MATANUSKA.

ANTHRACITE RIDGE. OUTCROP.

Sample.—Semianthracite coal; Matanuska field; analysis No. 12745 (p. 22).

Locality.—Anthracite Ridge, Matanuska Valley.

Coal bed.—Not named.

Notes.—The bed was sampled on November 1, 1911.

The coal beds of the Anthracite Ridge region are described in U. S. Geol. Survey Bull. 327, 1907, pp. 52-56.

ANTHRACITE RIDGE. OUTCROP.

Sample.—Bituminous coal; Matanuska field; analysis No. 12750 (p. 22).

Locality.—Anthracite Ridge, Matanuska Valley.

Coal bed.—Not named.

Notes.—This bed was sampled on October 27, 1911.

The coal beds of the Anthracite Ridge region are described in U. S. Geol. Survey Bull. 327, pp. 52-56.

ANTHRACITE RIDGE. OUTCROP.

Sample.—Semibituminous coal; Matanuska field; analysis No. 12753 (p. 22).

Locality.—Anthracite Ridge.

Coal bed.—Unnamed.

Notes.—The bed was sampled on October 24, 1911.

The coal beds of the Anthracite Ridge region are described in U. S. Geol. Survey Bull. 327, 1907, pp. 52-56.

ANTHRACITE RIDGE. OUTCROP.

Sample.—Semianthracite coal; Matanuska field; analysis No. 12746 (p. 22).

Locality.—Anthracite Ridge, Matanuska Valley.

Coal bed.—Not named.

Notes.—The bed was sampled at the outcrop near the middle of the south face of the ridge. Elevation (by barometer), 3,700 feet.

The coal beds of the Anthracite Ridge region are described in U. S. Geol. Survey Bull. 327, 1914, pp. 52-56.

ANTHRACITE RIDGE. OUTCROP.

Sample.—Semianthracite coal; Matanuska field; analysis No. 12757 (p. 22).

Locality.—Outcrop on Anthracite Ridge.

Coal bed.—Not named.

Notes.—The bed was sampled on October 26, 1911. The sample was taken at the outcrop, near the middle of the south slope. Elevation (by barometer), 4,100 feet.

The coal beds of the Anthracite Ridge region are described in U. S. Geol. Survey Bull. 327, 1914, pp. 52-56.

CHICKALOON CREEK. OUTCROP.

Sample.—Bituminous coal; Matanuska field; analyses Nos. 12752 and 12764 (p. 22).

Locality.—Outcrop at bend of Chickaloon Creek, one-half mile above Watson Camp.[a]

Coal bed.—Not named. Tertiary age, Chickaloon formation.

The bed was measured and sampled by J. A. Holmes on November 1, 1911.

Beginning on the south side of a series of beds of shale and sandstone, 180 to 200 feet thick, and measuring southward along the face of the bluff from a nearly vertical bed of sandstone 12 feet thick, this section was as follows:

Section of coal bed ½ mile above Watsons Camp.

	Ft.	in.
Shale	0	9
Bony coal and streaks of coal	0	10
Impure and other coal	1	0
Coal	[a] 2	9
Shale	1	4
Coal	0	2
Shale	0	1
Coal	0	4
Shallow coal	0	9
Bone	0	1
Coal	1	0
Shale and sandstone ("horse")	9	6
Coal	0	4
Shale	1	0
Coal	0	6
Shale	0	4
Coal	0	9
Shale with streaks of coal	1	4
Coal	0	8
Shale	0	2
Coal	[b] 4	10
	28	6

[a] Included in sample 12753. [b] Included in sample 12764.

Notes.—For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 327, 1907, p. 44; Bull. 500, 1912, p. 78.

CHICKALOON CREEK. WATSON TUNNEL NO. 4.

Sample.—Semibituminous coal; Matanuska field; analysis No. 12758 (p. 22).

Locality.—Watson Tunnel No. 4, near Watson Camp, Chickaloon Creek.

Coal bed.—Unnamed. Tertiary age, Chickaloon formation. Thickness, about 10 feet. The bed contained 4 feet 6 inches of crushed coal and two or three bands of bony coal. Strike, east and west; dip, 80° N. Floor and roof, carbonaceous shale, much jointed and slickensided.

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[a] See Martin's section, U. S. Geol. Survey Bull. 327, p. 44, Nos. 2 and 6.

Notes.—The sample was taken in the west crosscut off the entry, 111 feet from the entry mouth. The crosscut to the east had caved and the face could not be reached.

The coal beds in the vicinity of Chickaloon Creek are described in U. S. Geol. Survey Bull. 500, 1912, pp. 77–81.

Chickaloon Creek. Prospect.

Sample.—Semibituminous coal; Matanuska field; analysis No. 12761 (p. 22).

Locality.—Chickaloon Creek.

Coal bed.—Unnamed. Tertiary age, Chickaloon formation.

Notes.—The bed was sampled on November 2, 1911.

The coal beds in the vicinity of Chickaloon Creek are described in U. S. Geol. Survey Bull. 500, 1912, pp. 77–81.

Chickaloon Creek. Tunnel B.

Sample.—Semibituminous coal; Matanuska field; analysis Nos. 12756 and 12760 (pp. 22 and 23).

Locality.—Tunnel B, on Chickaloon Creek.

Coal bed.—Unnamed. Tertiary age, Chickaloon formation.

Notes.—The bed was measured and sampled at one point in the entry on October 25, 1911, as described below:

Section of coal bed in Tunnel B, Chickaloon Creek.

	Ft.	*in.*
Coal, impure	[a]3	3
Bone	0	4
Coal	1	0
Bone, lenticular	0	6
Coal	1	0
Shale	1	0
Coal	1	1
Shale	0	3
Coal	1	8
Shale	1	1
Coal, good	[b]3	10

[a] Included in sample 12760.

[b] Included in sample 12756.

Notes.—For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 327, 1907, p. 5; Bull. 500, 1912, p. 79.

Coal Creek. Prospect.

Sample.—Bituminous coal; Matanuska field; analysis No. 12744 (p. 23).

Locality.—Prospect on Coal Creek.

Coal bed.—Unnamed. Tertiary age, Chickaloon (?) formation.

Notes.—The bed was sampled on October 27, 1911.
available.

The coal beds in the vicinity of Coal Creek are described in U. S. Geol. Survey Bull. 500, 1912, pp. 77–78.

Coal Creek. Prospect.

Sample.—Bituminous coal; Matanuska field; analysis No. 12748 (p. 23).

Locality.—Prospect on Coal Creek.

Coal bed.—Unnamed. Tertiary age, Chickaloon (?) formation.

Notes.—The bed was sampled on November 7, 1911.

The coal beds in the vicinity of Coal Creek are described in U. S. Geol. Survey Bull. 327, 1912, pp. 77–78.

COAL CREEK (?).

Sample.—Bituminous coal; Matanuska field; analysis No. 12743 (p. 23).

Note.—No further information available.

MATANUSKA RIVER. O'BRIEN CLAIMS.

Sample.—Bituminous coal; Matanuska field; analysis Nos. 12755 and 12762 (p. 23).

Locality.—Outcrop 1 mile south of Matanuska River, on O'Brien group of claims.

Coal bed.—Unnamed. Tertiary age, Chickaloon formation.

About 1 mile south of Matanuska River the bed outcrops on the north face of a bluff, about 555 feet above the level of Matanuska River, probably near the 1,600-foot contour line and southwest of the east end of Anthracite Ridge. The coal is exposed just east of a shallow ravine extending up the face of this bluff. Strike, N. 75° E., dip 23° SE.

The bed was measured and sampled at this point by J. A. Holmes in September, 1911, as described below:

Section of coal bed, north face of bluff.

	Ft.	*in.*
Carbonaceous shale	0	12
Coal	0	4
Shale	0	½
Coal	0	10
Shale	1–2	0
Coal	0	9
Shale	0	2
Coal	0	10
Bony coal	0	1½
Coal	2	5
Shale	0	1–2 (?)
Coal	1	1
Bony coal	0	1
Sandstone	0	5–7
Coal	0	7
Bony coal	0	2½
Sandy shale and bony coal	0	2–3
Coal	1	2
Bone	0	1
Coal	4	1
Bone	0	1
Coal	0	2
Bone	0	½
Coal	0	10
Coal, slightly bony	0	2
Coal	0	8
Coal, slightly bony	0	10
Shale	0	2–6
Coal	1	0
Shale, with streaks of coal	1	0
Coal	1+	0

The base of this bed was covered up. The samples include the entire section as given above except the bottom 12 inches.

Sample 12755 represents the upper part of the bed, and sample 12762 the lower part of the bed.

MATANUSKA RIVER. O'BRIEN CLAIMS.

Sample.—Bituminous coal; Matanuska field; analysis No. 12763 (p. 23).

Locality.—Outcrop, on the O'Brien group of claims, Matanuska River.

Coal bed.—Unnamed. Tertiary age, Chickaloon formation.

The bed was measured and sampled by J. A. Holmes on November 2, 1911, as described below:

Section of coal bed at outcrop, south of Matanuska River.

Laboratory No.	12763	
	Ft.	*in.*
Carbonaceous shale.		
Coal	0	10
Shale	4	0
Bony coal with streaks of good coal	8	3
Shale	0	10
Coal	0	9
Bony coal	0	7
Shale with irregular bands of coal	3	5
Coal	3	3
Bony coal	1	7
Shale	0	5
Coal with impure or bony bands	1	5
Shale	3	5
Coal with bony bands	1	6
Shale	1	3
Bony coal	1	6
Coal	1	7
Shale.		

The sample was taken at an outcrop in the west face and bed of a ravine, about 100 feet below the top of the bluff. The point where the sample was taken is one-fourth of a mile south of the point where samples 12755 and 12762 were taken, and is southwest of the east end of Anthracite Ridge. Strike, N. 60 W., dip 20° to 40° SW.

MATANUSKA RIVER. O'BRIEN CLAIMS.

Sample.—Bituminous coal; Matanuska field; analysis No. 12747 (p. 23).

Locality.—Outcrop, O'Brien claims, Matanuska River.

Coal bed.—Unnamed. Tertiary age, Chickaloon formation.

The bed was sampled at one point on October 27, 1911.

Notes.—The sample was taken near and on the west side of the gulch, and 35 to 55 feet above the point where sample 12763 was taken. The bed was partly exposed and had apparently the same strike and dip as the bed represented by sample 12763. A section of the part exposed, measured from above downward, was as follows: Coal, 3½ to 4 feet; shale, 6 to 10 inches; coal, 1 to 3 feet.

MATANUSKA RIVER. O'BRIEN CLAIMS.

Sample.—Bituminous coal; Matanuska field; analysis No. 12749 (p. 23).

Locality.—Outcrop about 1 mile south of Matanuska River, on O'Brien claim.

Coal bed.—Unnamed. Tertiary age, Chickaloon formation.

The bed was measured and sampled at one point on October 27, 1911, as described below:

Section of coal bed at outcrop, 1 mile south of river.

Laboratory No.	12749	
	Ft.	*in.*
Shale with streaks of coal	[a] 2	7
Coal	2	10
Shale	[a] 0	2-5
Coal	[a] 1	0
Shale with bands of coal	[a] 2	0
Shale	[a] 4-5	0
Carbonaceous shale with streaks of coal	[a] 1-2	0

[a] Not included in sample.

The sample was taken from an outcrop in the east face of the gulch, and 50 to 75 feet above the point (across strata) where sample 12763 was taken and about 50 feet below top of bluff.

MATANUSKA RIVER. O'BRIEN CLAIMS.

Sample.—Bituminous coal; Matanuska field; analysis No. 12751 (p. 23).

Locality.—Outcrop one-half mile south of Matanuska River, near O'Brien cabin.

Coal bed.—Not named. Tertiary age, Chickaloon formation.

The bed was measured and sampled at one point by J. A. Holmes on November 1, 1911.

Sample 12751 was taken at a point about one-half mile south of Matanuska River, near a small stream which flows north (toward river) by the T. E. O'Brien cabin on the O'Brien coal claim and some 300 yards up this stream from the point where sample 12754 was taken in the west or right face of the bluff. The bed is exposed from a few feet above the level of the creek to near the top of the bluff. The bed here has a strike nearly east and west and a dip 60° to 70° NNW. The section measured from the north side southward was about as follows:

Section of coal bed, near O'Brien cabin.

Laboratory No.	12751	
	Ft.	*In.*
Shale.		
Coal, impure	[a] 1	0
Shale	[a] 0	4–6
Coal with few thin bony bands	6	0
Igneous rock	[a] 6–8	0
Coal with occasional bands of shale and bone (probably 75 per cent coal)	[a] 14	0
Shale	[a] 1–2	0
Altered rock, light-colored, earthy.		

[a] Not included in sample.

The shale and other rock strata on south side of the bed were faulted.

MATANUSKA RIVER. O'BRIEN CLAIMS.

Sample.—Bituminous coal; Matanuska field; analysis 12754 (p. 23).

Locality.—O'Brien claim, one-half mile south of Matanuska River.

Coal bed.—Unnamed. Tertiary age, Chickaloon formation.

The bed was measured and sampled at one point by J. A. Holmes on October 24, 1911.

Sample 12754 was taken from an outcrop on the west slope of a gulch (facing east), one-half mile south of Matanuska River and a short distance up a small stream from the cabin of T. E. O'Brien. The bed outcrops about 100 feet above the level of the creek and shows thick irregular beds of coal interbedded with thin irregular layers of ferruginous sandstone(?), with other thin or nearly vertical beds that resemble somewhat the eruptive rocks seen elsewhere in this field. The beds of coal are irregular and in places massive, and stand out prominently in the natural exposure. Strike nearly east and west. The bed is nearly vertical.

A section across the main beds of coal measured from north to south was as follows:

Section of coal bed one-half mile south of Matanuska River.

Laboratory No.	12754	
	Ft.	*in.*
Shale, with 5 to 10 inch bands of "iron ore" in more or less nodular masses.		
Coal	3	6
Bony coal	[a] 0	11
Carbonaceous shale	[a] 0	6–10
Coal	[a] 0	6–10
Shale	[a] 0	6–10
Coal, pinching out in places	[a] ½–1	0
Shale	[a] ½–2	0
Coal, with irregular bands of bony coal	[a] 1½–2	0
Shale, with streaks of coal	[a] ½–1	0
Impure bony coal	[a] 4	0
Shale, with nodular bands of "iron ore"	[a] 5	10
Bony coal in lenticular masses, 1 to 6 inches thick interbedded in carbonaceous shale	[a] 19	0
Shale, dark colored, impregnated and in places interbedded with nodular bands (1 to 8 inches thick) of "iron ore"	30–40	0

[a] Not included in sample.

MATANUSKA RIVER. O'BRIEN CLAIMS.

Sample.—Bituminous coal; Matanuska field; analysis No. 12759 (p. 23).

Locality.—Outcrop on O'Brien claim, about one-half mile south of Matanuska River.

Coal bed.—Unnamed. Tertiary age, Chickaloon formation.

The bed was measured and sampled at one point by J. A. Holmes on November 2, 1911.

Sample 12759 was taken from an outcrop on the west side and near top of bluff (facing east) across the gulch from the point where sample 12751 was taken. It is probably the same bed. The exposure was prominent, but the boundaries were not clearly defined. A section measured from the north side southward was as follows:

Section of coal bed, west side of bluff.

	Ft.	*in.*
Soft black shale	0	4
Coal	0	8
Shale	1	8
Coal	5	[a] 10
Shale	0	8
Impure coal	0	6
Shale	0	10
Coal	0	9
Shale	0	3
Coal	0	10
Bony coal	0	6
Shale	0	5
Coal	2	6
Shale	0	1
Coal	0	11
Bone	0	1
Coal	2	9
Shale, with irregular bands of bony coal	1-3	0
Coal	0	1-18
Bony coal with streaks of shale	1-3	0
Coal	1	4
Shale	0	2
Coal	1	2
Bony coal	1	2
Coal	0	8

[a] Or more.

Notes.—In the upper part of the bed the coal appears to fold on itself southward and thus gives the appearance of being 25 to 30 feet thicker than would be indicated by the above section.

YOUNG CREEK. OUTCROP.

Sample.—Bituminous coal; Matanuska field; analysis No. 11382 (p. 23).

Locality.—Outcrop near Young Creek; longitude 148° 42′ 12″ west, latitude 61° 47′ 40″ north.

Coal bed.—Unnamed. Tertiary age, Chickaloon formation. Dip, 54° SE.; strike, N. 67° E.

The bed was sampled by G. C. Martin on August 11, 1910, in the vicinity of Young Creek. The sample was cut from a natural outcrop after cleaning off with a pick as far as possible all weathered coal. The bed was not measured at this point, but a typical section measured on the north face of Red Mountain about 4 miles north of the mouth of Young Creek is as follows:

Section of coal bed on north face of Red Mountain, about 4 miles north of mouth of Young Creek.

	Ft.	in.
Roof, shale with ironstone nodules.		
Coal (exposure obscure; thickness estimated)	7	5
Shale	0	1½
Coal	1	2
Shale	0	1
Coal	0	6
Shale	0	1½
Coal	0	7
Shale	0	¾
Coal	0	4
Shale	0	½
Coal	0	2
Shale	0	¾
Coal	2	0
Thickness of bed	12	8

Notes.—Another coal bed at least 5 feet and possibly 10 feet thick is poorly exposed about 30 feet stratigraphically above this, the intervening rocks being shale with ironstone nodules and sandstone. About 30 feet higher another bed of unknown thickness is present and 50 feet still higher indications of another bed were seen.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 480, p. 129.

COLORADO.

ADAMS COUNTY.

BENNETT. THOMAS MINE.

Sample.—Subbituminous coal; Denver region; analysis No. 13141 (p. 23).

Mine.—Thomas, in sec. 12, T. 1 S., R. 61 W., about 25 miles northeast of Bennett, on the Union Pacific Railroad.

Coal bed.—No name. Cretaceous age, Laramie formation.

The bed was measured and sampled at one point in the mine by G. B. Richardson on October 23, 1911. The sample was from a 4-foot bench of coal. There was coal, reported 2 feet thick, above the bench sampled, and coal, reported 2 feet thick, below it.

The sample was obtained at the working face in the mine, about 100 feet south of the bottom of the shaft, which is 110 feet deep.

Notes.—The coal beds of the Denver Basin are described in U. S. Geol. Survey Bull. 381, 1910, pp. 297–306.

BOULDER COUNTY.

LAFAYETTE. SIMPSON MINE.

Sample.—Subbituminous coal; Denver region; analysis No. 15165 (p. 24).

Mine.—Simpson, a shaft mine in the Northern district; in T. 1 S., R. 69 W.; at Lafayette, on the Colorado & Southern Railway.

Coal bed.—Known as the Simpson or Lower. Cretaceous age, Laramie formation. The bed is practically horizontal. Thickness, 6 to 14 feet, average 12 feet. The characteristic impurities are local bone and "sulphur" partings. Roof, hard gray shale, overlain by sandstone; floor, soft smooth underclay. Particles of the roof and floor did not mix with the coal in loading.

The bed was measured and sampled at one point in the mine by J. C. Roberts on November 15, 1912, as described below:

Section of coal bed in Simpson mine.

Laboratory No.	15165	
Roof, shale.	*Ft.*	*In.*
Coal	[a]4	2
Coal	4	2
Bone	0	3
Coal	2	5
Floor, clay.		
Thickness of bed	11	0
Thickness of coal sampled	6	10

[a] Not included in sample; is good coal but is mined only when mining pillars on retreating.

Section A (sample 15165) was measured at the face of room 1, off entry 125.

Notes.—The mine is worked by the room-and-pillar system. At the time of sampling, the coal was undercut by air punching machines and broken down with black blasting powder. The undercuttings were loaded out with the coal. All coal was shipped as run-of-mine. There were three loading tracks with a capacity of 30 empty and 30 loaded cars. The mine had approximately 300 acres of undeveloped coal. The daily output was about 450 tons, most of which was taken from pillars.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 54, 55, 398, 399.

For results of tests of other coal from this mine, see mention of specific tests as follows—Steaming tests: U. S. Geol. Survey Bull. 261, p. 80; producer-gas tests: U. S. Geol. Survey Bull. 261, p. 90; Bureau of Mines Bull. 13, pp. 108, 272; briqueting tests: U. S. Geol. Survey Bull. 261, p. 152; coking tests: Bureau of Mines Bull. 5, pp. 21, 51.

DELTA COUNTY.

BOWIE. KING MINE.

Sample.—Bituminous (noncoking) coal; Grand Mesa field; analysis No. 15213 (p. 24).

Mine.—King, a drift mine in the Somerset district, 1 mile from Bowie, on the Denver & Rio Grande Railroad.

Coal bed.—King. Cretaceous age, Mesaverde formation. Average thickness, 10 feet; dip, N. 23° 33′ E.; cleat, N. 82° E. Occasional faults and frequent rolls or horsebacks are encountered. The immediate roof, a bony coal 2 to 6 feet thick, is overlain by 15 feet of sandstone. The floor is bony coal 2 to 6 inches thick, underlain by sandstone. Particles of the roof and floor did not become mixed with the coal in mining or loading. Workable beds of coal occur 20 feet above and 80 feet below the King bed. Cover at point of sampling, 900 feet.

The bed was measured and sampled at one point in the mine by W. D. Schofield on December 7, 1912. The section measured 10 feet 2 inches of clear coal, all of which was included in the sample.

The sample was cut from the face of room 5, off left entry 3, 3,600 feet from the mine mouth.

Notes.—The mine is worked on the room-and-pillar system. At the time of sampling the coal was undercut by hand and blasted down with FFF black blasting powder and a permissible explosive. The tipple had two shaking screens 8 feet wide by 12 feet long; one screen had 4-inch holes and the other had 1-inch holes, about 55 to 60 per cent of the coal passing the screens. No pickers were employed on the cars. The lumps were large and the coal had a good appearance on the cars. There were three loading tracks with a capacity of 30 empty and 60 loaded cars. The mine had a capacity of 1,000 tons a day; the average output being about 450 tons.

There were approximately 1,000 acres of unmined coal tributary to this tipple, about 33 per cent of the coal being obtained from advance workings.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 55, 401.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 341, 1909, p. 330.

DOMINGUEZ. WELLS GULCH MINE.

Sample.—Bituminous coal; Grand Mesa field; analysis No. 11106 (p.24).

Mine.—Wells Gulch; sec. 17, T. 4 S., R. 3 E.; in Wells Gulch, Gunnison Valley, about 2½ miles northeast of Dominguez, which is the nearest station on the Denver & Rio Grande Railroad.

Coal bed.—The geologic age of the bed has not been definitely determined. Thickness, 3 feet 11 inches; roof, sandstone; floor, shale. The bed was measured and sampled at one point in the mine by E. G. Woodruff, on October 19, 1910, as described below:

Section of coal bed in Wells Gulch Mine.

Laboratory No.	11106	
Roof, sandstone.	*Ft.*	*in.*
Coal	1	8
Shale, carbonaceous, or bone	[a] 2	3
Floor, shale:		
Thickness of bed	3	11
Thickness of coal sampled	1	8

[a] Not included in sample.

Sample 11106 was taken at the face of the entry, about 200 feet from the mine mouth. At the time of sampling, the mine was not being worked. The face had been exposed for several months, and was cleaned with a pick prior to sampling.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 471, 1912, p. 567.

EL PASO COUNTY.

CALHAN. MOSBY MINE.

Sample.—Subbituminous coal; Colorado Springs field; analysis No. 10732 (p. 24).

Mine.—Mosby, a shaft mine in sec. 18, T. 13 S., R. 62 W., 9 miles south of Calhan, which is the nearest station on the Chicago, Rock Island, & Pacific Railway.

Coal bed.—Mosby. Cretaceous age, Laramie formation. Thickness, 4 feet 8 inches. Roof, sandy shale. The bed was measured and sampled at one point in the mine by G. B. Richardson on August 3, 1910, as described below:

Section of coal bed in Mosby mine.

Laboratory No.	10732	
Roof, sandy shale.	*Ft.*	*in.*
Coal	1	2
Parting	[a] 0	1
Coal	1	6
Parting	[a] 0	3
Coal	1	8
Floor, not stated.		
Thickness of bed	4	8
Thickness of coal sampled	4	4

[a] Not included in sample.

The sample was taken about 30 feet from the bottom of the 40-foot shaft at the working face. The mine was worked intermittently.

PIKEVIEW. PIKEVIEW MINE.

Sample.—Subbituminous coal; Colorado Springs field; analysis No. 12099 (p. 24).

Mine.—Pikeview, a shaft mine in sec. 18, T. 13 S., R. 66 W., 1 mile north of Pikeview station, on the Denver & Rio Grande Railroad.

Coal bed.—A. Cretaceous age, Laramie formation. Average thickness, about 12 feet; dip, 4° N., 35° E. The bed contains numerous rolls. Cover at point of sampling, 181 feet. Roof, a thin blue shale, above which is a soft sandstone. The floor is a rough shale and a fire clay and is fairly hard.

The bed was measured and sampled at one point in the mine by J. C. Roberts, on April 12, 1911. The sample was taken at the face of north entry 2, 4,000 feet northwest of the shaft. The bed at this point was 10 feet 6 inches thick, but the sample represents only 7 feet 5 inches, the upper 3 feet of the bed being left for roof.

Notes.—Mining is by the room-and-pillar system. At the time of sampling the coal was undercut in the clay and blasted down with FFF black powder. The coal is soft and lignitic. About 60 per cent of the output was sold as run-of-mine. About 60 per cent of the coal would pass through a 4½-inch screen, leaving 40 per cent domestic lump. Pickers were employed both in the mine and on the car. There were three loading tracks with a capacity of 25 empty and 25 loaded cars. The output was 550 to 600 tons a day. The unmined area covered about 640 acres.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, pp. 322–332.

RAMAH. PURDON MINE.

Sample.—Subbituminous coal; Colorado Springs field; analysis No. 10741 (p. 24).

Mine.—Purdon; sec. 27, T. 11 S., R. 61 W.; 3 miles southwest of Ramah, which is the nearest station on the Chicago, Rock Island & Pacific Railway, and 6 miles northeast of Calhan.

Coal bed.—Locally known as Purdon. Cretaceous age, Laramie formation. Thickness, 6 feet 9½ inches. Roof, sandy shale. The bed was measured and sampled by G. B. Richardson on August 15, 1910, as described below:

Section of coal bed in Purdon mine.

Laboratory No.	10741	
Roof, sandy shale.	*Ft.*	*in.*
Coal	1	0
Shale, sandy	[a]0	1½
Coal, with 5 partings, each less than ⅛ inch thick	2	8
Shale, sandy, and coal	[a]0	2
Coal with 3 partings, each less than ⅛ inch thick	2	10
Floor, not stated.		
Thickness of bed	6	9½
Thickness of coal sampled	6	6

[a] Not included in sample.

The above sample was taken at the face of the workings, about 700 feet from the entry.

Notes.—The mine was worked intermittently.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 57, 408.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, pp. 319–335.

FREMONT COUNTY.

CHANDLER. CHANDLER MINE.

Sample.—Subbituminous coal; Canon City field; analyses Nos. 13393, 13394, and 13395 (pp. 24 and 25).

Mine.—Chandler, a shaft mine in sec. 22, T. 19 S., R. 70 W., near Chandler, on a spur of the Denver & Rio Grande Railroad.

Coal bed.—Canon. Cretaceous age, Vermejo formation. Average thickness, 4 feet 3 inches; dip, 4° north. The bed is free from faults but contains rolls and horsebacks. The roof is a "draw slate" 6 to 10 inches thick which comes down with the coal; above this "draw slate" is a cap rock. The floor is a soft, rough, shaly sand rock. Cover at points of sampling, about 700 feet.

The bed was measured and sampled at two points in the mine by J. C. Roberts on February 27, 1912, as described below:

Sections of coal bed in Chandler Mine.

Section	A.		B.	
Laboratory No.	13393		13394	
Roof, "draw slate."	*Ft.*	*in.*	*Ft.*	*in.*
Bone	[a]0	6	..	..
Coal, clean	4	2	4	9
Floor, shaly sand rock.				
Thickness of bed	4	8	4	9
Thickness of coal sampled	4	2	4	9

[a] Not included in sample.

Section A (sample 13393) was measured at the face of east entry 9.

Section B (sample 13394) was measured at the face of the main slope.

A composite sample was made by combining and mixing face samples 13393 and 13394. The ultimate analysis of this sample is given under laboratory No. 13395.

Notes.—The shaft is 484 feet deep. The coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by electric chain cutting machines and was shot down with a permissible explosive. "Forty per cent" dynamite was used for brushing the roof. The tipple was equipped with bar screens 18 feet long with 2-inch openings and a revolving screen 16 feet long with ¾-inch to 1¼-inch holes. About 40 per cent of the coal passed through the screens, none being shipped as run-of-mine. Pickers were employed on both picking table and car in loading. There were two loading tracks with a capacity of 36 empty cars and 30 loaded cars. The coal had a good appearance and contained large lumps. The average daily output of the mine was 550 tons. About 90 per cent of the coal was from advance workings and 10 per cent from pillars. The life of the mine would be probably 10 years.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 58, 414.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, p. 358.

ROCKVALE. ROCKVALE MINE.

Sample.—Subbituminous coal; Canon City field; analyses Nos. 11938, 11939, and 11941 (p. 25).

Mine.—Rockvale, a slope mine in secs. 25 and 36, T. 19 S., R. 70 W., and secs. 30 and 31, T. 19 S., R. 69 W., at Rockvale, on the Atchison, Topeka & Santa Fe Railway.

Coal bed.—Rockvale. Cretaceous age, Vermejo formation. Average thickness, 3 feet 6 inches; dip, 5° N. 60° W. The roof is sandstone and black shale, which is hard at some places and soft in others, is generally smooth, and does not adhere to the coal. The floor is a mixture of coal and clay and is called "blackjack"; it is tough and usually smooth. Cover, about 320 feet.

The bed was measured and sampled at two places in the mine by J. C. Roberts on March 7, 1911.

Sample 11938 was measured at the face of room 39, off the third dip, about 6,000 feet south from the shaft. The sample represents 3 feet 9½ inches of clear coal.

Sample 11939 was measured at the face of room 31, off the fourth dip, 6,200 feet south of the shaft. The sample represents 3 feet 9 inches of clear coal.

A composite sample was made by combining face samples 11938 and 11939. The results of an ultimate analysis of this sample are given under laboratory No. 11941.

Notes.—The long-wall method of mining is used. At the time of sampling all of the coal produced was from advance workings. The coal was undercut in the clay and broken down by black powder; "40 per cent" dynamite was used in brushing the roof. All of the coal was screened, none being sold as run-of-mine. About 43 per cent of the coal passed through a 3-inch screen, the screenings producing 18 per cent nut coal and 25 per cent slack; 57 per cent of the coal was sold as lump. Pickers were employed to pick the coal as it was loaded on the car. There were two loading tracks with a capacity of 100 empty cars and 50 loaded cars.

The daily output was about 900 tons; the unmined area tributary to the mine was about 1,200 acres.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 59, 415.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, p. 352.

GARFIELD COUNTY.

NEWCASTLE. VULCAN MINE.

Sample.—Bituminous coal; Grand Hogback field; analyses Nos. 12325, 12326, 12327, and 12420 (p. 25).

Mine.—Vulcan; Glenwood Springs district; sec. 1, T. 6 S., R. 91 W.; 2 miles east of Newcastle, on the Colorado Midland Railway.

Coal bed.—Allen. Cretaceous age, Mesaverde formation. Average thickness, about 14 feet. Roof, a poor quality of sandstone; floor, sandstone.

The bed was measured and sampled at three points in the mine by J. C. Roberts on June 14 and 18, 1911.

Sample 12325 was cut from the face of the east entry on the third level, 1,250 feet southeast of the mine mouth. The sample represents 10 feet 6 inches of coal from the lower part of the section.

Sample 12326 was cut from the face of the Allen entry on the second level, 1,700 feet from the mouth of the entry. The sample represents 10 feet of coal from the lower part of the section.

Sample 12420 was cut from the face of room 15, 250 feet up from the entry and 1,000 feet southeast of the mine mouth. The sample represents 10 feet of coal from the middle of the section.

A composite sample was made by combining and mixing face samples 12325 and 12326. An ultimate analysis of this sample is given under laboratory No. 12327.

Notes.—The coal is mined by the room-and-pillar system. The bottom bench of about 6 feet was first mined, then the top bench was taken down and pillars removed. At the time of sampling the entire output was shipped as run-of-mine except what was delivered by wagon for domestic purposes. The coal loaded on wagons was screened over bars 12 feet long with 2-inch openings. There were three loading tracks with a capacity of 20 empty cars and 25 loaded cars. The daily output was about

150 tons. The probable life of the mine was 50 years. All the coal mined was taken from advance workings.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 415, 1910, p. 112.

SOUTH CANYON. SOUTH CANYON MINE.

Sample.—Bituminous (noncoking) coal; Grand Hogback field; analyses Nos. 15166 and 15167 (p. 25).

Mine.—South Canyon, a drift mine in the Glenwood Springs district, in the NW. ¼ sec. 14, T. 6 S., R. 90 W., at South Canyon, 3 miles from the Colorado Midland Railway and connected with the railroad station by trolley line.

Coal bed.—Wheeler; Cretaceous age, Mesaverde formation. Average thickness, 18 feet; dip 55°. There are no cleats or faults but occasional rolls and horsebacks are encountered. The roof (hanging wall) is a hard black shale above which there is a sandstone cap rock. The floor is a smooth, soft, black shale. Particles of the roof and floor did not mix with the coal in mining. Seventy-two feet above this bed and 32 feet underneath are workable beds of coal. Cover at points of sampling, 500 to 550 feet.

The bed was measured and sampled at two points in the mine by J. C. Roberts on November 30, 1912, as described below:

Sections of coal bed in South Canyon Mine.

Section	A		B	
Laboratory No.	15166		15167	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Top coal	5	9	[a]10	0
Coal	([b])		7	8
Floor, shale.				
Thickness of bed	..	..	17	8
Thickness of coal sampled	5	9	7	8

[a] Not included in sample. [b] Not measured.

Section A (sample 15166) was measured in the main entry, 800 feet from the portal.

Section B (sample 15167) was measured at the face of the main entry, 1,600 feet from the portal.

Notes.—The mine is worked by the room-and-pillar system. At the time of sampling the coal was shot down with permissible explosives. None of the coal was shipped as run-of-mine. The coal was screened over bars 18 feet long with 4-inch openings, a shaker screen 14 feet long and 6 feet wide with 2½-inch openings and a shaker screen 14 feet long and 6 feet wide with 1-inch openings. Twenty per cent of the coal passed through the 1-inch screen. This fine coal was wasted, as there was no market for it. The coal was picked or trimmed on the car, and the loaded lump coal had a good appearance. There were three loading tracks with a capacity of 15 empty and 15 loaded cars. The unmined area tributary to this opening was about 1 mile long on the strike of the bed. The daily output was 350 tons.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 61, 421, 422.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 415, 1910, p. 130.

GUNNISON COUNTY.

SOMERSET. SOMERSET MINE.

Sample.—Bituminous coal; Grand Mesa field; analyses Nos. 12322, 12323, and 12324 (p. 25).

Mine.—Somerset; a slope mine in the Somerset district, near Somerset, on the Denver & Rio Grande Railroad.

Coal bed.—Not named. Cretaceous age, Mesaverde formation, Bowie shale. Thickness, 8 to 12 feet. Roof, sandstone; floor, bony coal about 8 inches thick.

The bed was measured and sampled at two points in the mine by J. C. Roberts on June 15, 1911, as described below:

Sections of coal bed in Somerset mine.

Section	A		B	
Laboratory No.	12322		12323	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	3	0	3	0
Coal, bony	[a]0	3	[a]0	3
Coal	8	9	4	9
Floor, bony coal.				
Thickness of bed	12	0	8	0
Thickness of coal sampled	11	9	7	9

[a] Not included in sample.

Section A (sample 12322) was measured at the face of room 46 off west entry 6, 10,000 feet north of the entry mouth.

Section B (sample 12323) was measured at the face of west entry 9, 4,500 feet north from the mouth.

A composite sample was made by combining face samples 12322 and 12323. The results of an ultimate analysis of this sample are given under laboratory No. 12324.

Notes.—At the time of sampling the coal was shot down with black powder. The output was 1,000 tons a day.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 341, 1909, p. 330.

HUERFANO COUNTY.

OAKVIEW. OAKDALE MINE.

Sample.—Bituminous coal; Trinidad field; analyses Nos. 13526, 13527, 13528, and 13529 (p. 26).

Mine.—Oakdale, a slope and drift mine in the La Veta district, at Oakview, on a spur of the Denver & Rio Grande Railroad, 7 miles west of La Veta.

Coal bed.—Not named. Cretaceous age, Vermejo formation. Average thickness, 12 feet; dip, 15° W.; cleat, north and south. Occasional faults, rolls and horsebacks are encountered. Roof, shale, overlain by sandstone; floor, soft smooth shaly underclay. Particles of both the roof and floor occasionally mixed with the coal in mining. Cover at points of sampling, 100 to 500 feet.

The bed was measured and sampled at three points in the mine by J. C. Roberts on March 8, 1912, as described below:

Sections of coal bed in Oakdale mine.

Section	A		B		C	
Laboratory No	13526		13527		13528	
Roof, shale and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	..	..	(a)		(a)	
Coal, clean	13	10	6	6	[b]3	6
Natural coke	..	..	..	..	3	6
Floor, shale.						
Thickness of coal mined	13	10	6	6	7	0
Thickness of coal sampled	13	10	6	6	3	6

[a] Not measured. [b] Not included in sample.

Section A (sample 13526) was measured at the face of room 11, off the second "cross," off the main entry.

Section B (sample 13527) was measured at the face of the main entry. The top coal was not mined in advance work and could not be measured.

Section C (sample 13528) was measured at the face of north entry 4. The top coal was not mined in advance work and could not be measured.

A composite sample was made by combining samples 13526 and 13527. The ultimate analysis of this sample is given under laboratory No. 13529.

Notes.—The mine is worked on the room-and-pillar panel system. At the time of sampling the coal was undercut in the coal with punching machines and shot down with black powder. The machine cuttings were loaded with the coal. Twenty-five per cent of the coal was shipped as run-of-mine. The tipple was equipped with two shaking screens each 12 feet long by 5 feet wide, one having 3-inch and the other 1-inch holes, and a revolving screen 12 feet long with ¾-inch holes. Forty per cent of the coal passed through the ¾-inch screen. No pickers were employed at the tipple. There were three loading tracks with a capacity of 20 empty cars and 30 loaded cars. The coal had an excellent appearance and the lumps were large. The average daily production of the mine was 1,200 tons, with a maximum capacity of 1,800 tons. All the coal was from advance workings. The probable life of the mine was 30 years.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 65, 435.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, p. 419.

JACKSON COUNTY.

COALMONT. RIACH MINE.

Sample.—Subbituminous coal; North Park field; analysis No. 12601 (p. 26).

Mine.—Riach, a slope mine in the Coalmont district, in the NW. ¼ of SW. ¼ sec. 24, T. 7 N., R. 81 W.; at Coalmont, on the extension of the Laramie, Hahn's Peak & Pacific Railway.

Coal bed.—Known as the Riach. Late Cretaceous or early Tertiary age, Coalmont formation. Both roof and floor are coal. Thickness, reported 77 feet; dip, 18° northeast.

The bed was sampled at one point in the mine by A. L. Beekly on August 18, 1911. The sample was collected from the north breast on a freshly stripped, fairly dry face, and represents 7 feet of coal.

Notes.—The mine had a capacity of about 800 tons a day. A considerable part of the output was sold for domestic use in the park and in various towns along the main line of the Union Pacific Railroad.

COALMONT. MITCHELL MINE.

Sample.—Subbituminous coal; North Park field; analysis No. 12539 (p. 26).

Mine.—Mitchell; in SW. ¼ SE. ¼ sec. 24, T. 8 N., R. 82 W.; 8 miles northwest of Coalmont and 15 miles southwest of Walden.

Coal bed.—Mitchell. Late Cretaceous or early Tertiary age, Coalmont formation, The bed consists of an upper bench 8 feet thick separated from a lower 2⅓-foot bench by 10 inches of carbonaceous shale. The shale parting is used as the floor of the mine and about 7 feet of the upper bench is mined, the upper foot being left for roof. Dip, 11° northeast. Strike, north 40° west.

The bed was measured and sampled by A. L. Beekly on August 16, 1911, as described below:

Section of coal bed in the Mitchell mine.

Laboratory No	12539	
Roof, dark, micaceous, sandy shale.	*Ft.*	*in.*
Coal, clean and black	3	6
Coal, soft and laminated	2	6
Coal, hard, black	2	0
Shale, carbonaceous, laminated	[a] 0	10
Coal, hard	[a] 2	4
Floor, shale.		
Thickness of bed	11	2
Thickness of coal sampled	8	0

[a] Not included in sample.

The sample was collected in the entry, 275 feet northeast of the mine mouth. The mine was worked intermittently.

HIGHO. MONAHAN MINE.

Sample.—Subbituminous coal; North Park field; analysis No. 12540 (p. 26).

Mine.—Monahan, in the SE. ¼ SE. ¼ sec. 31, T. 10 N., R. 81 W., 4 miles northwest of Higho post office and 18 miles northwest of Walden.

Coal bed.—Monahan. Late Cretaceous or early Tertiary age. Thickness, 4 feet 6 inches; strike N. 15° W.; dip, 22° N.; roof, buff shale; floor, sandstone.

The bed was sampled at one point in the mine by A. L. Beekly on August 3, 1911, the sample representing 4 feet 6 inches of coal.

Notes.—The sample was collected from the face of the entry, which was 125 feet long and was on the dip of the bed. The coal at the face was hard, black, and clean.

WALDEN. MCCALLUM MINE.

Sample.—Subbituminous coal; North Park field; analysis No. 12774 (p. 26).

Mine.—McCallum, a slope mine; McCallum anticline district, NE. ¼ NE. ¼ sec. 18, T. 9 N., R. 78 W.; about 5 miles northeast of Walden.

Coal bed.—Sudduth. Late Cretaceous or early Tertiary age. Base of Coalmont formation. Thickness, reported 35 feet; strike, north 50° west; dip, 18° northeast. Both roof and floor are coal.

The bed was sampled by A. L. Beekly on October 16, 1911.

Notes.—The sample was collected from a fresh face in the entry, 50 feet from the foot of the slope, which was 50 feet long and dips 18°. The sample represents 7 feet of coal.

WALDEN. MARR MINE.

Sample.—Subbituminous coal; North Park field; analysis No. 12415 (p. 27).

Mine.—Marr; McCallum anticline district, in the NW. ¼ NE. ¼ sec. 35, T. 9 N., R. 78 W.; 8½ miles east of Walden.

Coal bed.—Sudduth. Late Cretaceous or early Tertiary age, Coalmont formation. The bed is reported to be 53 feet thick and to consist throughout of a good grade of coal, which is free from partings except for several 1-inch sandy layers near the top. Dip, 56° east.

The bed was sampled by A. L. Beekly on June 27, 1911, the sample representing 6 feet of coal.

Notes.—The sample was collected from a clean surface just inside the entry mouth and about 15 feet from the top of the bed. Forty feet of this bed is exposed in an open cut; the base is covered and inaccessible. The mine consisted of two entries; both were filled with water at the time of sampling.

WALDEN. SUDDUTH MINE.

Sample.—Subbituminous coal; North Park field; analysis No. 12414 (p. 27).

Mine.—Sudduth; a drift mine in the McCallum anticline district, in NW. ¼ SW. ¼ sec. 15, T. 9 N., R. 78 W.; 7½ miles east of Walden.

Coal bed.—Sudduth. Late Cretaceous or early Tertiary age, Coalmont formation. Thickness, reported 50 feet; dip, 85° northeast.

The bed was sampled by A. L. Beekly on June 27, 1911.

Notes.—The mine consisted of an entry or room driven horizontally about 70 feet into the bed. The coal exposed in the mine was clean, hard, and glossy black. The sample was collected from a clean face at the end of the entry and represents 20 feet of coal.

WALDEN. WINSCOM MINE.

Sample.—Subbituminous coal; North Park field; analysis No. 12775 (p. 27).

Mine.—Winscom; McCallum anticline district, SW. ¼ NE. ¼ sec. 14, T. 8 N., R. 78 W.; 10½ miles southeast of Walden.

Coal bed.—Winscom. Late Cretaceous or early Tertiary age, Coalmont formation. Roof, coal; floor, shale. Strike, nearly north and south. Dip, 25° to 30° east.

The bed was measured and sampled by A. L. Beekly on October 16, 1911, as described below:

Section of coal bed in the Winscom mine.

Laboratory No.	12775	
Roof, coal.	*Ft.*	*in.*
Clay	[a]0	10
Coal	0	4
Clay, carbonaceous	[a]0	3
Coal	0	4
Shale, carbonaceous	[a]0	2
Coal	1	0
Shale, carbonaceous	[a]0	7
Coal	0	11
Floor, carbonaceous shale.		
Thickness of bed	4	5
Thickness of coal sampled	2	7

[a] Not included in sample.

Notes.—Sample was collected in the entry, 125 feet from the mouth of the mine.

JEFFERSON COUNTY.

MORRISON. WHITE ASH MINE.

Sample.—Subbituminous coal; Denver region; analysis No. 16615 (p. 27).

Mine.—White Ash, a shaft mine in secs. 23, 24, 25, and 26, T. 4 S., R. 70 W., 2 mile. from Morrison, 1 mile from Colorado & Southern Railway switch. Not on any railroads

Coal bed.—Jumbo. Cretaceous age, Laramie formation. Average thickness, 15 feet; dip, 86°. No rolls or horsebacks have been encountered. Roof (hanging wall), clay 3 inches thick, overlain by shale; above the shale is a sandstone cap rock. Small particles of the shale gradually break off on exposure. Floor (foot wall), a hard smooth underclay. Particles of the floor and roof did not mix with the coal in loading.

The bed was measured and sampled at one point in the mine by J. C. Roberts on March 12, 1913, as described below:

Section of coal bed in White Ash mine.

Laboratory No.	16615	
Roof, clay and shale.	*Ft.*	*in.*
Coal	[a]5	6
Coal	10	6
Floor, underclay.		
Thickness of bed	16	0
Thickness of coal sampled	10	6

[a] Not included in sample.

Sample 16615 was taken in a stope, 30 feet from the second level and 75 feet from the shaft.

Notes.—At the time of sampling the coal was shot off the solid with black blasting powder. The coal was screened through a bar screen 10 feet long with 1-inch openings, about 10 to 15 per cent passing through the screen. No pickers were employed. The coal had a good appearance on the wagons. The area to be mined was approximately 2 miles long on the strike. The output in the immediate future was to be derived from advance workings.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, p. 297.

LA PLATA COUNTY.

DURANGO. SAN JUAN MINE.

Sample.—Bituminous coal; Durango field; analyses Nos. 14770, 14771, 14772 (p. 27).

Mine.—San Juan, a drift mine 1¼ miles southeast of Durango, on the Rio Grande Southern Railroad.

Coal bed.—Not named. Cretaceous age, Mesaverde formation. Thickness, 3 feet 6 inches to 4 feet 10 inches, average about 4 feet. About the middle of the bed is a layer of bony coal 2 inches thick. The roof is a soft gray shale, which slacks on exposure; above the shale is about 10 inches of bony coal. The immediate roof sticks to the coal and often falls in the rooms. The floor is a hard rough shale. Cover at points of sampling, 200 to 225 feet.

The bed was measured and sampled at two points in the mine by W. D. Schofield on September 11, 1912, as described below:

Sections of coal bed in San Juan mine.

Section	A		B	
Laboratory No.	14770		14771	
Roof, shale, and coal.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	6	4	3
Bone, to shale	[a] 0	1	[a] 0	2
Coal	1	6	0	9
Floor, black shale.				
Thickness of bed	3	1	5	2
Thickness of coal sampled	3	0	5	0

[a] Not included in sample.

Section A (sample 14770) was measured at the face of the main entry, 1,200 feet west of the mine mouth.

Section B (sample 14771) was measured in room 6, off the first dip entry south, 300 feet from the mine mouth.

A composite sample was made by combining and mixing samples 14770 and 14771. The results of an ultimate analysis of this sample are given under laboratory No. 14772.

Notes.—The coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by hand and shot down with permissible explosives. Dynamite was used for brushing the floor and roof. The undercuttings were loaded out with the coal. About 30 per cent of the output was shipped as run-of-mine. Bar screens 30 feet long with 1¼-inch openings were used; about 60 per cent of the coal passed the screen. The screenings were coked. The loading track had a capacity of 7 cars and the "empty" track 25 cars. The daily output was about 160 tons. About one-half of the output was from advance workings and one-half from pillars. There was about 140 acres to be mined from this opening.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 315, 1910, p. 384.

HESPERUS. HESPERUS MINE.

Sample.—Bituminous coal; Durango field; analyses Nos. 14773, 14774, and 14775 (p. 27).

Mine.—Hesperus, a drift mine in sec. 14, T. 35 N., R. 11 W., at Hesperus, on the Rio Grande Southern Railroad.

Coal bed.—Hesperus. Cretaceous age, Mesaverde formation. Thickness, 4 feet 6 inches to 6 feet; average, about 5 feet. About 2 feet from the roof is an irregular shale parting. The immediate roof is smooth and does not fall in the rooms; the main cap rock is 2 inches to 2 feet above the coal. The floor is a soft smooth clay which mixed with the coal in loading. Cover at points of sampling, about 100 feet.

The bed was measured and sampled at two points in the mine by W. D. Schofield, on September 24, 1912, as described below.

Sections of coal bed in Hesperus mine.

Section	A		B	
Laboratory No.	14773		14774	
Roof, "draw slate."	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	2	2	1
Shale, carbonaceous	[a] 0	½	[a] 0	1
Coal	3	0	2	5
Floor, clay.				
Thickness of bed	5	2½	4	7
Thickness of coal sampled	5	2	4	6

[a] Not included in sample.

Section A (sample 14773) was measured at the face of room 3, off right entry 4; 4,000 feet from the mine mouth.

Section B (sample 14774) was measured in room 5, off left entry 5; 4,000 feet west of the mine mouth.

A composite sample was made by combining face samples 14773 and 14774. The results of an ultimate analysis of this sample are given under laboratory No. 14775.

Notes.—The coal is mined by the room-and-pillar system. At the time of sampling, the coal was undercut by hand and broken down with FF black blasting powder. None of the coal was shipped as run-of-mine. Bar screens 8 feet long with 3-inch openings and bar screens 8 feet long with 1-inch openings were used, about 60 per cent of the coal passing through the 1-inch screen. There were three loading tracks with a capacity of 20 loaded cars and 40 empty cars. The daily output was about 200 tons. About 60 per cent of the coal was from advance workings and 40 per cent from pillars. The probable lifetime of the mine is 20 years.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 67, 442.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull 316, 1907, p. 386.

PERINS. PERINS PEAK MINE.

Sample.—Bituminous (coking) coal; Durango field; analyses Nos. 14776, 14777, and 14778 (p. 27).

Mine.—Perins Peak, a drift mine in secs. 11 and 14, T. 35 N., R. 10 W., at Perins, 5 miles southeast of Durango, on a branch of the Rio Grande Southern Railroad.

Coal bed.—Peacock. Cretaceous age, Mesaverde formation. Thickness, 1 foot 6 inches to 6 feet 6 inches. The only impurities are irregular "sulphur" balls in pockets. The immediate roof is good and does not fall in the rooms. The floor is soft and hard shale. Particles of the roof and floor did not mix with the coal in loading. Cover at points of sampling, 70 to 120 feet.

The bed was measured and sampled at two points in the mine by W. D. Schofield on September 17, 1912.

Sample 14776 was cut from the face of room 3, off the main entry, 3,200 feet from the mine mouth. The sample represents 5 feet 1 inch of clear coal.

Sample 14777 was cut from the face of room 13, off the main back entry. The sample represents 5 feet 5 inches of clear coal.

A composite sample was made by combining face samples 14776 and 14777. The results of an ultimate analysis of this sample are given under laboratory No. 14778.

Notes.—The room-and-pillar system of mining is used. At the time of sampling the coal was undercut by hand and broken down with black powder and a permissible explosive. Dynamite was used for brushing the floor and roof. The undercuttings were loaded with the coal. Ninety-five per cent of the coal was shipped as run-of-mine. Ninety per cent of the coal going to the screens passed through. Two bar screens, one being 16 feet long with 8-inch openings and the other 16 feet long with ¾-inch openings, were used. Pickers were employed on the cars. The average daily output was 350 tons. About one-third of the coal was from advance workings and two-thirds from pillars. There was possibly 160 acres of unmined coal. There were three loading tracks, with a capacity of 20 loaded cars and 30 empty cars.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 67, 443.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 316, 1907, p. 386.

LAS ANIMAS COUNTY.

COKEDALE. COKEDALE MINE.

Sample.—Bituminous (coking) coal; Trinidad field; analyses Nos. 11768, 11769, 11770, and 11771 (p. 28).

Mine.—Cokedale, a drift mine in sec. 25, T. 33 S., R. 65 W., at Cokedale, on a branch of the Denver & Rio Grande Railroad.

Coal bed.—Known as the Cokedale. Cretaceous age, Vermejo formation. Average thickness, 6 feet. Roof, sandstone; floor, shale. Cover, 40 to 60 feet.

The bed was measured and sampled at three points in the mine by J. C. Roberts on February 24, 1911, as described below:

Sections of coal bed in the Cokedale mine.

Section	A.		B.		C.	
Laboratory No	11768		11769		11770	
Roof, sandstone:	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	4	2	7	2	5
Bone	[a]0	2	0	5	0	7
Coal	0	9	0	4	0	7
Bone	[a]0	7	[a]1	2	[a]0	3
Coal	2	3	0	7	2	3
Bone	..	..	[a]0	6	..	..
Coal	..	..	0	4	..	..
Bone	..	..	[a]0	1½	..	..
Coal	..	..	0	10	..	..
Floor, shale:						
Thickness of bed	6	1	6	10½	6	1
Thickness of coal sampled	5	4	4	9½	5	3

[a] Not included in sample.

Section A (sample 11768) was measured at the face of room 8, off west entry 4.

Section B (sample 11769) was measured at the face of room 21, off west entry 1.

Section C (sample 11770) was measured at the face of room 10, off west entry 6.

A composite sample was made by combining face samples 11768, 11769, and 11770. The results of an ultimate analysis of this sample are given under laboratory No. 11771.

Notes.—At the time of sampling the output was 1,200 tons a day.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 69, 447.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, p. 399.

DELAGUA. DELAGUA MINE.

Sample.—Bituminous (coking) coal; Trinidad field; analyses Nos. 11435, 11436, 11437, and 11443 (p. 28).

Mine.—Delagua, a shaft mine in sec. 15, T. 31 S., R. 65 W., at Delagua, 20 miles north of Trinidad, on the Colorado & Southeastern Railway.

Coal bed.—Known as the Delagua. Cretaceous or Tertiary age, Raton formation. Thickness, 5 feet to 6 feet. Roof, shale 12 to 20 inches thick. Cover at points of sampling, 450 to 500 feet.

The bed was measured and sampled at three points in the mine by J. C. Roberts on December 9, 1910, as described below:

Sections of coal bed in the Delagua Mine.

Section	A		B		C	
Laboratory No	11435		11436		11437	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	10	6	6	2	8
Bone	0	¼	..	..	[a]0	4
Coal	2	9¾	..	..	2	8
Floor, not stated.						
Thickness of bed	5	8	6	6	5	8
Thickness of coal sampled	5	8	6	6	5	4

[a] Not included in sample.

Section A (sample 11435) was measured at the face of room 1, east entry 24, off north entry 3.

Section B (sample 11436) was measured at the face of east entry 18, off north entry 4.

Section C (sample 11437) was measured at the face of room 5, west entry 3, off north entry 4.

A composite sample was made by combining face samples 11435, 11436, and 11437. The results of an ultimate analysis of this sample are given under laboratory No. 11443.

Notes.—At the time of sampling the output was 1,000 to 1,500 tons a day.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 69, 448.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, p. 410.

For results of tests of other coal from this mine see mention of specific tests as follows—Washing tests: U. S. Geol. Survey Bull. 368, p. 27; coking tests: U. S. Geol. Survey Bull. 368, p. 39.

FORBES. FORBES NO. 9 (OR COX) MINE.

Sample.—Bituminous coal; Trinidad field; analyses Nos. 11926, 11927, 11929, and 11930 (pp. 28 and 29).

Mine.—Forbes, a drift mine in secs. 15 and 16, T. 32 S., R. 64 W., at Forbes, on a spur of the Denver & Rio Grande Railroad.

Coal bed.—Walsen. Cretaceous age, Vermejo formation. Average thickness about 4 feet; dip 1° southwest. One fault showing a displacement of 3 feet has been found. The roof is a hard, smooth, dark-gray shale, overlain by sandstone. The floor is a thin, hard, smooth shale, underlain by 18 inches of coal; beneath the coal is shale. Cover at points of sampling, 300 to 400 feet.

The bed was measured and sampled at three points in the mine by J. C. Roberts on March 16, 1911.

Sample 11926 was cut from the face of the first crosscut, off west entry 1; 2,800 feet southwest of the slope opening. The sample represents 4 feet 4 inches of clean coal.

Sample 11927 was cut from the face of west entry 6; 3,800 feet southwest of the slope opening. The sample represents 3 feet 9 inches of clean coal.

Sample 11929 was taken in room 6, off west entry 4; 3,200 feet southwest of the slope opening. The sample represents 3 feet 11 inches of clean coal.

A composite sample was made by combining face samples 11926, 11927, and 11929. The results of an ultimate analysis of this sample are given under laboratory No. 11930.

Notes.—The room-and-pillar system of mining is employed. At the time the mine was visited the coal was undercut by hand and broken down with black powder; "40 per cent" dynamite was used for brushing the roof and taking up the bottom. Nearly 95 per cent of the coal was shipped as run-of-mine. The coal was friable, about 38 per cent passing through a 2-inch screen. Pickers were employed on the cars in loading. There were three loading tracks, with a capacity of 30 empty cars and 60 loaded cars. About 80 per cent of the coal was recovered, 40 per cent being from advance workings and 40 per cent from pillar workings. The daily output was 600 to 700 tons.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, p. 402.

HASTINGS. HASTINGS MINE.

Sample.—Bituminous (coking) coal; Raton Mesa field; analyses Nos. 12028, 12029, and 12031 (p. 29).

Mine.—Hastings, a slope mine in sec. 25, T. 32 S., R. 64 W., at Hastings, on the Colorado & Southeastern Railroad, with connections to the Colorado & Southern Railway and the Denver & Rio Grande Railroad.

Coal bed.—Berwind (?) and Hastings. Cretaceous age, Vermejo formation.

The Hastings bed averages about 8 feet in thickness and dips 5° west. The roof and floor are shale. Cover at points of sampling, about 800 feet.

The bed was measured and sampled at two points in the mine by J. C. Roberts on April 6, 1911, as described below:

Section of Hastings bed in Hastings mine.

Laboratory No.	12028	
Roof, shale.	*Ft.*	*in.*
Coal	6	0
Coal, bony	[a]1	0
Coal	3	5
Floor, shale.		
Thickness of bed	10	5
Thickness of coal sampled	9	5

[a] Not included in sample.

The sample was measured on the pillar between south entries 7 and 8, 8,300 feet from the mouth of the slope.

Sample 12031 was taken in south entry 7, at the fifth crosscut, 8,200 feet from the mouth of the slope.

The Berwind (?) bed is about 40 feet below the Hastings. The roof is sandstone; it is smooth and does not adhere to the coal in mining. The floor is a hard, smooth sandstone. The cover is 800 feet thick.

The bed was measured and sampled at one point in the mine by J. C. Roberts on April 6, 1911, as described below:

Section of Berwind (?) bed in Hastings mine.

Laboratory No.	12029	
Roof, sandstone.	*Ft.*	*in.*
Coal	1	0
Coal, bony	0	1
Coal	0	6
Coal, bony	0	1
Coal	0	6
Coal, bony	0	1
Coal	2	10
Floor, sandstone.		
Thickness of bed	5	1
Thickness of coal sampled	5	1

The above section (sample 12029) was measured at the face of the slope.

Notes.—The double entry, room-and-pillar system of mining is used. At the time of sampling, the coal was undercut with electric chain cutters and shot down with a permissible explosive.

None of the coal was sold as run-of-mine. The coal was screened over a 2½-inch bar screen, about 50 per cent passing through the screen. The screenings were washed with Stewart jigs and yielded a 2-inch product and slack. The capacity of the washery was 400 tons a day. There were coke ovens with a capacity of 300 tons and storage bins with a capacity of 500 tons. There were two loading tracks capable of handling 150 empty and 150 loaded cars.

The capacity of the mine was 1,000 tons a day. There was 640 acres of unmined coal tributary to the main entry.

For results of tests of other coal from this mine, see mention of specific tests, as follows: Washing tests, U. S. Geol. Survey Bull. 368, p. 27; coking tests, U. S. Geol. Survey Bull. 368, p. 45.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 69, 449.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, p. 404.

MORLEY. MORLEY MINE.

Sample.—Bituminous (coking) coal; Trinidad field; analyses Nos. 13361, 13362, 13363, and 13364 (p. 29).

Mine.—Morley, a drift and slope mine at Morley, on the main line of the Atchison, Topeka & Santa Fe Railway.

Coal bed.—Known as the Engle-Starkville. Cretaceous age, Vermejo formation. Average thickness, 4 to 7 feet; dip, 15° to 18° west. The roof is a shale about 18 inches thick; above the shale is sandstone. Floor, shale or sandstone. Particles from the roof and floor did not mix with the coal in mining. The cover at points of sampling ranges from 250 to 450 feet.

The bed was measured and sampled at three points in the mine by J. C. Roberts on February 17, 1912, as described below:

Sections of coal bed in Morley mine.

Section	A		B		C	
Laboratory No.	13361		13362		13363	
Roof, 18 inches of coal left for safety; shale and sandstone above.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	0	0	11	5	1
Shale, soft	0	2	0	1	..	..
Coal	3	1	5	1	..	..
Floor, shale and sandstone.						
Thickness of coal mined	5	3	6	1	5	1
Thickness of coal sampled	5	3	6	1	5	1

Section A (sample 13361) was measured at the face of room 2, off third plane, off right entry 4.

Section B (sample 13362) was measured at the face of fifth left entry.

Section C (sample 13363) was measured at the face of fourth right entry; 3,000 feet from the mine mouth.

A composite sample was made by combining and mixing face samples 13361, 13362, and 13363. The results of an ultimate analysis of this sample are given under laboratory No. 13364.

Notes.—The Morley mine is opened on the east side by a drift and on the west side by a slope. The room-and-pillar system of mining is used. At the time of sampling the coal was undercut by hand and broken down with a permissible explosive, which was also used for brushing the roof. Seventy-five per cent of the coal was shipped as run-of-mine, and 25 per cent was screened. Shaker screens, 16 feet long by 6 feet wide, with 3-inch and 1½-inch holes were used; 60 per cent passed through the screens. Pickers were employed in loading screened coal on the cars. The lumps were of medium size and of good appearance. The screenings were coked; they were not washed before coking. There was no storage bin at this mine. There were two tracks, with a capacity of 45 empty and 45 loaded cars. The probable lifetime of the mine is about 25 years.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 69, 450.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, p. 424.

PRIMERO. FIRST EAST, FIRST WEST, AND FIRST NORTH PRIMERO MINES.

Sample.—Bituminous coal; Trinidad field; analyses Nos. 12025, 12027, W31033 to W31043, 14058, 14059, 14060, W31131 to W31151 (pp. 29–32).

Mines.—First East, First West, and First North Primero, drift mines, in secs. 23, 24, 25, and 26, T. 33 S., R. 66 W., at Primero, on the Colorado & Wyoming Railway.

Coal bed.—Primero. Cretaceous or Tertiary age, Raton formation. Average thickness, 7 feet; dip, 4° northwest. Faults, rolls, and horsebacks are frequent. The roof is a hard black shale; above the shale is a sandstone cap rock. Particles of the shale mix with the coal in mining. The floor is a soft smooth shale, particles of which occasionally mix with the coal in mining. The cover at points of sampling to ranges from 50 to 500 feet at the First West mine; from 500 to 550 feet at the First North mine; and from 600 to 850 feet at the First East mine.

The bed was measured and sampled at 16 points in the mine as follows: 2 points on March 25, 1911, by J. C. Roberts; 5 points in the First West Primero mine on May 16, 1912; 3 points in the First North Primero mine on May 19, 1912; and 6 points in the First East Primero mine on May 19, 1912, by J. C. Roberts and W. D. Schofield; also 9 sets of duplicate samples were taken from cars that were being loaded on May 16 to 19, inclusive, 1912, as described below:

Sample 12025 was cut from the face of room 1, off the second blind back entry. The bed at this point was 5 feet 10 inches thick, all of which was included in the sample.

Sample 12027 was cut from the face of the first west slope, 1,500 feet from the drift mouth.

Sections of coal bed in First East Primero mine.

Section	A		B		C	
Laboratory No.	W31144		W31145		W31146	
Roof, "draw slate" and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	11	0	9	1	0
Shale	0	½	0	½	0	½
Coal	1	0	3	10	1	2½
Shale	0	½	..	..	..	..
Natural coke	..	..	0	1½	0	1
Coal	3	1	1	8	3	7
Floor, shale.						
Thickness of bed	5	1	6	5	5	11
Thickness of coal sampled	5	1	6	5	5	11

Section A (sample W31144) was measured on a pillar of room 5, off entry B–9, off first blind entry.

Section B (sample W31145) was measured on a chain pillar of east entry B–9.

Section C (sample W31146) was measured at the face of room 4, off east entry B–10.

Sample W31143 was measured on a pillar of right entry 6, between rooms 1 and 2, off third blind entry, off entry B–8. The bed measured 6 feet 1 inch at this point, all of which was included in the sample.

Sample W31147 was measured at the face of room 3, off eleventh east entry. The bed measured 7 feet 4 inches at this point, all of which was included in the sample.

Sample W31149 was measured on a chain pillar of third blind entry, off east entry B-10. The bed measured 7 feet at this point, all of which was included in the sample.

A composite sample was made by mixing face samples W31143, W31144, W31145, W31146, W31147, and W31149. The results of an ultimate analysis of this sample are given under laboratory No. 14060.

Sections of coal bed in First North Primero mine.

Section	A		B		C	
Laboratory No.	W31148		W31150		W31151	
Roof, "draw slate" and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	4	..	..	..	..
Bone	0	1	0	6	..	..
Coal	1	11	2	0	3	5
Bone or "sulphur"	0	¼	[a]0	1	[a]0	1½
Coal	0	2	..	..	..	..
Shale	0	¼	..	..	..	..
Coal	2	8	2	2	..	..
Bone	0	½	0	½	..	..
Coal	2	8	2	2½	3	7½
Floor, underclay:						
Thickness of bed	7	11	7	0	7	2
Thickness of coal sampled	7	11	6	11	7	½

[a] Not included in sample.

Section A (sample W31148) was measured at the face of room 5, off third back entry, off entry A-12.

Section B (sample W31150) was measured on a pillar of room 7, off entry A-12.

Section C (sample W31151) was measured at the face of entry B-9.

A composite sample was made by combining face samples 31148, 31150, and 31151. The results of an ultimate analysis of this sample are given under laboratory No. 14059.

Sections of coal bed in First West Primero mine.

Section	A		B		C		D		E	
Laboratory No.	W31039		W31040		W31041		W31042		W31043	
Roof, sandstone and "draw slate."	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	8	2	5	..	..	..	..	..	..
Bone	0	½	0	1	..	..	..	..	..	..
Coal	0	8	0	1	..	..	..	..	..	..
Bone, "sulphur" band	[a]0	1½	0	½	..	..	..	..	..	..
Coal	1	7	0	6	..	..	..	..	..	..
Shale	[a]0	2	0	½	..	..	..	..	..	..
Coal	..	..	0	8	0	11	2	2½	2	10
Natural coke	..	..	..	..	..	..	[a]0	4½	..	..
Shale	..	..	0	1	[a]0	2	..	..	0	1
Coal	1	6	1	5	3	8	2	5	2	1
Floor, clay and sandstone.										
Thickness of bed	4	9	5	4	4	9	5	0	5	0
Thickness of coal sampled	4	5½	5	4	4	7	4	7½	5	0

[a] Not included in sample.

Section A (sample W31039) was measured at the face of north entry 2.

Section B (sample W31040) was measured at the face of room 10, off first north entry.

Section C (sample W31041) was measured at the face of room 9, off second south entry.

Section D (sample W31042) was measured at the face of room 5, off first right entry, off the first west air course.

Section E (sample W31043) was measured at the face of room 17, off second right entry, off the first west air course.

A composite sample was made by combining and mixing face samples 31039 to 31043, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14058.

Nine samples were taken in duplicate at the mine from the cars while being loaded. Fifty pounds of coal was taken from each of four cars. The four lots were combined, quartered, and sampled in duplicate.

Duplicate samples W31033 and W31034, W31035 and W31036, W31131 and W31132, W31133 and W31134, W31135 and W31136, W31137 and W31138, and W31141 and W31142, were taken from four cars; duplicate samples W31139 and W31140 were taken from one car only; 29 cars being sampled in all.

Notes.—The Primero mines are worked on the room-and-pillar system. At the time of sampling the coal was undercut by hand. Permissible explosives were used for breaking down the coal and for brushing the floor and roof. When the mine was sampled in 1912, all the coal was shipped as run-of-mine, but when screened, 65 per cent of the coal passed through. The screens were bars 18 feet long spaced 1½ inches apart. The screenings were washed and coked at Segundo, the capacity of the coking plant being 1,300 tons a day and the capacity of the washery 1,600 tons a day. Pickers were employed on the cars. The lumps on the cars were of medium size and of good appearance. There were four loading tracks with a capacity of 80 empty and 80 loaded cars. The daily average output was 1,200 tons of coal with a capacity of 1,600 tons. Seventy-five per cent of the output was from pillars and 25 per cent from advance workings. The probable lifetime of the mine was 25 years.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 69–70, 450–451.

For results of tests of coal from this mine see mention of specific tests as follows—Washing tests: Bureau of Mines Bull. 5, p. 32; coking tests: Bureau of Mines Bull. 5, p. 44.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, p. 426.

STARKVILLE. STARKVILLE MINE.

Sample.—Bituminous (coking) coal; Trinidad field; analyses Nos. 11134, 11135, 12196, 12197, and 12198 (p. 32).

Mine.—Starkville, a drift mine at Starkville, on the main line of the Atchison, Topeka & Santa Fe Railway.

Coal bed.—Engle-Starkville. Cretaceous age, Vermejo formation. Average thickness, 6 feet; dip 1° southwest. Roof, smooth shale overlain by sandstone; floor, hard rough shale; cover, at points of sampling, 150 to 550 feet.

The bed was measured and sampled at two points in the mine by H. M. Wolflin (Nos. 11134 and 11135) on October 20, 1910, and at two points by J. C. Roberts (Nos. 12196 and 12197) on May 9, 1911, as described below:

Sections of coal bed in Starkville mine.

Section	A		B		C		D	
Laboratory No.	11134		11135		12196		12197	
Roof, shale and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Bone	0	3	..	..	..	..	..	..
Coal, clean and bright	0	7	0	6	..	..	0	10
Shale	0	½	..	..	..	..	..	..
Coal, clean	1	½	..	..	..	..	..	..
Coal, hard and bright	0	½	0	½	..	..	..	..
Coal, clean	1	2¼	1	3	..	..	..	..
Bone	[a]0	2¼	..	..	..	..	[a]0	8
Coal, clean	0	7	..	..	..	..	0	7
Bone	[a]0	1½	[a]0	1½	..	..	0	2
Coal, bright	1	5	..	..	..	..	1	9
Bone	0	3½	..	..	..	..	[a]0	3
Shale	[a]0	1¼	..	..	..	..	..	..
Coal, clean	1	4½	1	2	5	9	0	10
Floor, shale.								
Thickness of bed	7	2¾	3	1	5	9	5	1
Thickness of coal sampled	6	9¾	2	11½	5	9	4	2

[a] Not included in sample.

Section A (sample 11134) was measured on the rib of west entry 8, 30 feet outby slant from new entry.

Section B (sample 11135) was measured in south entry 3 near the second crosscut from the new entry.

Section C (sample 12196) was measured in entry I-5 on the pillar between rooms 26 and 27, 15,000 feet southwest of the opening.

Section D (sample 12197) was measured at the face of south entry 1, 1,200 feet southeast of the mouth of the main drift.

A composite sample was made from face samples 12196 and 12197. The results of an ultimate analysis of this sample are given under Laboratory No. 12198.

Notes.—The room-and-pillar system of mining is used. At the time of sampling the coal was undercut by hand; no explosives were used for breaking down the coal. A large part of the output was sold as run-of-mine. About 70 per cent of the coal going to the screens passed through a 2½-inch bar screen; pickers were employed on the cars. The screenings were washed; the capacity of the washery was 700 tons a day; coke ovens, 600 tons; storage bins, 1,400 tons. There were three loading tracks with a capacity of 70 empty and 70 loaded cars.

The average output was 900 tons a day; unmined area, 640 acres.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, p. 396.

TOLLERBURG. TOLLER MINE.

Sample.—Bituminous coal; Trinidad field; analyses Nos. 13253 and 13254 (p. 32).

Mine.—Toller, a shaft mine near Tollerburg, on a spur of the Colorado & Southern Railway.

Coal bed.—Berwind. Cretaceous age, Vermejo formation. Average thickness, 6 feet; dip, 4 to 5 per cent, southwest; cleat, east, and west. Faults, rolls and horsebacks occur frequently. The roof is a hard shale which in places is full of slips. Above this shale is 15 inches of good quality of roof coal; above the roof coal is a cap rock, the cap rock being 5 to 6 feet above the coal mined. The floor is a soft smooth sandstone. Neither the roof nor floor materials mixed with the coal in mining. Cover, at points of sampling, 400 to 450 feet.

The bed was measured and sampled at two points in the mine by J. C. Roberts on January 8, 1912, as described below:

Sections of coal bed in Toller mine.

Section	A		B	
Laboratory No.	13253		13254	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Coal, clean	7	0	1	3
Bone	..	..	0	4
Coal, clean	..	..	5	5
Floor, shale.				
Thickness of bed	7	0	7	0
Thickness of coal sampled	7	0	7	0

Section A (sample 13253) was measured at the face of the slope, 2,500 feet from the shaft bottom.

Section B (sample 13254) was measured at the face of north entry 5, 3,000 feet from the shaft bottom.

Notes.—The shaft is 350 feet deep. The coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by hand. Permissible explosives were used for breaking down the coal and for brushing the roof and floor. Ninety per cent of the coal was shipped as run-of-mine. The tipple had shaker screens 15 feet long by 5 feet wide with 3-inch holes. Pickers were employed on the cars in loading.

The lumps were of medium size and had a good appearance. There were three tracks with a capacity of 30 empty cars and 25 loaded cars. The average daily production was 700 tons, with a capacity of 900 tons. Three-fourths of the coal was from advance workings and one-fourth from pillars. About 400 acres remain to be mined.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 381, 1910, p. 403.

MESA COUNTY.

GRAND JUNCTION. PROSPECT.

Sample.—Bituminous coal; Gunnison Valley field; analyses Nos. 11108 and 11109 (p. 33).

Mine.—Prospect, near the junction of the Grand and Gunnison Rivers in sec. 26, T. 1 S., R. 1 W., 1 mile south of Grand Junction, which is the nearest station on the Denver & Rio Grande Railroad.

Coal bed.—Unnamed. Cretaceous age, Dakota (?) formation. Thickness, about 3 feet. Both the roof and floor are carbonaceous shale. Cover, about 30 feet.

The bed was measured and samples were taken at one point in the mine by E. G. Woodruff on October 20, 1910, as described below:

Sections of coal bed in prospect near Grand Junction.

Section	A		B	
Laboratory No.	11108		11109	
Roof, carbonaceous shale.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	11	[a]1	11
Shale	[a]0	1	[a]0	1
Coal	[a]0	10	0	10
Floor, carbonaceous shale.				
Thickness of bed	2	10	2	10
Thickness of coal sampled	1	11	0	10

[a] Not included in sample.

The above samples were taken in the entry, about 125 feet from the mouth of the entry. At the time the samples were taken the prospect was not being worked.

Note.—The coal is hard and black and looks like good coal, but contains a very high percentage of ash.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 471, 1912, p. 568.

GRAND JUNCTION. PROSPECT.

Sample.—Subbituminous coal; Gunnison Valley region; analyses Nos. 11104 and 11105 (p. 33).

Mine.—A prospect opening between Grand Junction and White Water, in sec. 5; T. 2 S., R. 1 E., 5 miles southeast of Grand Junction, which is the nearest station on the Denver & Rio Grande Railroad.

Coal bed.—Unnamed bed. Cretaceous age, Dakota(?) formation. Thickness, 2 feet 6 inches. Roof, sandstone; floor, carbonaceous shale.

The bed was measured and samples were taken from one point in the mine by E. G. Woodruff on October 10, 1910, as described below:

Sections of coal bed in prospect, 5 miles south of Grand Junction.

Section	A		B	
Laboratory No.	11104		11105	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	3	[a]1	3
Shale, gray	[a]0	3	[a]0	3
Coal	[a]1	0	1	0
Floor, carbonaceous shale.				
Thickness of bed	2	6	2	6
Thickness of coal sampled	1	3	1	0

[a] Not included in sample.

Both of these samples were taken in the entry, about 100 feet from the mouth of the entry. At the time of sampling this prospect was not being worked.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 471, 1910, p. 569.

MOFFAT COUNTY.

AXIAL. BATTLE ERA MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 14528 (p. 33).

Mine.—Battle Era; NW. ¼ sec. 14, T. 4 N., R. 94 W.; in Morgan Gulch, 6 miles west of Axial, 59 miles west and 14 miles south of the terminus of the Denver & Salt Lake Railroad.

Coal bed.—No name. Cretaceous age, Mesaverde formation. Dip, 8½° south.

The bed was measured and sampled at one point in the mine on July 29, 1912, by E. T. Hancock, as shown below:

Section of coal bed in Battle Era mine.

Laboratory No	14528	
Roof, sandy shale.	*Ft.*	*in.*
Coal	2	½
Shale, carbonaceous	[a]0	6
Coal	8	11½
Floor, carbonaceous shale.		
Thickness of bed	11	6
Thickness of coal sampled	11	0

[a] Not included in sample.

The sample was taken in the entry 130 feet northeast of the mine mouth.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 415, 1910, p. 167.

AXIAL. ED COLLOM MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 14529 (p. 33).

Mine.—Ed Collom; SW. ¼ sec. 13, T. 4 N., R. 94 W.; in Collom gulch, 5½ miles northwest of Axial, 14 miles south and 56 miles west of the terminus of the Denver & Salt Lake Railroad.

Coal bed.—No name. Cretaceous age, Mesaverde formation. Dip, about 40° southwest.

The bed was measured and sampled at one point in the mine by E. T. Hancock on July 29, 1912.

The sample was taken in the mine 75 feet northwest of the mine mouth.

The bed was 13 feet 6 inches thick at the point sampled, all of which was included in the sample. The roof was shaly sandstone, the floor was carbonaceous shale. The coal was mined for local use only.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 415, 1910, p. 173.

AXIAL. JAMES MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 12704 (p. 33).

Mine.—James; in sec. 14, T. 3 N., R. 93 W.; on Spring Creek, 4 miles south of Axial, on the Meeker-Axial stage road.

Coal bed.—No name. Cretaceous age; Mesaverde formation. Dip, 13° S., 66° E.

The bed was measured and sampled on October 9, 1911, by E. T. Hancock.

The sample was taken in the main entry 100 feet north of the mine mouth, in 6 feet 4 inches of coal. There was coal above the part sampled. The floor was brown shale.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 79, 476.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 415, 1910, p. 168.

AXIAL. JOE COLLOM MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 14543 (p. 33).

Mine.—Joe Collom; SW. ¼ NE. ¼ sec. 2, T. 3 N., R. 93 W.; on Spring Creek, 1½ miles south of Axial; 65 miles north of the Denver & Rio Grande Railroad.

Coal bed.—Collom. Cretaceous age; Mesaverde formation. Dip, 3½° S., 29° W.

The bed was measured and sampled at one point in the mine on July 30, 1912, by E. T. Hancock, as described below:

Section of coal bed measured near opening of Joe Collom mine.

Laboratory No.	14543	
Roof, brown shale.	*Ft.*	*in.*
Coal	[a]6	9½
Shale, brown, sandy	[a]0	½
Coal	[a]4	4½
Coal	8	11
Floor, brown shale.		
Thickness of bed	20	½
Thickness of coal sampled	8	11

[a] Not included in sample.

Only 8 feet 11 inches of the lower bench of the coal bed was exposed in the mine. Only that portion was sampled. The sample was taken 400 feet northwest of the mine mouth.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 79, 476.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 415, 1910, p. 167.

AXIAL. SHAFER MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 14909 (p. 33).

Mine.—Shafer; sec. 31, T. 4 N., R. 92 W.; on Milk Creek, 2 miles east of Axial, 50 miles west and 16 miles south of the western terminus of the Denver & Salt Lake Railroad.

Coal bed.—No name. Cretaceous age; Mesaverde formation. Dip, 34° S., 50° E.

The bed was measured and sampled at one point in the mine on October 11, 1912, by E. T. Hancock, as shown below:

Section of coal bed in Shafer mine.

Laboratory No.	14909	
Roof, brown shale.	*Ft.*	*in.*
Coal	9	6
Coal, impure	0	6
Coal	3	9
Shale, black, carbonaceous (3 inches).		
Floor, brown shale.		
Thickness of bed	13	9
Thickness of coal sampled	13	9

The sample was taken in the mine 189 feet northeast of the mine mouth.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 415, p. 170.

CRAIG. WALKER MINE.

Sample.—Subbituminous coal; Yampa field; analysis No. 12400 (p. 33).

Mine.—Walker; sec. 17, T. 6 N., R. 90 W.; about 3 miles southeast of Craig, not on any railroad.

Coal bed.—Unnamed. Cretaceous age, Mesaverde formation. The coal is 8 feet thick and very free from foreign matter. The bed was sampled at one point in the mine by A. J. Hazelwood on June 20, 1911.

The sample was cut from the face of the main entry, 300 feet west of the opening.

Notes.—The Walker mine is an old country bank and had been in operation 20 to 30 years. At the time of sampling the coal was run out of the mine in cars and dumped into wagons. The coal was used entirely for domestic purposes. The coal was screened in the mine by means of forks and the fine coal was left in the gob. It was estimated that there was about 1,000 tons of slack in the worked-over parts of the mine. About one-fifth of the coal is slack. The output averaged approximately 20 tons per month. There was 1,000 acres of coal tributary to this opening.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 297, 1906, p. 62.

MONTEZUMA COUNTY.

CORTEZ. CORTEZ MINE.

Sample.—Bituminous coal; Ute Mountain-Mancos field; analysis No. 12785 (p. 33).

Mine.—Cortez; NW. ¼ sec. 23, T. 36 N., R. 16 W.; one-half mile west of Cortez.

Coal bed.—The lowest bed in the Dakota formation; Cretaceous age. Dip 6°, northwest. In many places this bed is about 2 feet thick and exceeds that thickness only locally. The coal is extremely irregular in character, and sections taken only a few feet apart may not be alike. In many places the roof is sandstone, at other places it is shale, the shale is only a few inches, scarcely ever over 1 foot thick. Sandstone lenses interbedded with the coal are not uncommon.

The bed was measured and sampled at one point in the mine on October 20, 1911. by F. M. Handy and M. A. Pishel, as shown below:

Section of coal bed in Cortez mine.

Laboratory No	12785	
Roof, sandstone.	*Ft.*	*in.*
Coal, bony	0	6
Clay	[a]0	6
Shale	[a]0	11½
Bone	[a]0	4
Coal, bony	[a]0	7½
Coal	0	2
Bone	[a]0	2½
Coal	0	11
Bone	[a]0	6
Floor, shale.		
Thickness of bed	4	8½
Thickness of coal sampled	1	7

[a] Not included in sample.

Notes.—The mine is small and is worked intermittently for local trade. No one was at the mine at the time the sample was taken. There were no screens at the mine; the entire output being loaded as run-of-mine coal. The sample is probably slightly weathered.

CORTEZ. HAMILTON PROSPECT.

Sample.—Bituminous coal; Ute Mountain-Mancos field; analysis No. 12686 (p. 33).

Mine.—Hamilton prospect; in SE. ¼ sec. 3, T. 35 N., R. 18 W., 16 miles southwest of Cortez.

Coal bed.—Unnamed bed. Cretaceous age, Dakota formation. The bed lies nearly horizontal, in many places is about 2 feet thick and exceeds that thickness only locally. The coal is extremely irregular in character and sections taken only a few feet apart may not be alike. The roof is sandstone; at places the sandstone is replaced by shale, which is only a few inches, scarcely ever over 1 foot thick. It is not uncommon to find sandstone lenses interbedded with the coal.

The bed was measured and sampled at one point in the mine on September 17, 1911, by M. A. Pishel, as shown below:

Section of coal bed in Hamilton prospect, about 16 miles southwest of Cortez.

Laboratory No.	12686	
	Ft.	*in.*
Coal	0	6
Shale	[a]0	4
Bone	[a]0	7
Coal, bony	0	7
Bone	[a]0	1
Coal	0	6
Bone	[a]0	1
Thickness of bed	2	8
Thickness of coal sampled	1	7

[a] Not included in sample.

The sample was taken near the surface in an open cut.

Notes.—This prospect was within a mile of an igneous sheet and it is probable that this sheet extended over the bed only a short distance above the coal. The coal at the prospect had a semimetallic luster resembling semianthracite, whereas the coal of this bed in the neighborhood of Cortez has the appearance of typical bituminous coal. It is probable that this phenomenon is due to the proximity of this prospect to the igneous sheet referred to above.

CORTEZ. JACKSON MINE.

Sample.—Bituminous coal; Ute Mountain-Mancos field; analyses Nos. 16472, 16473, 16474, and 16475 (p. 34).

Mine.—Jackson, a drift mine in the SW. ¼ NE. ¼ sec. 33, T. 35 N., R. 16 W., 10 miles southeast of Cortez, in the Mesa Verde National Park. There are no railroad connections.

Coal bed.—Lower bed of the Mesaverde formation, possibly the Spencer bed; Cretaceous age. Average thickness, 2 feet 8 inches; dip, 2° southwest; cleat, northeast and southwest. Faults, rolls, and horsebacks occur frequently. The roof is a hard blue shale; above the shale is a sandstone cap rock. Floor is hard smooth shale. Cover at points of sampling, 30 feet.

The bed was measured and sampled at three points in the mine by J. C. Roberts on February 13, 1913, as described below:

Sections of coal bed in Jackson mine.

Section	A		B		C	
Laboratory No.	16472		16473		16474	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	10	2	7	2	7
Shale and sandstone	..	..	..	..	[a]1	8
Coal	..	..	..	..	[a]1	0
Floor, shale.						
Thickness of bed	2	10	2	7	5	3
Thickness of coal sampled	2	10	2	7	2	7

[a] Not included in sample.

Section A (sample 16472) was measured at the face of room 4, off the north entry, 200 feet from the mine mouth.

Section B (sample 16473) was measured at the face of the main entry, 170 feet north of the mine mouth.

Section C (sample 16474) was measured at the face of the south entry, 95 feet from the mine mouth.

A composite sample was made by mixing samples 16472, 16473, and 16474. The results of an ultimate analysis of this sample are given under laboratory No. 16475.

Notes.—The mine is worked on the room-and-pillar system. At the time of sampling the development consisted of a pair of entries driven slightly north of east with four rooms turned. The coal was undercut by hand and shot down with black blasting powder. The coal was screened over a "grizzley" with wood bars 2 inches by 4 inches by 10 feet long spaced three-fourths inch apart. The coal was used entirely for domestic purposes. The daily output of the mine was 1 ton. Sixty acres remained to be mined from this opening.

CORTEZ. MOWRY MINE.

Sample.—Bituminous (coking) coal; Durango field; analyses Nos. 12586 and 16479 (p. 34).

Mine.—Mowry, a drift mine, 1½ miles southeast of Cortez, no railroad connection.

Coal bed.—Unnamed bed. Cretaceous age, Dakota formation. The bed varies greatly in thickness and completely pinches out a quarter of a mile from the mine. The bed averages about 2 feet thick and exceeds that thickness only locally. Dip, 3° northwest. There is a layer of bony coal in the lower part which varies from 2 to 6 inches. No faults, rolls, or horsebacks have been encountered. The roof is a smooth sandstone; at places the sandstone is replaced by shale. The floor is shale and a smooth sandy clay. Particles from the roof and floor did not mix with the coal in mining. Cover at point of sampling, about 70 feet.

The bed was measured and sampled at one point in the mine by M. A. Pishel on August 28, 1911, and at one point by J. C. Roberts on February 14, 1913, as described below:

Sections of coal bed in Mowry mine.

Section	A		B	
Laboratory No.	12586		16479	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	2	1	9
Bone	[a]0	6	[a]0	5
Floor, clay and shale.				
Thickness of bed	2	8	2	2
Thickness of coal sampled	2	2	1	9

[a] Not included in sample.

Section A (sample 12586) was taken near the face, 75 feet from the mine mouth. Fine streaks of bony coal were interlaminated with the coal at this point.

Section B (sample 16479) was taken in the main entry 100 feet from the mine mouth.

Notes.—At the time of sampling the coal was blasted off the solid with black blasting powder. The coal was marketed for domestic purposes and was forked and dumped directly into the wagons for delivery. The yearly output was about 150 tons.

CORTEZ. TODD MINE.

Sample.—Bituminous coal; Durango field; analyses Nos. 16476, 16477, and 16478 (p. 34).

Mine.—Todd, a drift mine, in sec. 25, T. 35 N., R. 16 W., in the Mesa Verde National Park, about 7 miles from Cortez and 19 miles from the Rio Grande Southern Railroad.

Coal bed.—Lowest bed of the Mesaverde formation, Cretaceous age. Thickness, 2 feet 6 inches to 3 feet 6 inches. The bed has a thin persistent parting of shale one-fourth inch to three-fourths inch thick near the middle. Roof, shale of poor quality; above the shale is a sandstone cap rock. Floor, hard smooth underclay. Particles of the roof and floor did not mix with the coal in loading. Cover at points of sampling, 80 to 100 feet.

The bed was measured and sampled at two points in the mine by J. C. Roberts on February 12, 1913, as described below:

Sections of coal bed in the Todd mine.

Section	A		B	
Laboratory No	16476		16477	
Roof, "draw slate".	*Ft.*	*In.*	*Ft.*	*In.*
Coal	1	½	1	3½
Shale	0	½	0	½
Coal	1	9	1	8
Floor, clay.				
Thickness of bed	2	10	3	0
Thickness of coal sampled	2	10	3	0

Section A (sample 16476) was measured at the face of room 9.

Section B (sample 16477) was measured at the face of room 10.

A composite sample was made by combining face samples 16476 and 16477. The results of an ultimate analysis of this sample are given under laboratory No. 16478.

Notes.—The mine is worked by the room-and-pillar system. At the time of sampling the coal was undercut and mined by hand, no explosives being used. The undercuttings were thrown into the gob. The coal was screened over bars 12 feet long with one-half-inch openings and picked as it was being loaded. The daily output was 4 tons, which was hauled from the mine in wagons. There were about 80 acres to be mined from this opening.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 75, 467.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 316, p. 389.

DOLORES. PROSPECT.

Sample.—Bituminous coal; Ute Mountain-Mancos field; analysis No. 12551 (p. 35).

Mine.—Unnamed prospect in the NE. ¼ sec. 29, T. 36 N., R. 15 W., 11 miles south of Dolores.

Coal bed.—Unnamed bed. Cretaceous age, Dakota formation. Dip, 2° northeast. In many places this bed is about 2 feet thick and exceeds that thickness only locally. The coal is extremely irregular in character, and sections taken only a few feet apart may not be alike. The roof is sandstone, which in places is replaced by shale; the shale is only a few inches, scarcely ever over 1 foot thick. Sandstone lenses interbedded in the coal are not uncommon.

The bed was measured and sampled by M. A. Pishel on August 18, 1911, as shown below:

Section of coal bed in prospect entry, 11 miles south of Dolores.

Laboratory No	12551	
Roof, sandy shale.	*Ft.*	*In.*
Coal	0	3½
Shale	[a]0	4½
Coal	0	4
Bone	[a]0	2
Coal, slightly bony	0	10
Bone	[a]0	4
Coal, bony	[a]0	8
Bone	[a]0	1
Coal	0	9
Bone	[a]0	2
Coal	0	4
Bone (1½ feet).		
Floor, not stated.		
Thickness of bed	4	4
Thickness of coal sampled	2	6½

[a] Not included in sample.

Note.—The prospect apparently had not been worked for some time before the sample was taken, so that the sample is probably somewhat weathered. It was taken near the face of the entry, 20 feet from the mouth.

MANCOS. HALLER MINE.

Sample.—Bituminous coal; Durango field; analysis No. 12504 (p. 35).

Mine.—Haller; a drift mine in NE. ¼ sec. 30, T. 37 N., R. 13 W., 6 miles north of Mancos.

Coal bed.—Unnamed bed. Cretaceous age, Dakota formation. Dip, 2°. In many places this bed is about 2 feet thick and exceeds that thickness only locally. The coal is extremely irregular in character, being interlaminated with bone, shale, and sandstone lenses, and sections taken only a few feet apart may not be alike. In many places the roof is sandstone, at others it is shale; the latter is only a few inches, scarcely ever over 1 foot thick.

The bed was measured and sampled at one point in the mine on July 31, 1911, by M. A. Pishel, as shown below:

Section of coal bed in Haller mine.

Laboratory No.	12504	
Roof, shale.	*Ft.*	*in.*
Coal	0	7
Bone	[a] 0	6
Coal	0	3
Bone	[a] 1	2
Coal	0	5
Bone	[a] 0	5
Coal	0	6
Floor, shale.		
Thickness of bed	3	10
Thickness of coal sampled	1	9

[a] Not included in sample.

The sample was taken in the entry, about 50 feet from the mine mouth.

Note.—The mine had not been worked for at least six months previous to the time this sample was taken, so the sample is probably somewhat weathered.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 75,467.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 316, p. 387.

MANCOS, PROSPECT.

Sample.—Bituminous coal; Ute Mountain-Mancos field; analysis No. 12444 (p. 35).

Location.—An unnamed prospect in NE. ¼ sec. 21, T. 36 N., R. 12 W., 8 miles northeast of Mancos.

Coal bed.—Unnamed bed. Cretaceous age, Dakota formation. Dip 6½° south. In many places this bed is about 2 feet thick, and exceeds that thickness only locally. The coal is extremely irregular in character, being irregularly interlaminated with bone, shale, and sandstone lenses, and sections taken only a few feet apart may not be alike. In many places sandstone forms the roof, at others it is shale; the latter is a few inches to 1 foot thick.

The bed was measured and sampled July 16, 1911, by M. A. Pishel, as shown below:

Section of coal bed in prospect entry 8 miles northeast of Mancos.

Laboratory No.	12444	
Roof, sandstone, good.	*Ft.*	*in.*
Shale	[a] 0	2
Coal	0	6
Coal, bony	0	3
Bone	[a] 0	2
Coal	0	2
Shale	[a] 0	2¼
Coal, slightly bony	0	3
Floor, shale.		
Thickness of bed	1	8¼
Thickness of coal sampled	1	2

[a] Not included in sample.

Note.—This sample was taken 20 feet from mouth of entry. The coal was somewhat weathered, as apparently coal had not been taken out of the prospect for at least one year. This bed is not known to be thicker at any place within 3 or 4 miles of this prospect.

OURAY COUNTY.

RIDGWAY. LOU CREEK MINE.

Sample.—Bituminous coal; Tongue Mesa field; analysis No. 12651 (p. 35).

Mine.—Lou Creek, a drift mine in NE. ¼ NE. ¼ sec. 22, T. 46 N., R. 7 W., 10 miles northeast of Ridgway.

Coal bed.—Lou Creek. Cretaceous age, Mesaverde formation. The character of the bed outside the mine is not known, and neither top nor bottom was exposed. The entry was run in on the coal, which appeared to be at least 20 feet thick, but this thickness may be due in part to crushing.

The bed was measured and sampled by W. T. Lee on September 12, 1911.

The sample was collected from the lower 6 feet of the bed as exposed in the mine, 600 feet from the mouth of the mine. Very little coal was mined; it was used for local purposes.

RIO BLANCO COUNTY.

MEEKER. BLACK DIAMOND MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 12776 (p. 35).

Mine.—Black Diamond; in sec. 15, T. 1 N., R. 94 W.; 1½ miles northwest of Meeker, 47 miles north of the Denver & Rio Grande Railroad.

Coal bed.—Lord. Cretaceous age, Mesaverde formation. Thickness at least 14 feet.

The bed was measured and sampled at one point in the mine on October 18, 1911, by E. T. Hancock, as shown below:

Section of coal bed in Black Diamond mine.

Laboratory No.	12776	
Sandstone.	*Ft.*	*in.*
Coal	3	10
Coal, streaked with shale	0	4
Coal	2	3
Thickness of bench mined	6	5
Thickness of coal sampled	6	5

There was coal below this section. The sample was taken in the mine 174 feet northeast of the entrance.

For analyses and descriptions of other samples of coal from this mine, see Bull. 22, Bureau of Mines, 1913, pp. 77, 471–472.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 415, 1910, p. 153.

MEEKER. CORNRIKE MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 12695 (p. 35).

Mine.—Cornrike mine; SW. ¼ sec. 10, T. 2 N., R. 93 W.; 10 miles northeast of Meeker; 54 miles north of Rifle, which is on the Denver & Rio Grande Railroad.

Coal bed.—No name. Cretaceous age, Mesaverde formation. Dip 20° S., 27° E. Total thickness measured outside of mine, 21 feet 7 inches.

The bed was measured and sampled at one point in the mine by E. T. Hancock, on October 5, 1911.

The sample was taken in the mine, 30 feet northeast of the mine mouth, in a 7-foot 6-inch bench of clean coal. There was coal above and below the bench.

Notes.—For a description of the coal beds in this township, see U. S. Geol. Survey Bull. 415, 1910, pp. 160–162.

MEEKER. FAIRFIELD MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 12518 (p. 35).

Mine.—Fairfield; SW. ¼ sec. 28, T. 1 N., R. 94 W.; 2½ miles west of Meeker and 45 miles north of the Denver & Rio Grande Railroad.

Coal bed.—No name. Cretaceous age, Mesaverde formation. Dip, 42° N., 67° W.

The bed was measured and sampled at one point in the mine by E. T. Hancock on August 12, 1911, as shown below:

Section of coal bed in Fairfield mine.

Laboratory No.	12518
Roof, coal (not mined).	*Ft. in.*
Coal	6 0
Bone	[a]0 4
Coal	0 4
Floor, carbonaceous shale.	
Thickness of coal mined	6 8
Thickness of coal sampled	6 4

[a] Excluded from sample.

The sample was taken in the mine, 300 feet northeast of the mine mouth.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 316, 1910, p. 143.

MEEKER. MEEKER MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 12426 (p. 35).

Mine.—Meeker NW. ¼ sec. 22, T. 1 N., R. 94 W., three-fourths mile northwest of Meeker and 46 miles north of the Denver & Rio Grande Railroad.

Coal bed.—Old Pollard. Cretaceous age, Mesaverde formation. Dip, 20° NW.

The bed was measured and sampled at one point in the mine by E. T. Hancock on July 15, 1911.

The sample was taken in the mine about 500 feet northeast of the mine mouth and represents 6 feet 6 inches of coal. There was sandstone above the coal and brown shale below it.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 316, 1910, p. 151.

MEEKER. MONTGOMERY MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 12777 (p. 35).

Mine.—Montgomery; SE. ¼ NW. ¼ sec. 29, T. 1 N., R. 94 W.; 3 miles west of Meeker, on the main stage road, and about 42 miles north of the Denver & Rio Grande Railroad.

Coal bed.—No name. Cretaceous age, Mesaverde formation. Thickness, 7 feet 6 inches. Dip, 60° NW.

The bed was measured and sampled by E. T. Hancock on October 18, 1911.

The sample was taken in the mine 474 feet northeast of the mine mouth, in 7 feet 6 inches of coal. There was sandstone above the coal and a brown shale below it.

For a description of the coal beds of Meeker Township see U. S. Geol. Survey Bull. 316, 1910, pp. 141–160.

MEEKER. SULPHUR CREEK MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 12773 (p. 35).

Mine.—Sulphur Creek, in SE. ¼ sec. 3, T. 1 N., R. 94 W., on Sulphur Creek, 2½ miles north of Meeker and 49 miles north of the Denver & Rio Grande Railroad. No railroad connections.

Coal bed.—No name. Cretaceous age; Mesaverde formation.

The bed was measured and sampled at one point in the mine by E. T. Hancock on October 17, 1911, as shown below:

Section of coal bed in Sulphur Creek mine.

Laboratory No.	12773	
Roof, coal (not mined).	*Ft.*	*in.*
Coal	1	8
Sandstone, shaly	0	¼
Coal	0	1
Sandstone, shaly	[a] 0	½
Coal	1	6
Shale, carbonaceous	[a] 0	¾
Coal	2	4½
Floor, shale.		
Thickness of bench mined	5	9
Thickness of coal sampled	5	7¾

[a] Not included in sample.

The sample was taken in the mine, 245 feet north of the mine mouth.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 77, 78, 473.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 415, 1910, p. 155.

MEEKER. WESSON MINE.

Sample.—Bituminous coal; Danforth Hills field; analysis No. 12705 (p. 35).

Mine.—Wesson, on the west side of Coal Creek, in sec. 30, T. 2 N., R. 92 W., 12 miles northeast of Meeker and 55 miles north of the Denver & Rio Grande Railroad.

Coal bed.—No name. Cretaceous age, Mesaverde formation. Thickness, about 22 feet 3½ inches. Dip, 60° SW.

The bed was measured and sampled at one point in the mine by E. T. Hancock on October 6, 1911, as shown below:

Section of coal bed in the Wesson mine.

Laboratory No.	12705	
Coal (not mined).	*Ft.*	*in.*
Coal	2	1
Coal, shaly	0	1
Coal	11	0
Shale, carbonaceous	[a] 0	1
Coal	2	4
Sandstone	[a] 0	½
Coal	2	8
Coal (not mined).		
Thickness of coal mined	18	3½
Thickness of coal sampled	18	2

[a] Not included in sample.

Sample 12705 was taken in the mine, 258 feet northwest of the mine mouth.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 415, 1910, p. 164.

IDAHO.

BOISE COUNTY.

HORSESHOE BEND. HENRY MINE.

Sample.—Subbituminous coal; Horseshoe Bend field; analyses Nos. 12391 and 12703 (p. 35).

Mine.—Henry, a slope mine in NW. ¼ sec. 3, T. 6 N., R. 2 E., at Horseshoe Bend, about 20 miles north of Boise.

Coal bed.—No name. Tertiary age, Payette formation. Thickness irregular, average 18 to 20 inches; dip about 50° west. Roof, black shale; floor, clay.

The bed was sampled and measured at one point in the mine by C. T. Kirk and C. F. Bowen, on June 22 and at one point by C. F. Bowen on October 8, 1911, as shown below:

Section of coal bed in Henry mine.

Laboratory No	12703	
Roof, shale.	*Ft.*	*in.*
Coal	1	6
Clay	[a]0	4
Bone	[a]0	2
Coal	[a]0	4
Bone	[a]0	3
Clay	[a]0	2
Floor, sandstone.		
Thickness of bed	2	9
Thickness of coal sampled	1	6

[a] Not included in sample.

Notes.—Sample 12703 was taken in the Jensen level, 140 feet from the mouth of the slope and about 175 feet below the surface. It represents freshly mined coal.

Sample 12391 was taken about 60 feet from the mouth of the slope and represents 2 feet of coal. As the face had been exposed to the atmosphere for about a year the coal was perhaps more or less weathered.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 531, 1911, p. 249.

CASSIA COUNTY.

OAKLEY. WORTHINGTON MINE.

Sample.—Lignite; Goose Creek district; analysis No. 12643 (p. 36).

Mine.—Worthington; a slope mine SE. ¼ sec. 26 (unsurveyed), T. 16 S., R. 20 E., on Goose Creek, about 22 miles southwest of Oakley.

Lignite bed.—Worthington. Tertiary age, Payette (?) formation. Dip, about 10° east. The bed was measured and sampled at one point in the mine, by C. F. Bowen on September 19, 1911, as shown below:

Section of lignite bed in Worthington mine.

Laboratory No	12643	
Roof, clay.	*Ft.*	*in.*
Lignite	1	8
Clay	[a]0	1½
Lignite	2	4
Thickness of bed	4	1½
Thickness of coal sampled	4	0

[a] Not included in sample.

The sample was taken in the slope about 200 feet from its mouth.

Notes.—This is the most important bed in the Goose Creek district. At the time of sampling in 1911 the mine had been idle for more than a year. For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 531 1913, p. 258.

FREMONT COUNTY.

HADEN. BROWN BEAR MINE.

Sample.--Bituminous coal; Teton Basin field; analysis No. 15115 (p. 36).

Mine.—Brown Bear; in sec. 25, T. 5 N., R. 43 E.; Horseshoe Creek district, about 10 miles southwest of Haden.

Coal bed.—Not named. Cretaceous age; formation not yet determined.

The bed was measured and sampled at one point in the mine by E. G. Woodruff on September 28, 1912, as shown below:

Section of coal bed in Brown Bear mine.

Laboratory No.	15115
Roof, shale.	*Ft. in.*
Coal	3 11
Shale	[a]0 3
Coal	1 1
Shale.	
Thickness of bed	5 3
Thickness of coal sampled	5 0

[a] Not included in sample.

Sample 15115 was taken at the end of the north entry, 950 feet from the portal, where mining had been done recently and where the coal was unweathered.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 541, 1914.

HADEN. HORSESHOE MINE.

Sample.—Bituminous coal; Teton Basin field; analysis No. 15116 (p. 36).

Mine.—Horseshoe; in sec. 6, T. 4 N., R. 44 E.; Horseshoe Creek district, about 10 miles southwest of Haden.

Coal bed.—Not named. Cretaceous age; formation not yet determined.

The bed was measured and sampled at one point in the mine by E. G. Woodruff on September 28, 1912, as shown below:

Section of coal bed in Horseshoe mine.

Laboratory No.	15116
Roof, shale.	*Ft. in.*
Coal	1 11
Sandstone	[a]0 10
Coal	3 4
Coal, very impure	[a]1 9
Coal	3 3
Floor, shale.	
Thickness of bed	11 1
Thickness of coal sampled	8 6

[a] Not included in sample.

Notes.—This sample was taken in an abandoned mine about 200 feet from the portal at a point where the bed is under 60 feet of cover. The mine was dry. The face had been exposed for more than a year and was cleaned before sampling. The coal was probably slightly weathered.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 541, 1914.

ILLINOIS.

MADISON COUNTY.

COLLINSVILLE. CANTINE No. 3 MINE.

Sample.—Bituminous coal; Illinois field; analyses Nos. 10956, 10957, 10958, and 10959 (p. 36).

Mine.—Cantine No. 3, a shaft mine in the Central Illinois district, 1 mile east of Collinsville, on the Vandalia Railroad.

Coal bed.—Herrin coal (Belleville, No. 6) of the United States Geological Survey. Carboniferous age, Carbondale formation. Average thickness, 7 feet 3 inches. The roof is a gray shale, above the shale is a limestone cap rock. The floor is a smooth, hard fire clay.

The bed was measured and sampled at three points in the mine by P. M. Riefkin on September 23, 1910, as described below:

Sections of coal bed in Cantine No. 3 mine.

Section	A		B		C	
Laboratory No.	10956		10957		10958	
Roof, gray shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	4	1	1½	1	6½
"Sulphur"	0	¼	0	⅛	0	¼
Coal	1	4¾	1	2	0	8
"Sulphur"	[a]0	¼	[a]0	1	0	⅛
Coal	0	5¾	0	5½	0	5
"Sulphur"	[a]0	¼	[a]0	¼	[a]0	½
Coal	0	8¾	0	8	1	0
Mother coal	0	¼	[a]0	1	0	¼
Coal	1	3½	1	2½	1	6¼
"Sulphur"	[a]0	1¼	[a]0	1	[a]0	¾
Coal	0	9½	0	5½	0	5¼
Hard blue band	[a]0	1	[a]0	1¼	[a]0	1
Bottom coal	1	5	1	5½	1	10
Floor, fire clay.						
Thickness of bed	7	8½	6	10⅞	7	7⅞
Thickness of coal sampled	7	5¾	6	6⅜	7	5⅝

[a] Not included in sample.

Section A (sample 10956) was measured on the rib, 10 feet from the face of west entry 10 off the north heading, 3,200 feet northwest of the shaft.

Section B (sample 10957) was measured at the face of the first stub on east entry 8 off the north heading, 4,200 feet north of the shaft.

Section C (sample 10958) was measured on the rib 8 feet from the face of east entry 7, off the south heading, 2,500 feet southeast of the shaft.

A composite sample was made by combining face samples 10956, 10957, and 10958. The results of an ultimate analysis of this sample are shown under laboratory No. 10959.

Notes.—The mine is opened by a shaft 165 feet deep. At the time of sampling the coal was undercut with punching machines and broken down with black powder. The tipple was equipped with shaker screens to prepare the coal for various commercial and domestic sizes. There were four loading tracks with a capacity of 40 loaded and 20 empty cars.

MONTGOMERY COUNTY.

PANAMA. PANAMA NO. 1 MINE.

Sample.—Bituminous coal; Illinois field; analyses Nos. 11352, 11353, and 11354 (p. 37).

Mine.—Panama No. 1, a shaft mine in Gresham Township, one-fourth of a mile north of Panama, on the Toledo, St. Louis & Western Railroad.

Coal bed.—Herrin coal (Belleville, No. 6) of the United States Geological Survey. Carboniferous age, Carbondale formation. Thickness, 6 feet 6 inches to 7 feet 6 inches. Roof, limestone; floor, clay. The bed was measured and sampled at two points in the mine by R. Y. Williams, on November 23, 1910, as shown below:

Sections of coal bed in Panama No. 1 mine.

Section	A		B	
Laboratory No.	11352		11353	
Roof, limestone.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	11	1	7
"Sulphur"	[a] Knife edge.		[a]0	½
Coal	0	4½	0	9½
"Sulphur"	[a]0	¼	[a]0	¼
Coal	0	6	0	5½

[a] Not included in sample.

Sections of coal bed in Panama No. 1 mine—Continued.

	Ft.	in.	Ft.	in.
Roof, limestone—Continued.				
"Sulphur"	[a]0	½	[a]0	⅞
Coal	0	2		
Mineral charcoal	0	½		
Coal	1	0	1	4
"Sulphur"	[a]0	¼	[a] Knife edge.	
Coal	0	1	0	7¾
"Sulphur" (blue band)	[a]0	¼	[a]0	⅞
Coal	0	10	0	11¾
"Sulphur"	[a]0	½	[a]0	⅞
Coal	0	11	0	2
"Sulphur"	[a]0	1½	[a]0	½
Coal	0	2	0	1
"Sulphur"	[a]0	¾	[a]0	1
Coal	1	2	0	4
Floor, clay.				
Thickness of bed	7	6	6	7¾
Thickness of coal sampled	7	3	6	3⅝

[a] Not included in sample.

Section A (sample 11352) was measured at the face of room 1, third south stub off the third southeast entry, about 2,000 feet from the shaft.

Section B (sample 11353) was measured at the face of room 4, third north stub off west entry 6, off north entry, 2,000 feet from the shaft.

A composite sample was made by combining face samples 11352 and 11353. The results of an ultimate analysis of this sample are given under laboratory No. 11354.

Notes.—The mine is opened by a shaft about 375 feet deep. The coal is mined by both the room-and-pillar and the panel systems. Entry pillars are left to protect the entries. At the time of sampling the mine was equipped to produce 2,500 tons of coal per 8-hour day.

SALINE COUNTY.

HARRISBURG. O'GARA NO. 9 MINE.

Sample.—Bituminous coal; Illinois field; analysis Nos. 12794 and 12795 (p. 37).

Mine.—O'Gara No. 9, a shaft mine in the Saline County district, 1 mile south of Harrisburg, on the Big Four Railroad.

Coal bed.—No. 5 of the Illinois Geological Survey. Carboniferous age, Carbondale formation. Average thickness, 6 feet; dip 3 per cent northeast. Cleat, northeast and southwest. Occasional dikes are encountered. Roof, a blue shale; 2 to 14 inches of this shale comes down with the coal in mining; floor, a hard clay. Cover at points of sampling, 150 to 240 feet.

The bed was measured and sampled at two points in the mine by H. I. Smith on August 27, 1911, as described below:

Sections of coal bed in O'Gara No. 9 mine.

Section	A		B	
Laboratory No.	12794		12795	
Roof, blue shale.	Ft.	in.	Ft.	in.
Coal, bright	0	7¼	0	6
Mother coal or "sulphur"	[a]0	1½	..	..
Coal, bright	0	6	..	..
"Sulphur"	[a]0	½	[a]0	¼
Coal, laminated with dirt bands	1	3	..	..
Bone or "sulphur"	0	½	..	..
Coal, bright	4	10¼	6	1
Floor, hard clay.				
Thickness of bed	7	5	6	7¼
Thickness of coal sampled	7	3	6	7

[a] Not included in sample.

Section A (sample 12794) was measured at the face of west entry 3, off the main south entry, between rooms 18 and 19, about 3,000 feet from the shaft.

Section B (sample 12795) was measured at the face of the second crosscut in room 11, off west entry 3, off the main south entry.

Notes.—The mine is opened by a shaft 152 feet deep. The coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by electric chain cutting machines and shot down with C, F, FF, and FFF black blasting powder. No explosives were used for brushing the roof or floor. None of the coal was shipped as run-of-mine. The screening equipment consisted of a double-decked bar screen 21 feet long by 7 feet wide, the upper screen having openings 2 inches by 2½ inches and the lower screen openings 1¼ inches square. Pickers were employed on the car in loading. There were three loading tracks with a capacity of 40 empty and 60 loaded cars. The daily output averaged 2,500 tons; the capacity was 3,100 tons.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 90, 507.

For results of tests of coal from this mine see mention of specific tests as follows—Steaming tests: U. S. Geol. Survey Bull. 332, p. 119; Bureau of Mines Bull. 23, pp. 61, 159; briquetting tests, U. S. Geol. Survey Bull. 332, p. 119.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 316, 1907, p. 130.

VERMILION COUNTY.

WESTVILLE. LITTLE VERMILION MINE.

Sample.—Bituminous coal; Illinois field; analysis No. 14682 (p. 37).

Mine.—Little Vermilion, a mine in the Grape Creek district, sec. 19, T. 18 N., R. 11 W., 2 miles south of Westville.

Coal bed.—No. 6 of the Illinois Geological Survey. Carboniferous age, Carbondale formation.

Sample 14682 represents a car of run-of-mine coal used in combustion test (test No. 336) at the Bureau of Mines testing plant at Pittsburgh, Pa. The sample was taken by E. R. Linkenhoker while the car was being unloaded.

INDIANA.

SULLIVAN COUNTY.

DUGGER. VANDALIA NO. 10 MINE.

Sample.—Bituminous coal; bituminous coal field; analyses Nos. 10960 and 10961 (p. 37).

Mine.—Vandalia No. 10, 1½ miles southwest of Dugger, on the Vandalia Railroad.

Coal bed.—No. 4 of the Indiana Geological Survey. Carboniferous age, Allegheny formation. Average thickness, about 5 feet 6 inches. Roof, a gray shale 20 feet thick; above the shale is a sandstone cap rock. Floor, clay 6 to 10 feet thick; below the clay is a sandstone. The bed was measured and sampled at two points in the mine by R. Y. Williams on September 21, 1910, as described below:

Sections of coal bed in Vandalia No. 10 mine.

Section	A		B	
Laboratory No.	10960		10961	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Rash	..	..	[a]0	½
Coal, hard and bright	1	5	2	8
Bone and shale	..	..	[a]0	2
Natural charcoal	[a]0	¾	..	..
Coal, hard and bright	0	5½	0	3
"Sulphur" and bone	..	..	[a]0	½
Natural charcoal	0	½	..	..
Coal, hard and bright	1	¾	1	11½
Bottom coal	..	..	0	2
Shale parting	[a]0	2	..	..
Coal, hard and bright	2	3	..	..
Floor, clay.				
Thickness of bed	5	5½	5	3½
Thickness of coal sampled	5	2¾	5	½

[a] Not included in sample.

Section A (sample 10960) was measured at the face of room 2, off east entry 6 off north entry 2, off the main east entry.

Section B (sample 10961) was measured at the face of room 12, off east entry 6 off north entry 2, off the main east entry.

Note.—At the time of sampling the output was 1,600 tons a day.

WARRICK COUNTY.

ELBERFELD. ELBERFELD MINE.

Sample.—Bituminous coal; bituminous coal field; analysis No. 16578 (p. 37).

Mine.—Elberfeld, a shaft mine 1 mile south of Elberfeld station on the E. & I. siding.

Coal bed.—No. 5 of the Indiana Geological Survey. Carboniferous age, Allegheny formation. Average thickness, 4 feet 6 inches; dip southwest, in places as much as 5°. The coal has a cleat, but no faults or horsebacks. There are no persistent partings, but kidney "sulphur" and "sulphur" flakes occur at intervals. The roof is a hard black shale which does not come down in mining, overlain by a softer shale. The floor is a hard fire clay which mixed slightly with the coal in loading. Cover at point of sampling, 185 feet.

The bed was measured and sampled at one point in the mine by J. J. Rutledge and H. D. Mason, jr., on February 6, 1913, as described below:

Section of coal bed in Elberfeld mine.

Section	A
Laboratory No	16578
Roof, hard black shale.	*Ft. in.*
Coal	1 0
"Steel" band	0 ¼
Coal	0 8
"Steel" band	0 ¼
Coal	2 4
Floor, hard fire clay.	
Thickness of bed	4 ½
Thickness of coal sampled	4 ½

Section A (sample 16578) was measured in the new manway off the main west entry, 200 feet north of the shaft bottom.

Notes.—The mine is opened by a shaft 185 feet deep, and the coal is mined by the room-and-pillar, double-entry system. At the time of sampling the coal was undercut by hand and shot down with C and F black blasting powder. Seventy-five per cent of the coal was shipped as run-of-mine. The coal was hoisted on a self-dumping cage to a wooden tipple. The tipple had shaking screens 40 feet long by 5 feet wide with 2½-inch holes. Pickers were employed on the cars while loading. There were three loading tracks, with a capacity of 25 empty and 25 loaded cars. The average daily output was 250 tons, the capacity 300 tons. About 50 acres remained to be mined from the present opening.

KANSAS.

CRAWFORD COUNTY.

FRANKLIN. WESTERN NO. 16 MINE.

Sample.—Bituminous coal; Kansas field; analysis No. 17032 (p. 37).

Mine.—Western No. 16; one-fourth of a mile west of Franklin, on the Missouri Pacific Railway and the Atchison, Topeka & Santa Fe Railway.

Coal bed.—Weir-Pittsburg. Carboniferous age, Cherokee shale. Average thickness, 2 feet 10 inches; dip, northwest. The coal is very clean except for thin streaks of pyrite near the middle of the coal bed. Rolls, clay veins, horsebacks, and slips are numerous and a source of much added expense in mining. Roof, bluish shale; floor, underclay, which at times heaves. Cover at point of sampling, about 220 feet.

The bed was measured and sampled at one point in the mine by C. S. Stevenson on April 16, 1913.

Sample 17032 was taken at a point in the mine 2,000 feet southwest of the shaft, and represents 2 feet 11 inches of clean coal. There was shale above the coal and clay below it.

Notes.—The mine is opened by a shaft 219 feet deep. The coal is mined by the double-entry, room-and-pillar system. At the time of sampling the coal was shot off the solid and gave trouble in sticking to the roof. In the haulageways the top was brushed to a height of 2 feet, but in the air course the top was not brushed. The coal was principally marketed for railroad purposes. The output was about 700 tons a day.

FULLER. SHERIDAN NO. 2 MINE.

Sample.—Bituminous coal; Kansas field; analyses Nos. 11181, 11182, 11183, and 11184 (p. 38).

Mine.—Sheridan No. 2, a shaft mine in sec. 12, T. 29 N., R. 25 E., in the southwest corner of the town of Fuller, on the Kansas City Southern Railway.

Coal bed.—Wier-Pittsburg. Carboniferous age, Cherokee shale. Average thickness, 3 feet; bed horizontal. Roof, dark gray shale 10 to 12 feet thick; floor, hard shale.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 12, 1910, as described below:

Sections of coal bed in Sheridan No. 2 mine.

Section	A		B		C	
Laboratory No.	11181		11182		11183	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	0	1	0	1	0
"Sulphur" band	[a]0	½	[a]0	½	[a]0	½
Coal	1	4	1	4	1	4
"Sulphur" band	[a]0	½	[a]0	½	[a]0	½
Coal	0	7	0	7	0	7
Floor, shale.						
Thickness of bed	3	0	3	0	3	0
Thickness of coal sampled	2	11	2	11	2	11

[a] Excluded from sample.

Section A (sample 11181) was measured at the face, 2,500 feet southwest of the shaft.

Section B (sample 11182) was measured at the face, 2,500 feet northeast of the shaft.

Section C (sample 11183) was measured at the face, 2,500 feet northwest of the shaft.

A composite sample was made by combining face samples 11181, 11182, and 11183. The results of an ultimate analysis of this sample are shown under laboratory No. 11184.

Notes.—At the time of sampling the output was 375 to 400 tons a day.

YALE. WESTERN NO. 13 MINE.

Sample.—Bituminous coal; Kansas field; analyses Nos. 11209, 11210, 11211, and 11212 (p. 38).

Mine.—Western No. 13, a shaft mine in sec. 26, T. 30 N., R. 25 E., three-fourths of a mile northwest of Yale, on the Missouri Pacific Railroad.

Coal bed.—Wier-Pittsburg (?). Carboniferous age, Cherokee shale. Average thickness, 3 feet; practically horizontal. Roof, dark gray shale 30 feet thick; floor, soft shale. Cover at points of sampling, 87 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 14, 1910, as described below:

Sections of coal bed in Western No. 13 mine.

Section	A		B		C	
Laboratory No.	11209		11210		11211	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	8	1	8	1	8
"Sulphur" band	[a]0	½	[a]0	½	[a]0	½
Coal	1	3½	1	3½	1	3½
Floor, shale.						
Thickness of bed	3	0	3	0	3	0
Thickness of coal sampled	2	11½	2	11½	2	11½

[a] Not included in sample.

Section A (sample 11209) was measured at the face, 2,600 feet north of the shaft.

Section B (sample 11210) was measured at the face, 2,800 feet southeast of the shaft.

Section C (sample 11211) was measured at the face, 2,800 feet southwest of the shaft.

A composite sample was made by combining face samples 11209, 11210, and 11211. The results of an ultimate analysis of this sample are shown under laboratory No. 11212.

Notes.—The mine is opened by a shaft about 90 feet deep. At the time of sampling the output was 500 tons a day.

LEAVENWORTH COUNTY.

LANSING. PENITENTIARY MINE.

Sample.—Bituminous coal; Kansas field; analyses Nos. 12841, 12842, 12843, and 12844 (pp. 38, 39).

Mine.—Penitentiary; sec. 19, T.9S., R. 23E.; Leavenworth district; on Missouri Pacific, Atchison, Topeka & Santa Fe, and Union Pacific railroads.

Coal bed.—Bevier. Carboniferous age, Pennsylvanian series, Cherokee shale. Roof, chiefly firm, dark drab shale; floor, thin underclay on limestone. Thickness, regular, average 22 inches. Cover at points of sampling, 720 feet.

The bed was measured and sampled at three points in the mine by M. Albertson on September 15, 1911.

Sample 12841 was taken 1,100 feet south and 1,400 feet east of the shaft in 22 inches of coal.

Sample 12842 was taken 4,400 feet east and 500 feet north of the shaft in 22 inches of coal.

Sample 12843 was taken 2,300 feet east and 2,600 feet north of the shaft in 22 inches of coal.

A composite sample was made by mixing face samples 12841, 12842, and 12843. The results of the ultimate analysis of this sample are shown under laboratory No. 12844.

LEAVENWORTH. NO. 1 MINE.

Sample.—Bituminous coal; Kansas field; analyses Nos. 12849, 12850, 12851, and 12852 (p. 39).

Mine.—No. 1; Leavenworth district, at Leavenworth; sec. 36, T. 8 S., R. 22 E.; on the Missouri Pacific Railway.

Coal bed.—Bevier. Carboniferous age, Pennsylvanian series, Cherokee shale. Roof, chiefly firm, dark drab shale; floor, thin underclay on limestone. Thickness, regular, average 22 inches. Cover at points of sampling, 720 feet.

The bed was measured and sampled at two points in the mine by M. Albertson on September 16, 1911.

Sample 12849 was taken 3,200 feet north and 560 feet west of the shaft in 21 inches of coal.

Sample 12850 was taken 160 feet west and 3,100 feet south of the shaft in 21 inches of coal.

Sample 12851 was taken 110 feet east and 3,310 feet north of the shaft in 22 inches of coal.

A composite sample was made by mixing face samples 12849, 12850, and 12851. The results of an ultimate analysis of this sample are shown under laboratory No. 12852.

LEAVENWORTH. NO. 3 MINE.

Sample.—Bituminous coal; Kansas field; analyses Nos. 12845, 12846, 12847, and 12848 (p. 39).

Mine.—No. 3; Leavenworth district, at Leavenworth; sec. 25, T. 8 S., R. 22 E.; on Missouri Pacific Railway.

Coal bed.—Bevier. Carboniferous age, Pennsylvanian series, Cherokee shale. Roof, chiefly firm, dark drab shale; floor, thin underclay on limestone. Thickness, regular, average 22 inches. Cover, 720 feet. The bed was measured and sampled at two points in the mine by M. Albertson on September 18, 1911.

Sample 12845 was taken 3,400 feet southeast of the shaft in 22 inches of coal.

Sample 12846 was taken 3,800 feet southeast of the shaft in 22 inches of coal.

Sample 12847 was taken 3,350 feet southeast of the shaft in 22 inches of coal.

A composite sample was made by mixing face samples 12845, 12846, and 12847. The results of an ultimate analysis of this sample are shown under laboratory No. 12848.

KENTUCKY.

LETCHER COUNTY.

FLAT GAP, VA. LOCAL MINE.

Sample.—Bituminous coal; eastern Kentucky field; analysis No. 15172 (p. 39).

Mine.—A local drift mine, 1½ miles south of Flat Gap, Va., 10 miles from the nearest railroad.

Coal bed.—Lower Standiford. Carboniferous age, Wise formation. Thickness uniform, nearly flat. Roof, shale; floor, clay. Cover at points of sampling, 25 feet.

The bed was measured and sampled at one point in the mine by Charles Butts on November 1, 1912.

Sample 15172 was taken in the entry about 15 feet from the mine mouth, in 3 feet 2 inches of coal, all of which was included in the sample. The roof at this point was shale and the floor was clay.

Notes.—Bright, firm coal, sold for domestic use.

FLAT GAP, VA. J. H. MULLIN MINE.

Sample.—Bituminous coal; eastern Kentucky field; analysis No. 15173 (p. 39).

Mine.—J. H. Mullin, a local mine 3½ miles southwest of Flat Gap, Va., and 10 miles from the nearest railroad.

Coal bed.—Lower Bolling. Carboniferous age, Wise formation. The bed was measured and sampled at one point in the mine by Charles Butts on November 1, 1912, as described below:

Section of coal bed in J. H. Mullin mine.

Laboratory No.	15173	
	Ft.	*in.*
Coal	2	3
Clay	[a]0	2
Coal	1	6
Clay	[a]0	4
Coal	0	4
Thickness of bed	4	7
Thickness of coal sampled	4	1

[a] Not included in sample.

The above section was measured in the main entry, 40 feet from the mine mouth.

JENKINS. CONSOLIDATION No. 204 MINE.

Sample.—Bituminous coal; eastern Kentucky field; analyses Nos. 14904 and 14905 (p. 39).

Mine.—Consolidation No. 204, a drift mine at Jenkins, on the Baltimore & Ohio Railroad.

Coal bed.—Upper Elkhorn. Carboniferous age, Pottsville group. Thickness, uniform, flat. Roof, shale; floor, clay.

The bed was measured and sampled at two points in the mine by Charles Butts on October 12, 1912, as described below:

Sections of coal bed in Consolidation No. 204 mine.

Section	A		B	
Laboratory No.	14904		14905	
	Ft.	in.	Ft.	in.
Roof, shale.				
Coal	3	7	4	0
Clay	[a]0	3	[a]0	5
Coal	3	8½	4	0
Floor, clay.				
Thickness of bed	7	6½	8	5
Thickness of coal sampled	7	3½	8	0

[a] Not included in sample.

Section A (sample 14904) was measured at the face of the main heading, 1,700 feet from the mine mouth.

Section B (sample 14905) was measured at the face of the east heading, 1,700 feet from mine mouth.

Notes.—The clay parting near the middle of the bed is soft and friable and is utilized as a mining bench. Most of the bed is coal of medium hardness and brightness, but the lower 18 inches includes a considerable proportion of tough hard splint coal. A large part of the output was being shipped to Michigan, where it was utilized in by-product ovens.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 348, p. 37.

JENKINS. LOCAL MINE.

Sample.—Bituminous coal; eastern Kentucky field; analyses Nos. 14970 and 14971 (p. 40).

Mine.—A local drift mine, 5 miles northeast of Jenkins, near the Baltimore & Ohio Railroad.

Coal bed.—Lower Elkhorn; Carboniferous age, Pottsville group. Thickness uniform, flat. Roof, sandstone; floor, clay.

The bed was measured and sampled by Charles Butts, October 16, 1912, as described below:

Section of coal bed in mine 5 miles north of Jenkins.

Laboratory No.	14970 14971	
	Ft.	in.
Roof, sandstone.		
Coal	[a]0	8
Coal, laminated	[b]1	0
Coal	[a]1	4½
Floor, clay.		
Thickness of bed	3	½

[a] Included in sample 14970. [b] Included in sample 14971.

The section was measured at the face of the entry, 50 feet from the mouth of the mine.

Notes.—The laminated bench of soft coal is 1 foot thick and is high in impurities. The output is used for local, domestic, and steam purposes.

MUHLENBERG COUNTY.

CENTRAL CITY. CENTRAL MINE.

Sample.—Bituminous coal; western Kentucky field; analyses Nos. 13257, 13258, 13259, and 13260 (p. 40).

Mine.—Central, a shaft mine one-eighth of a mile south of the depot at Central City, on the Owensburg-Russelville branch of the Louisville & Nashville Railroad and on the Louisville branch of the Illinois Central Railroad.

Coal bed.—No. 9 of the Kentucky Geological Survey. Carboniferous age, Pottsville group. Thickness, 4 feet 9 inches to 6 feet. The bed dips north about 15 feet per mile. The coal contains thin streaks of "sulphur" and some "sulphur" balls. The roof is a black shale 2 to 4 feet thick and is of good quality. The floor is a clay which weathers when exposed to air. Upon exposure for a year it weathers to an average depth of 6 inches and this feature is utilized by driving the narrow work about one year in advance of turning the rooms. Cover at points of sampling, about 170 feet.

The bed was measured and sampled at three points in the mine by C. S. Stevenson on January 22 and 23, 1912.

Sample 13257 was taken at the face of the main south entry, and represents 5 feet 6 inches of clear coal.

Sample 13258 was taken in room 20, off the main south blind entry, and represents 5 feet 6 inches of clear coal.

Sample 13259 was taken in room 3, off sixth south entry, off the main west entry, and represents 5 feet 6 inches of clear coal.

A composite sample was made by mixing face samples 13257, 13258, and 13259. The results of an ultimate analysis of this sample are given under laboratory No. 13260.

Notes.—The mine is opened by a shaft 170 feet deep and is worked on the room-and. pillar system. At the time of sampling, all the coal was undercut with picks to one-half the depth of the drill hole and broken down with F and FF black blasting powder. Most of the output was used for domestic purposes, a small amount being sold for transportation purposes.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 106, 542.

For results of tests of coal from this mine see mention of specific tests as follows—Steaming tests: Bull. 13, Bureau of Mines, pp. 137, 274; coking tests: U. S. Geol. Survey Bull. 336, 1908, pp. 23, 30, 39; cupola tests of coke: U. S. Geol. Survey Bull. 336, pp. 50, 53, 56, 59, 62.

CLEATON. BEVIER MINE.

Sample.—Bituminous coal, western Kentucky field; analyses Nos. 15339, 15340, and 15341 (p. 40).

Mine.—Bevier, a shaft mine at Cleaton, opposite the railroad station, on the Louisville & Nashville Railroad.

Coal bed.—No. 9 of the Kentucky Geological Survey. Carboniferous age, Pottsville group. Average thickness, 5 feet 4 inches. Dip, 1° southwest. The coal bed has no persistent partings, but contains discontinuous "sulphur" flakes and bands. The roof is a firm, hard, black shale about 6 inches thick; above the shale is a main cap rock. The floor is a hard, smooth fire clay, which crumbles on exposure to the air. The material from the roof and floor did not mix with the coal in mining. Cover at points of sampling, about 130 feet.

The bed was measured and sampled at three points in the mine by H. D. Mason, jr., on December 23, 1912.

Sample 15339 was taken at the face of room 15, off the eighth north air course, 4,100 feet from the shaft. The sample represents 5 feet 3 inches of coal.

Sample 15340 was taken at the face of room 21, off seventh north entry, 3,800 feet from the shaft. The sample represents 5 feet 6 inches of coal.

Sample 15341 was taken at the face of eighth south entry, 4,500 feet from the shaft. The sample represents 5 feet 3 inches of coal.

At all three places some thin "sulphur" bands and flakes were present and are included in the samples.

Notes.—The mine is opened by a shaft 69 feet deep and the coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by compressed-air machines and broken down with FF black blasting powder. Machine cuttings were loaded with the coal. The tipple had a perforated screen 12 feet long and 5 feet wide, with 1 to 4 inch openings. About 38 per cent of the coal going to the screen passed through the 1-inch holes. Thirty per cent of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. The lumps loaded were large and of good appearance. There were three loading tracks, with a capacity of 22 empty and 22 loaded cars. The output was about 750 tons a day, all from advance work, but the intent was to increase it to 1,000 tons. There was approximately 900 acres of unmined coal tributary to this tipple.

GRAHAM. SKIBO MINE.

Sample.—Bituminous coal; western Kentucky field; analyses Nos. 16674 and 16675 (p. 41).

Mine.—Skibo, a slope mine, 2 miles north of Graham, on a spur of the Illinois Central Railroad.

Coal bed.—No. 9 of the Kentucky Geological Survey. Carboniferous age, Pottsville group. Average thickness, 4 feet 10 inches; dip, 1 per cent southwest. Only one fault has been encountered; it extends miles. There are no persistent partings in the coal, although "sulphur" occurs in flakes and bands. The roof is a gray shale, 18 inches thick, which comes down with the coal in places; above the shale is a cap rock. The floor is a hard smooth fire clay. Particles from the roof and floor did not mix with the coal in mining. Cover at points of sampling, about 150 feet.

The bed was measured and sampled at two points in the mine by H. D. Mason, jr., on May 5, 1913.

Sample 16674 was cut from the face of west entry 2, off north entry 1, 4,000 feet north of the mine mouth. The bed at this point was 4 feet 10 inches thick. The sample represents the entire section.

Sample 16675 was cut from the face of room 2, off seventh east entry, 3,600 feet north of the mine mouth. The bed at this point was 4 feet 9 inches thick. The sample represents the entire thickness of the bed.

Notes.—The coal is mined by the room-and-pillar double-entry system. At the time of sampling the coal was undercut with chain machines and broken down with FF black blasting powder. The cuttings were loaded out with the coal. About 40 per cent of the coal was shipped as run-of-mine. The tipple had shaking screens 25 feet long and 6 feet wide, with holes 1 to 2½ inches in diameter. About 22 per cent of the coal going to the screens passed through the screens. Pickers were employed on the cars in loading. The lumps were large and of good appearance. There were three loading tracks, with a capacity of 50 empty and 50 loaded cars. The daily output was 922 tons, with a capacity of 1,000 tons. The greater part of the coal was from advance workings, although some of it was from pillars. There were approximately 2,000 acres of unmined coal tributary to this opening.

OHIO COUNTY.

McHENRY. McHENRY MINE.

Sample.—Bituminous coal; western Kentucky field; analyses Nos. 15102 and 15103 (p. 41).

Mine.—McHenry, a slope mine near McHenry, on the Illinois Central Railroad.

Coal bed.—No. 9 of the Kentucky Geological Survey. Carboniferous age, Pottsville group. Thickness, fairly uniform, average 4 feet 6 inches. Roof, dark shale, overlain by a blue sandstone. Floor, hard smooth underclay. Particles of the floor got mixed with the coal in mining.

The bed was measured and sampled at two points in the mine by H. D. Mason, jr., on November 13, 1912.

Sample 15102 was taken at the face of room 4, off south entry 3; 9,000 feet from the mine mouth. The sample includes the entire section, 4 feet 6 inches, except a "sulphur" band 26 inches above the bottom.

Sample 15103 was taken at the face of room 1, off north entry 1; 8,500 feet from the mine mouth. The sample includes the entire section, 4 feet 6 inches, except a "sulphur" band 26 inches above the bottom.

Notes.—The mine was not being operated at the time these samples were taken. In mining, the coal was undercut by hand in the coal and broken down with FF black blasting powder. About 50 per cent of the coal was shipped as run-of-mine. Pickers were employed on the cars. The lumps were large and made a good appearance on the cars. The track had a capacity of 40 loaded cars and 30 empty cars. The average daily production had been 700 tons. There was approximately 1,000 acres of unmined coal tributary to this opening.

PIKE COUNTY.

HELLIER. EDGEWATER MINE.

Sample.—Bituminous coal; Russell Fork field; analysis No. 12004 (p. 41).

Mine.—Edgewater, 1 mile southeast of Hellier post office.

Coal bed.—Upper Elkhorn; Carboniferous age; Kanawha (?) formation. Thickness, 3 feet 6 inches. A sample of run-of-mine coal from this mine was taken on March 28, 1911, by C. H. Wegemann, from three loaded railroad cars, the coal having been mined the previous day.

MISSOURI.[a]

ADAIR COUNTY.

KIRKSVILLE. STAR MINE.

Sample.—Bituminous coal; Novinger field; analyses Nos. 14799 and 14800 (p. 41).

Mine.—Star, a shaft mine in the Kirksville district, in sec. 8, T. 63 N., R. 15 W., one-half mile west of Kirksville.

Coal bed.—Bevier. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 14 to 36 inches. Cover, 290 feet. The bed was measured and sampled at two points in the mine by M. Albertson on September 28, 1912.

Sample 14799 was taken 30 feet south of the shaft and represented 17½ inches of coal.

Sample 14880 was taken 12 feet northeast of the shaft and represented 20 inches of coal.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 114, 562.

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[a] The coal fields of Missouri are described in detail in The Coal Deposits of Missouri, by Henry Hinds, Missouri Bureau of Geology and Mines, vol. 11, ser. 2, 1913.

AUDRAIN COUNTY.

MARTINSBURG. No. 1 MINE.

Sample.—Bituminous coal; Vandalia field; analyses Nos. 11483, 11484, 11485, and 11478 (p. 41).

Mine.—No. 1, a shaft mine in sec. 14, T. 50 N., R. 7 W., one-fourth mile west of Martinsburg, with connections to the Wabash Railroad and the Chicago & Alton Railroad.

Coal bed.—Mulky. Carboniferous age, Pennsylvanian series, Cherokee shale. Average thickness, 2 feet 6 inches; bed practically horizontal. Roof, calcareous shale. Floor, fire clay, 10 inches thick, underlain by limestone. Cover at points of sampling, 135 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 19, 1910.

Sample 11483 was taken at the long-wall face, 600 feet south of the shaft.

Sample 11484 was taken at the long-wall face, 600 feet southwest of the shaft.

Sample 11485 was taken at the face, 800 feet southwest of the shaft.

At each of the points sampled the coal bed was 2½ feet thick, overlain by 8 inches of "draw slate," above which was the shale roof. The floor was clay.

A composite sample was made by combining face samples 11483, 11484, and 11485. The results of an ultimate analysis of this sample are given under laboratory No. 11478.

Notes.—The shaft is 135 feet deep; the long-wall system of mining is employed. At the time of sampling the output was 40 tons a day.

BARTON COUNTY.

MINDENMINES. No. 17 MINE.

Sample.—Bituminous coal; Minden field; analyses Nos. 11185, 11186, 11187, and 11188 (p. 42).

Mine.—No. 17, a shaft mine in sec. 30 T. 32 N., R. 33 W., 2 miles northwest of Mindenmines post office, on the Missouri Pacific Railway.

Coal bed.—Wier-Pittsburg Lower. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 10 inches to 3 feet; bed, horizontal. Roof, shale 20 feet thick, overlain by sandstone 2½ feet thick; floor, soft shale. Cover, 60 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 11, 1910.

Sample 11185 was taken at the face, 1,000 feet northeast of the shaft, where there was 2 feet 10 inches of clean coal.

Sample 11186 was taken at the face, 1,000 feet southeast of the shaft, where there was 2 feet 10 inches of clean coal.

Sample 11187 was taken at the face, 1,500 feet northwest of the shaft, where there was 2 feet 10 inches of clean coal.

There was shale above and below the coal at each of the points sampled. The samples represent the entire thickness of the bed.

A composite sample was made from face samples 11185, 11186, and 11187. The results of an ultimate analysis of this sample are given under laboratory No. 11188.

Notes.—The shaft is 64 feet deep. The coal is mined by the room-and-pillar system. At the time of sampling the coal was shot off the solid. The mine was comparatively new and was equipped to produce 400 tons a day.

MINDENMINES. PULLEN No. 1 MINE.

Sample.—Bituminous coal; Minden field; analyses Nos. 11177, 11178, 11179, and 11180 (p. 42).

Mine.—Pullen No. 1, a shaft mine in sec. 28, T. 32 N, R. 33 W., 1 mile northeast of Mindenmines post office, on a branch of the Missouri Pacific Railway.

Coal bed.—Wier-Pittsburg Lower. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 6 inches. Bed practically horizontal. Roof, shale 5 to 10 feet thick; floor, soft clay. Cover at points of sampling, about 40 feet.

The bed was measured and sampled at three points in Pullen No. 1 mine by R. D. Killian on November 10, 1910.

Sample 11177 was taken at the south face, 600 feet from the shaft, and represents 2 feet 6 inches of coal. There was a "sulphur" band one-sixteenth inch thick near the middle of the bed.

Sample 11178 was taken at the west face, 400 feet from the shaft, and represents 2 feet 7 inches of coal. There was a soft-coal ("mother coal") parting about one-sixteenth inch thick near the middle of the bed.

Sample 11179 was taken at the northeast face, 600 feet from the shaft, and represents 2 feet 6 inches of coal. There was a "sulphur" band one-sixteenth inch thick near the middle of the bed.

A composite sample was made by combining face samples 11177, 11178, and 11179. The results of an ultimate analysis of this sample are given under laboratory No. 11180.

Notes.—The shaft is 40 feet deep. The coal is mined by the room-and-pillar system. At the time of sampling the coal was blasted off the solid. The output was 60 tons a day.

BATES COUNTY.

AMSTERDAM. No. 1 MINE.

Sample.—Bituminous coal; Foster field; analyses Nos. 11306, 11307, 11308, and 11309 (pp. 42, 43).

Mine.—No. 1, in sec. 20, T. 41 N., R. 33 W., in the southern part of the village of Amsterdam, on the Kansas City Southern Railway.

Coal bed.—Mulberry. Carboniferous age, Pennsylvanian series, Pleasonton shale. Thickness variable, average about 2 feet; bed horizontal. Roof, calcareous shale, overlain by limestone; floor, hard clay, underlain by limestone. Cover, 90 to 100 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 3, 1910.

Sample 11306 was taken at the face, 400 feet east of the shaft.

Sample 11307 was taken at the face, 400 feet south of the shaft.

Sample 11308 was taken 400 feet southeast of the shaft.

The lower 1 foot of coal in each section contained bands of sulphur and bone.

Each sample represents 2 feet 6 inches of coal, the thickness of the bed.

A composite sample was made by combining face samples 11306, 11307, and 11308. The results of an ultimate analysis of the sample are given under laboratory No. 11309.

Notes.—The shaft is 100 feet deep. The mine is worked by the room-and-pillar system. At the time of sampling the output was about 25 tons a day.

HUME. No. 1 MINE.

Sample.—Bituminous coal; Foster field; analyses Nos. 11226, 11227, 11228, and 11229 (p. 43).

Mine.—No. 1, in sec. 4, T. 38 N., R. 33 W., Howard Township, 4 miles east of Hume and 10 miles west of Rich Hill, on a branch of the St. Louis & San Francisco Railroad.

Coal bed.—Mulberry. Carboniferous age, Pennsylvanian series, Pleasonton shale. Thickness, 2 feet 6 inches to 3 feet. The bed lies horizontal. Roof, "draw slate" 6 inches thick, overlain by shale 15 feet thick; floor, soft clay. Cover, about 50 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 21, 1910.

Sample 11226 was taken at the face, 900 feet northwest of the shaft.

Sample 11227 was taken at the face, 1,100 feet west of the shaft.

Sample 11228 was taken at the face, 700 feet southwest of the shaft.

At each of the points sampled the coal bed was 2½ feet thick.

A composite sample was made by combining face samples 11226, 11227, and 11228. The results of an ultimate analysis of this sample are given under laboratory No. 11229.

Notes.—The shaft is 35 feet deep. The coal is mined by the room-and-pillar system. At the time of sampling the capacity was 50 tons a day.

NEW HOME. NEW HOME NO. 1 MINE.

Sample.—Bituminous coal; Rich Hill field; analyses Nos. 11222, 11223, 11224, and 11225 (p. 43).

Mine.—No. 1, a shaft mine in sec. 30, T. 39 N., R. 32 W., at New Home. It has railroad facilities for shipment over the St. Louis & San Francisco, the Missouri Pacific, and the Kansas City Southern railroads.

Coal bed.—Lower Rich Hill. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 5 feet. The bed is horizontal in general but more or less undulating. Roof, limestone, 1 foot thick, overlain by shale about 4 feet thick; floor, clay, 6 feet thick. Cover, 260 to 280 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 22, 1910.

Sample 11222 was taken at the face, 200 feet northeast of the shaft, and represents 5 feet of clean coal.

Sample 11223 was measured at the face, 150 feet southeast of the shaft, and represents 5 feet of clean coal.

Sample 11224 was measured at the face, 200 feet south of the shaft, and represents 5 feet of clean coal.

At each of the points sampled the entire thickness of the bed was included.

A composite sample was made by combining face samples 11222, 11223, and 11224. The results of an ultimate analysis of this sample are given under laboratory No. 11225.

Notes.—The shaft is 278 feet deep. The coal is mined by the room-and-pillar system. The output at the time of sampling was about 50 tons a day.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 116, 567.

RICH HILL. FLEMING PIT NO.1.

Samples.—Bituminous coal; Rich Hill field; analyses Nos. 11197, 11198, 11199, and 11200 (p. 43).

Mine.—Fleming Pit No. 1, an open-pit mine in sec. 36, T. 39 N., R. 32 W., 2 miles northwest of Rich Hill, on a branch of the Missouri Pacific Railway.

Coal bed.—Lower Rich Hill. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 5 feet. The bed lies nearly horizontal. Roof, shale, 4 feet thick; floor, hard clay. Cover, 13 feet.

The bed was measured and sampled at three points in the pit by R. D. Killian on November 19, 1910.

Sample 11197 was taken at the face on the north side of the pit.

Sample 11198 was taken at the face on the east side of the pit.

Sample 11199 was taken at the face on the south side of the pit.

At each of the points sampled the bed was 5 feet thick, and had a 2-inch "sulphur" band 1 foot 10 inches from the bottom. Each sample represents 4 feet 10 inches of coal, the 2-inch band being excluded.

A composite sample was made by combining face samples 11197, 11198, and 11199. The results of an ultimate analysis of this sample are shown under laboratory No. 11200.

Notes.—At the time of sampling Fleming Pit No. 1 was one of the largest strippings near Rich Hill. The daily output was about 50 tons.

RICH HILL. RITCHIE PITS NOS. 1 AND 2.

Sample.—Bituminous coal; Rich Hill field; analyses Nos. 11193, 11194, 11195, and 11196 (p. 44).

Mine.—Ritchie Pits No. 1 and No. 2, in sec. 17, T. 38 N., R. 31 W., 2 miles south of Rich Hill, on a branch of the Missouri Pacific Railroad.

Coal bed.—Lower Rich Hill. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, about 5 feet. The bed is slightly inclined. Roof, black shale, 4 feet thick; floor, clay. Cover, 10 to 16 feet.

The bed was measured and sampled at two points in No. 1 pit and at one point in No. 2 pit by R. D. Killian on November 18, 1910.

Sample 11193 was taken at the face on the north side of open pit No. 1.

Sample 11194 was taken at the face on the south side of open pit No. 1.

At each of the above points the bed measured 4 feet 11 inches. There was a "sulphur" band 1 inch thick at the bottom and a 2-inch band of "lignite" about 1 foot 8 inches from the bottom. These parts were excluded from the samples.

A composite sample was made by combining face samples 11193 and 11194. The results of an ultimate analysis of this sample are given under laboratory No. 11195.

Sample 11196 was cut from the face of the south side of open pit No. 2. The bed measured 2 feet at this point, all of which is represented in the sample.

Notes.—At pit No. 1 a slope had been opened and underground work was being carried on. The output was 30 tons a day.

BOONE COUNTY.

COLUMBIA. NO. 1 AND NO. 2 MINES.

Samples.—Bituminous coal; Bevier field; analyses Nos. 11475, 11476, and 11477 (p. 44).

Mines.—No. 1 and No. 2, shaft mines in the Columbia district, in sec. 4, T. 48 N., R. 12 W., 3½ miles northeast of Columbia, a shipping station on the Wabash Railroad.

Coal bed.—Bevier (?). Carboniferous age, Pennsylvanian series, Cherokee shale. Average thickness, 3 feet; slight dip to southeast. Roof, blue calcareous shale, 3 feet thick; floor, fire clay. Cover, 100 to 110 feet.

The bed was measured and sampled at two points in No. 1 mine and at one point in No. 2 mine by R. D. Killian on December 17, 1910.

Sample 11475 was taken at the face, 50 feet west of the shaft, mine No. 2.

Sample 11476 was taken at the face, 110 feet north of the shaft, mine No. 1.

Sample 11477 was taken 50 feet west of the shaft, mine No. 1.

Each sample represents 3 feet of coal, the thickness of the bed.

Notes.—The shafts are 110 feet deep. The coal is mined by the room-and-pillar system. At the time of sampling the output was 30 tons a day. The coal was used for local consumption.

COLUMBIA. PRATHER NO. 1 MINE.

Sample.—Bituminous coal; Bevier field; analyses Nos. 11479, 11480, 11481, and 11482 (p. 44).

Mine.—Prather No. 1, a shaft mine in sec. 11, T. 49 N., R. 12 W., 4½ miles north of Columbia, which is the nearest shipping point on the Wabash Railroad.

Coal bed.—Bevier. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 40 to 42 inches. The bed is more or less undulating with a slight dip to the southeast. Roof, black shale; floor, hard fire clay. Cover, 35 to 65 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 17, 1910.

Sample 11479 was taken at the face, 600 feet southwest of the shaft.

Sample 11480 was taken at the face, 600 feet west of the shaft.

Sample 11481 was taken at the face, 800 feet east of the shaft.

At each of the points sampled the bed was 3 feet 6 inches thick, all of which was included in the sample.

There is an 18-inch bed of coal 30 feet above the bed mined and an 18-inch bed of coal about 10 feet below it.

A composite sample was made from face samples 11479, 11480, and 11481. The results of an ultimate analysis of this sample are given under laboratory No. 11482.

Notes.—The shaft is 36 feet deep. The coal is mined by the room-and-pillar method. At the time of sampling the capacity was 35 tons a day; the coal was used for local purposes.

CALLAWAY COUNTY.

FULTON. NO. 1 MINE.

Sample.—Bituminous coal; Bevier field; analyses Nos. 11501, 11502, 11503, and 11504 (p. 45).

Mine.—No. 1, a shaft mine in sec. 18, T. 47 N., R. 9 W., 2 miles southwest of Fulton, on a spur of the Chicago & Alton Railroad.

Coal bed.—Bevier. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 6 inches. The bed lies practically horizontal. Roof, soft shale mixed with clay, about 12 inches thick, above the shale is a limestone cap rock 14 feet thick; floor, hard fire clay. Cover, 90 to 110 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 20, 1910.

Sample 11501 was taken at the face, 1,000 feet east of the shaft.

Sample 11502 was taken at the face, 600 feet northeast of the shaft.

Sample 11503 was taken at the face, 600 feet southwest of the shaft.

Each sample represents 2 feet 6 inches of coal, the thickness of the bed.

A composite sample was made by combining face samples 11501, 11502, and 11503. The results of an ultimate analysis of this sample are given under laboratory No. 11504.

Notes.—The mine is opened by a shaft 92 feet deep and is worked by the longwall system. At the time of sampling the output was 40 tons a day; most of the coal was used locally.

HARRISON COUNTY.

CAINESVILLE. CAINESVILLE MINE.

Sample.—Bituminous coal; Cainesville field; analyses Nos. 12679, 12680, and 12681 (p. 45).

Mine.—Cainesville, a shaft mine in sec. 13, T. 65 N., R. 26 W., at Cainesville, on the Chicago, Burlington & Quincy Railroad.

Coal bed.—Cainesville. Carboniferous age, Pennsylvanian series, Cherokee shale. Roof, drab to black shale; floor, hard sandy clay. Thickness, 3 feet 8 inches to 5 feet; average, 4 feet. Cover, 480 feet.

The bed was measured and sampled at three points in the mine by Henry Hinds on September 26, 1911.

Sample 12679 was taken in the main west air course, 600 feet west and 50 feet north of the shaft, and represents 3 feet 10½ inches of coal.

Sample 12680 was taken in south entry 2, off the main east entry, 150 feet east and 350 feet south of the shaft, and represents 3 feet 9½ inches of coal.

Sample 12681 was taken in south entry 2, off the main west entry, 400 feet southwest of the shaft, and represents 3 feet 9½ inches of coal. Each sample represents the entire thickness of the bed.

MELBOURNE. TRENTON NO. 1 MINE.

Sample.—Bituminous coal; analyses Nos. 11374, 11375, 11376, and 11377 (p. 45).

Mine.—Trenton No. 1, a shaft mine in sec. 25, T. 62 N., R. 26 W., one-fourth mile east of Melbourne, near the county line, and on the Quincy, Omaha & Kansas City Railroad.

Coal bed.—Lexington. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 3 feet. The bed lies practically horizontal. Roof, shale 8 inches thick, overlain by calcareous shale 2 feet thick. Floor, hard clay. Cover, 138 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 7, 1910.

Sample 11374 was taken at the face, 90 feet south of the shaft.

Sample 11375 was taken at the face, 85 feet northeast of the shaft.

Sample 11376 was taken at the face, 30 feet north of the shaft.

At each of the points sampled the bed was 3 feet thick. There was a 2-inch band of clay and "sulphur" 1 foot below the top of the coal. This was excluded in sampling the coal. There was shale above the coal. Below the coal was soft clay about 3 inches thick, underlain by hard clay.

A composite sample was made by combining face samples 11374, 11375, and 11376. The results of an ultimate analysis of this sample are given under laboratory No. 11377.

Notes.—The mine is opened by a shaft about 140 feet deep and is worked by the longwall system. At the time of sampling the output was 25 tons a day.

HENRY COUNTY.

CALHOUN. PARKS NO. 1 MINE.

Sample.—Bituminous coal; Lewis field; analyses Nos. 11314, 11315, 11316, and 11317 (p. 46).

Mine.—Parks No. 1, a drift mine in sec. 25, T. 34 N., R. 26 W., 2½ miles west of Calhoun, on the Missouri, Kansas & Texas Railway.

Coal bed.—Tebo. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 8 inches. The bed lies horizontal. Roof, shale 3½ feet thick, overlain by limestone 20 inches thick. Floor, soft fire clay. Cover, about 15 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 30, 1910.

Sample 11314 was taken at the face, 700 feet southeast of the mine mouth.

Sample 11315 was taken at the face, 700 feet southwest of the mine mouth.

Sample 11316 was taken at the face, 800 feet west of the mine mouth.

At each of the points sampled the bed measured 2 feet 8 inches. There was a "sulphur" band 1 inch thick about 1 foot 4 inches from the top of the coal. This was excluded in sampling.

A composite sample was made by combining face samples 11314, 11315, and 11316. The results of an ultimate analysis of this sample are given under laboratory No. 11317.

Notes.—The coal is mined by the longwall system. At the time of sampling the output was 35 tons a day.

CLINTON. LANE NO. 1 MINE.

Sample.—Bituminous coal; Lewis field; analyses Nos. 11248, 11249, 11250, and 11251 (p. 46).

Mine.—Lane No. 1, a slope mine in sec. 28, T. 42 N., R. 26 W., 3½ miles northwest of Clinton, with connections to the St. Louis & San Francisco Railroad.

Coal bed.—Tebo. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 6 inches. The bed dips slightly. Roof, calcareous shale; floor, clay. Cover, 18 to 20 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 25, 1910.

Sample 11248 was taken at the face, 210 feet northwest of the mine mouth, and represents 2 feet 6 inches of clean coal.

Sample 11249 was taken at the face, 300 feet northeast of the mine mouth, and represents 2 feet 6 inches of clean coal.

Sample 11250 was taken at the face, 230 feet north of the mine mouth, and represents 2 feet 6 inches of clean coal.

At each of the points sampled the entire thickness of the bed was included.

A composite sample was made from face samples 11248, 11249, and 11250. The results of an ultimate analysis of this sample are given under laboratory No. 11251.

Notes.—The mine is worked by the longwall system. At the time of sampling the daily output was 20 tons.

CLINTON. NEIL NO. 1 MINE.

Sample.—Bituminous coal; analyses Nos. 11302, 11303, 11304, and 11305 (p. 46).

Mine.—Neil No. 1, a shaft mine in sec. 34, T. 42 N., R. 26 W., 1 mile north of Clinton, on the St. Louis & San Francisco Railroad.

Coal bed.—Not correlated. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 6 inches. The bed lies practically horizontal. Roof, siliceous limestone 6 feet thick; floor, clay. Cover at points of sampling, 25 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 1, 1910.

Sample 11302 was taken at the face, 200 feet southwest of the shaft.

Sample 11303 was taken at the face, 100 feet northwest of the shaft.

Sample 11304 was taken 75 feet east of the shaft.

At each of the points sampled the bed was 2 feet 6 inches thick, with a 2-inch "sulphur" band 10 inches from the top. This was excluded from the samples, which represent 2 feet 4 inches of coal.

A composite sample was made by combining face samples 11302, 11303, and 11304. The results of an ultimate analysis of this sample are given under laboratory No. 11305.

Notes.—The mine is opened by a shaft about 30 feet deep and is worked by the longwall method. At the time of sampling the output was 12 tons a day. The coal was used for local purposes.

CLINTON. PHARIS NO. 1 MINE.

Sample.—Bituminous coal; Jordan field; analyses Nos. 11370, 11371, 11372, and 11373 (p. 47).

Mine.—Pharis No. 1, a shaft mine in sec. 7, T. 41 N., R. 26 W., 2 miles east of Clinton, on the St. Louis & San Francisco Railroad.

Coal bed.—Jordan. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 1 foot 10 inches. The bed lies practically horizontal. Roof, blue shale 2½ feet thick, overlain by calcareous shale 10 feet thick; floor, fire clay. Cover at point of sampling, 30 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 1, 1910.

Sample 11370 was taken at the face, 100 feet west of the shaft.

Sample 11371 was measured at the face, 100 feet south of the shaft.

Sample 11372 was measured at the face, 50 feet east of the shaft.

At each of the points sampled the bed was 1 foot 10 inches thick, with a 2-inch "sulphur" band about 8 inches from the top of the bed. This was excluded in sampling, each sample representing 1 foot 8 inches of coal.

A composite sample was made by combining face samples 11370, 11371, and 11372. The results of an ultimate analysis of this sample are given under laboratory No. 11373.

Notes.—The mine is opened by a shaft about 35 feet deep, and is worked by the room-and-pillar system. The daily output at the time of sampling was 14 tons.

CLINTON. SHELDON NO. 1 MINE.

Sample.—Bituminous coal, Jordan field; analyses Nos. 11262, 11263, 11264, and 11265 (p. 47).

Mine.—Sheldon No. 1, a shaft mine in sec. 25, T. 41 N., R. 26 W., 3 miles southeast of Clinton, on the St. Louis & San Francisco Railroad and the Kansas City Southern Railway.

Coal bed.—Jordan. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 2 inches. The bed lies practically horizontal. Roof, shale, 7 feet thick; floor, fire clay. Cover at points of sampling, 35 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 23, 1910.

Sample 11262 was taken at the face, 80 feet southwest of the shaft, and represents 2 feet 2 inches of clean coal.

Sample 11263 was taken at the face, 110 feet south of the shaft, and represents 2 feet 2 inches of clean coal.

Sample 11264 was taken at the face, 110 feet southeast of the shaft, and represents 2 feet 2 inches of clean coal.

At each of the points sampled the entire thickness of the bed was included.

A composite sample was made from face samples 11262, 11263, and 11264. The results of an ultimate analysis of this sample are given under laboratory No. 11265.

Notes.—The shaft is about 40 feet deep. The coal is mined by the longwall system. At the time of sampling the output was 20 tons a day.

DEEPWATER. DICKEY NO. 1 PIT.

Sample.—Bituminous coal, Jordan field; analyses Nos. 11310, 11311, 11312, and 11313 (p. 47).

Mine.—Dickey No. 1, an open-pit mine in sec. 14, T. 40 N., R. 26 W., in the eastern part of the city of Deepwater, on the Kansas City, Clinton & Springfield Railway.

Coal bed.—Jordan. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 26 inches. The bed is horizontal. Roof, shale, 2 to 4 feet thick, overlain by 15 feet of clay and soil; floor, fire clay.

The bed was measured and sampled at three points in the pit by R. D. Killian on December 1, 1910.

Sample 11310 was taken at the face on the north side of the pit.

Sample 11311 was taken at the face on the east side of the pit.

Sample 11312 was taken at the face on the west side of the pit.

At each of the points sampled the bed was 2 feet 2 inches thick, with a 2-inch "sulphur" band about 8 inches from the top. This band was excluded from the samples, which represent 2 feet of coal.

A composite sample was made from face samples 11310, 11311, and 11312. The results of an ultimate analysis of this sample are given under laboratory No. 11313.

Notes.—The mine is worked as an open pit. At the time of sampling the output was 10 tons a day, for local use.

DEEPWATER. HURST NO. 1 MINE.

Sample.—Bituminous coal, Jordan field; analyses Nos. 11252, 11253, 11254, and 11255 (p. 47).

Mine.—Hurst No. 1, a shaft mine in sec. 13, T. 40 N., R. 26 W., three-fourths of a mile east of Deepwater, with connections to the St. Louis & San Francisco Railroad.

Coal bed.—Jordan. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 3 feet 4 inches. The bed lies practically horizontal. Roof, dark gray shale; floor, soft clay. Cover, 54 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 26, 1910.

Sample 11252 was taken at the face, 100 feet northeast of the shaft, and represents 3 feet 4 inches of clean coal.

Sample 11253 was taken at the face, 350 feet west of the shaft, and represents 3 feet 4 inches of clean coal.

Sample 11254 was taken at the face, 450 feet northwest of the shaft, and represents 3 feet 4 inches of clean coal.

At each of the points sampled the entire thickness of the bed was included.

A composite sample was made by combining face samples 11252, 11253, and 11254. The results of an ultimate analysis of this sample are given under laboratory No. 11255.

Notes.—The shaft is 60 feet deep. The room-and-pillar system of mining is employed. The coal is delivered to the railroad by a tramway 1,100 feet long. At the time of sampling the output was 65 tons a day. The coal was marketed in the neighboring towns along the Frisco Railroad.

LEWIS. PIGG NO. 1 MINE.

Sample.—Bituminous coal, Lewis field; analyses Nos. 11267, 11268, 11269, and 11270 (p. 48).

Mine.—Pigg No. 1, a shaft mine in sec. 9, T. 42 N., R. 25 W., three-fourths of a mile north of Lewis, connected by a tramway to the Missouri, Kansas & Texas Railway.

Coal bed.—Tebo. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 6 inches. The bed lies practically horizontal. Roof, shale 3 feet thick, overlain by hard limestone 10 to 20 inches thick; floor, soft clay. Cover at points of sampling, 34 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian, on November 28, 1910.

Sample 11267 was taken at the face, 1,300 feet west of the shaft, and represents 2 feet 6 inches of clean coal.

Sample 11268 was taken at the face, 1,300 feet southwest of the shaft, and represents 2 feet 6 inches of coal.

Sample 11269 was taken at the face, 1,400 feet southwest of the shaft, and represents 2 feet 6 inches of coal.

At each of the points sampled the entire thickness of the bed was included.

A composite sample was made by combining face samples 11267, 11268, and 11269. The results of an ultimate analysis of this sample are given under laboratory No. 11270.

Notes.—The mine is opened by a shaft 50 feet deep and is worked by the longwall system. At the time of sampling, punching machines operated by compressed air were used for undercutting the coal. The daily output was 65 tons.

LINN COUNTY.

BROOKFIELD. CRANDALL NO. 1 MINE.

Sample.—Bituminous coal; Marceline field; analyses Nos. 11421, 11422, 11423, and 11424 (p. 48).

Mine.—Crandall No. 1, a shaft mine in sec. 20, T. 57 N., R. 19 W., 1¾ miles east of Brookfield near the Chicago, Burlington & Quincy Railroad.

Coal bed.—Tebo. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 2 inches. The bed lies horizontal. Roof, fire clay, 4 inches thick, overlain by blue shale 51 feet thick; floor, soft fire clay 3 inches thick, underlain by hard clay. Cover, 155 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian, on December 12, 1910.

Sample 11421 was taken at the face, 750 feet southeast of the shaft.

Sample 11422 was taken at the face, 750 feet southwest of the shaft.

Sample 11423 was taken at the face, 800 feet east of the shaft.

At each of the points sampled the bed was 2 feet 5 inches thick, with a 3-inch parting of soft fire clay, about 2 feet 2 inches from the top. Each sample represents 2 feet 2 inches of coal, the parting being excluded.

A composite sample was made by combining face samples 11421, 11422, and 11423. The results of an ultimate analysis of this sample are given under laboratory No. 11424.

Notes.—The shaft is 140 feet deep. The longwall system of mining is employed. At the time of sampling the daily output was 20 tons. The entire output was sold in and around Brookfield.

BROOKFIELD. WALKER NO. 1 MINE.

Sample.—Bituminous coal; Marceline field; analyses Nos. 11413, 11414, 11415, and 11416 (p. 49).

Mine.—Walker No. 1, a shaft mine in sec. 16, T. 57 N., R. 19 W., 3 miles southeast of Brookfield, on the Chicago, Burlington & Quincy Railroad.

Coal bed.—Tebo. Carboniferous age, Pennsylvanian series, Cherokee formation. Thickness, 2 feet 2 inches. The bed lies practically horizontal. Roof, blue shale; floor, soft clay 4 inches thick, underlain by hard clay. Cover at points of sampling, 170 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 12, 1910.

Sample 11413 was taken at the face, 250 feet southwest of the shaft.

Sample 11414 was taken at the face, 200 feet northeast of the shaft.

Sample 11415 was taken at the face, 200 feet southeast of the shaft.

At each of the points sampled the bed was 2 feet 2 inches thick, all of which is represented in the samples.

A composite sample was made by combining face samples 11413, 11414, and 11415. The results of an ultimate analysis of this sample are given under laboratory No. 11416.

Notes.—The shaft is about 175 feet deep. The mine is worked by the longwall method. At the time of sampling the daily output was 10 tons; it was used locally.

MARCELINE. LANDRETH NO. 1 MINE.

Sample.—Bituminous coal; Marceline field; analyses Nos. 11417, 11418, 11419, and 11420 (p. 49).

Mine.—Landreth No. 1, a shaft mine in sec. 20, T. 57 N., R 18 W., three-fourths of a mile west of Marceline, near the Atchison, Topeka & Santa Fe Railway.

Coal bed.—Tebo. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 4 inches. The bed lies practically horizontal. Roof, soft blue shale 31 feet thick; floor, clay. Cover, 130 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 13, 1910.

Sample 11417 was taken at the face, 600 feet east of the shaft, and represents 2 feet 4 inches of clean coal.

Sample 11418 was taken at the face, 600 feet north of the shaft, and represents 2 feet 4 inches of clean coal.

Sample 11419 was taken at the face, 600 feet south of the shaft, and represents 2 feet 4 inches of coal.

At each of the points sampled the entire thickness of the bed was included.

A composite sample was made by combining face samples 11417, 11418, and 11419. The results of an ultimate analysis of this sample are given under laboratory No. 11420.

Notes.—The mine is opened by a shaft about 130 feet deep, and is worked by the longwall system. At the time of sampling the daily output was 25 tons. The coal is hauled by wagon to Marceline, where it is used for domestic and steaming purposes.

PUTNAM COUNTY.

MENDOTA. MENDOTA NO. 2 MINE.

Sample.—Bituminous coal; Mendota field; analyses Nos. 11398, 11399, 11400, and 11401 (p. 49).

Mine.—Mendota No. 2, a shaft mine in sec. 1, T. 66 N., R. 18 W., 1 mile southeast of Mendota on the Chicago, Burlington & Quincy Railroad.

Coal bed.—Lexington. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 3 feet. The bed lies horizontal. Roof, shale 18 inches thick, overlain by calcareous shale 16 feet thick; floor, hard fire clay. Cover, 65 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 10, 1910.

Sample 11398 was taken at the face, 2,500 feet southwest of the shaft.

Sample 11399 was taken at the face, 3,200 feet south of the shaft.

Sample 11400 was taken at the face, 3,100 feet southeast of the shaft.

At each of the places sampled the coal was 3 feet thick, with a 2-inch clay parting about the middle of the bed. The parting was excluded from the samples.

A composite sample was made from face samples 11398, 11399, and 11400. The results of an ultimate analysis of this sample are given under laboratory No. 11401.

Notes.—The shaft is 67 feet deep. A combination of the room-and-pillar and the longwall systems of mining are used. At the time of sampling the output was 100 tons a day. The coal was used for domestic fuel and steaming purposes.

UNIONVILLE. ANDERSON MINE.

Sample.—Bituminous coal; Mendota field; analyses Nos. 14880 and 14881 (p. 49).

Mine.—Anderson, a shaft mine, Unionville district; NW. ¼ sec. 30, T. 66 N., R. 19 W.; 2½ miles northeast of Unionville.

Coal bed.—Lexington. Carboniferous age; Pennsylvanian series; Cherokee shale. Roof, black "slaty" shale, overlain by limestone; floor, clay. Thickness, 32 inches, fairly uniform.

The bed was measured and sampled at two points in the mine by M. Albertson on October 10, 1912, as described below·

Sections of coal bed in Anderson mine.

Section	A	B
Laboratory No.	14880	14881
	Ft. in.	*Ft. in.*
Coal	1 7	1 6
Clay	[a]0 2¼	[a]0 2¼
Coal	0 11	0 11
Clay	.. ..	[a]0 1½
Coal	.. ..	0 1½
Thickness of bed	2 8¼	2 10¼
Thickness of coal sampled	2 6	2 6½

[a] Excluded from sample.

Section A (sample 14880) was taken 175 feet southwest of the shaft.

Section B (sample 14881) was taken 150 feet east of the shaft.

RANDOLPH COUNTY.

HUNTSVILLE. CARSON NO. 1 MINE.

Sample.—Bituminous coal, Bevier field; analyses Nos. 11452, 11453, 11454, and 11455 (p. 50).

Mine.—Carson No. 1, a shaft mine in the Huntsville district; sec. 36, T. 54 N., R. 15 W.; in the southern part of Huntsville, on the Wabash Railroad.

Coal bed.—Bevier. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 4 feet. The bed lies horizontal. Roof, blue calcareous shale 20 feet thick; floor, hard fire clay. Cover at points of sampling, 40 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 14, 1910.

Sample 11452 was taken at the face, 250 feet southeast of the shaft, and represents 4 feet of clear coal, the thickness of the bed.

Sample 11453 was taken at the face, 300 feet north of the shaft, and represents 4 feet of clear coal, the thickness of the bed.

Sample 11454 was taken at the face, 200 feet southwest of the shaft, and represents 4 feet of clear coal, the thickness of the bed.

A composite sample was made by combining face samples 11452, 11453, and 11454. The results of an ultimate analysis of this sample are given under laboratory No. 11455.

Notes.—The mine is opened by a shaft about 40 feet deep; it is worked by the room-and-pillar system. At the time of sampling the daily output was 10 tons. The coal is sold in the local markets at Huntsville.

HUNTSVILLE. NORTHERN NO. 2 MINE.

Sample.—Bituminous coal; Bevier field; analyses Nos. 11448, 11449, 11450, and 11451 (p. 50).

Mine.—Northern No. 2, a shaft mine in the Huntsville district; SE. ¼ sec. 36, T. 54 N., R. 15 W.; three-fourths of a mile south of Huntsville, on the Wabash Railroad.

Coal bed.—Bevier. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 4 feet. The bed lies horizontal. Roof, blue calcareous shale 30 feet thick; floor, hard fire clay. Cover at points of sampling 70 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 14, 1910.

Sample 11448 was taken at the face, 4,000 feet south of the shaft, and represents 4 feet of clear coal.

Sample 11449 was taken at the face, 3,800 feet northeast of the shaft, and represents 4 feet of clear coal.

Sample 11450 was taken at the face, 3,500 feet southeast of the shaft, and represents 4 feet of clear coal.

A composite sample was made by combining face samples 11448, 11449, and 11450. The results of an ultimate analysis of this sample are given under laboratory No. 11451.

Notes.—The shaft is about 70 feet deep; the room-and-pillar system of mining is used. At the time of sampling the output was 600 tons a day.

RENICK. ORRIS NO. 1 MINE.

Sample.—Bituminous coal; Bevier field; analyses Nos. 11497, 11498, 11499, and 11500 (pp. 50, 51).

Mine.—Orris No. 1, a shaft mine in the Higbee district; sec. 31, T. 53 N., R. 13 W., one-quarter mile west of Renick, on the Wabash Railroad.

Coal bed.—Mulky. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness 1 foot 6 inches. The bed lies horizontal. Roof, shale 4 feet thick, overlain by limestone 6 feet thick; floor, fire clay 1 foot 6 inches thick. Cover at points of sampling, 60 feet.

The bed was measured and sampled at three places in the mine by R. D. Killian on December 15, 1910.

Sample 11497 was taken at the face, 300 feet southwest of the shaft.

Sample 11498, was taken at the face, 300 feet northwest of the shaft.

Sample 11499 was taken at the face, 300 feet west of the shaft.

At each of the points sampled the bed was 1 foot 6 inches thick, all of which was included in sampling.

A composite sample was made by mixing face samples 11497, 11498, and 11499. The results of an ultimate analysis of this sample are given under laboratory No. 11500.

Notes.—The shaft is about 60 feet deep; the room-and-pillar system of mining is employed. At the time of sampling the daily output was 10 tons; the coal was sold in local markets.

RAY COUNTY.

VIBBARD. VIBBARD NO. 1 MINE.

Sample.—Bituminous coal; Lexington field; analyses Nos. 11365, 11366, 11367, and 11368 (p. 51).

Mine.—Vibbard No. 1, a shaft mine in the Richmond district; sec. 27, T. 53 N., R. 29 W.; one-fourth of a mile south of Vibbard, on the Atchison, Topeka & Santa Fe Railway.

Coal bed.—Lexington. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 2 feet 2 inches. The bed lies horizontal. Roof, "slate" 8 inches thick, overlain by calcareous shale 7 feet thick; floor, clay. Cover at points of sampling, 400 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on December 5, 1910.

Sample 11365 was taken at the face, 250 feet south of the shaft.

Sample 11366 was taken at the face, 250 feet east of the shaft.

Sample 11367 was taken at the face, 200 feet west of the shaft.

At each of the points sampled the bed was 2 feet 2 inches thick, with a 2-inch "sulphur" band 8 inches from the top. Each sample represents 2 feet of coal, the 2-inch band being excluded.

A composite sample was made by combining face samples 11365, 11366, and 11367. The results of an ultimate analysis of this sample are given under laboratory No. 11368.

Notes.—The mine is worked by the longwall system; at the time of sampling the output was 12 tons a day. The coal was sold in the local market.

VERNON COUNTY.

MOUNDVILLE. BROWN No. 1 MINE.

Sample.—Bituminous coal; analyses Nos. 11201, 11202, 11203, and 11204 (p. 51).

Mine.—Brown No. 1, a shaft mine in sec. 5, T. 34 N., R. 32 W., at the northwest corner of Moundville, on the Missouri Pacific Railway.

Coal bed.—Not correlated. Carboniferous age, Pennsylvanian series. Thickness, 5 feet 10 inches. The bed lies horizontal. Roof, dark bluish shale 8 feet thick; floor, hard shale. Cover at points of sampling, 28 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 15, 1910.

Sample 11201 was taken at the face, 40 feet northwest of the shaft.

Sample 11202 was taken at the face, 100 feet northeast of the shaft.

Sample 11203 was taken at the face, 125 feet north of the shaft.

Each sample represents 2 feet 2 inches of coal. A lower bench of coal, 2 feet 4 inches thick, was separated from the one sampled by 1 foot 4 inches of shale.

A composite sample was made by combining face samples 11201, 11202, and 11203. The results of an ultimate analysis of this sample are given under laboratory No. 11204.

Notes.—The shaft is 26 feet deep; the mine is worked by room-and-pillar system. At the time of sampling the daily output was 40 tons.

PANAMA. JONES No. 1 MINE.

Sample.—Bituminous coal; Rich Hill field; analyses Nos. 11205, 11206, 11207, and 11208 (p 51).

Mine.—Jones No. 1, a shaft mine in sec. 4, T. 37 N., R. 31 W., 2 miles south of Panama, on the Missouri Pacific Railway.

Coal bed.—Rich Hill. Carboniferous age, Pennsylvanian series, Cherokee shale. Thickness, 5 feet. The bed lies practically horizontal. Roof, shale 1 foot thick; floor, soft shale. Cover 40 to 60 feet.

The bed was measured and sampled at three points in the mine by R. D. Killian on November 16, 1910.

Sample 11205 was taken at the face, 450 feet east of the shaft.

Sample 11206 was taken at the face, 1,000 feet southeast of the shaft.

Sample 11207 was taken at the face, 100 feet east of the shaft.

At each of the points sampled the coal bed was 5 feet thick. Above the coal was a "sulphur" band 5 inches thick, which was overlain by the shale roof. There was shale below the coal.

A composite sample was made by combining face samples 11205, 11206, and 11207. The results of an ultimate analysis of this sample are shown under laboratory No. 11208.

Notes.—The shaft is 27 feet deep; the mine is worked by the room-and-pillar system. At the time of sampling the coal was blasted off the solid; the daily output was 100 tons.

MONTANA.

BLAINE COUNTY.

CLEVELAND. COOK MINE.

Sample.—Subbituminous coal; Cleveland field; analysis No. 14782 (p. 52).

Mine.—Cook; SW. ¼ sec. 35, T. 30 N., R. 20 E., about 4 miles northeast of Cleveland post office.

Coal bed.—Not named; Judith River formation; Cretaceous age. The bed was measured and sampled at one point in the mine by C. F. Bowen and Harvey Bassler on September 22, 1912, as shown below:

Section of coal bed in Cook mine.

Laboratory No.	14782
Roof, shale.	*Ft. In.*
Coal	1 3
Clay	[a]0 5½
Coal	1 10
Floor, shale.	
Thickness of bed	3 6½
Thickness of coal sampled	3 1

[a] Not included in sample.

The sample was taken from the face of the entry, about 500 feet from the mine mouth.

Notes.—The mine was operated during the winter months and has an output of about 600 tons a year.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 541. (In press).

CARBON COUNTY.

BEAR CREEK. No. 2 MINE.

Sample.—Subbituminous coal; Red Lodge field; analyses Nos. 15145 and 15151; 15146 and 15152; 15147 and 15153; 15148 and 15154; 15149 and 15155; and 15150 and 15156. (pp. 52, 53).

Mine.—No. 2, a drift mine in sec. 6, T. 8 S., R. 21 E., 1½ miles west of Bear Creek, on the Montana, Wyoming & Southern Railroad.

Coal bed.—No. 2, Tertiary age, Fort Union formation. Average thickness, 6 feet 6 inches; dip, about 5° southwest, not uniform. Roof, dark shale 4 to 10 feet thick, overlain by sandstone; floor, soft smooth fire clay. Particles of the roof and floor did not mix with the coal in mining. Cover at the points of sampling, 85 to 105 feet.

The bed was measured and sampled at five points in the mine by O. U. Bradley on November 28 to 30, 1912, as described below:

Sections of coal bed in No. 2 mine.

Section	A	B	C	D	E
Laboratory No.	15145 15151	15146 15152	15147 15153	15148 15154	15149 15155
Roof, clay (overlain by shale).	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	0 4	0 6	0 5	0 7	0 5
Coal, bony	0 3	0 ½	0 3	0 3½	0 2½
Coal	2 7	2 8	4 10½	2 8	2 5
Clay and bony coal	0 1	0 ½	.. ..	0 ¼	0 ½
Coal	0 3	.. ..	.. ..	0 3¾	0 3
Clay or bony coal	0 ½	.. ..	.. ..	0 ¼	0 ½
Coal	2 8	3 1	.. ..	2 5	2 4½
Floor, fire clay.					
Thickness of bed	6 2½	6 4	5 6½	6 3¾	5 9
Thickness of coal sampled	6 2½	6 4	5 6½	6 3¾	5 9

Section A (samples 15145 and 15151) was measured on a pillar between rooms 9 and 10, off north entry 2.

Section B (samples 15146 and 15152) was measured at the face of room 10; 125 feet off west entry 1.

Section C (samples 15147 and 15153) was measured at the face of room 9, 45 feet off east entry 9.

Section D (samples 15148 and 15154) was measured at the face of the cut-through in room 22, off east entry 4.

Section E (samples 15149 and 15155) was measured at the face of room 11, 35 feet off rock tunnel entry.

A composite sample was made by mixing samples 15146, 15147, 15148, and 15149. The results of an ultimate analysis of this sample are shown under laboratory No. 15150.

A composite sample was made by mixing samples Nos. 15152, 15153, 15154, and 15155. The results of an ultimate analysis of this sample are shown under laboratory No. 15156.

Notes.—The coal is mined by the room-and-pillar system. At the time of sampling, the coal was undercut by chain cutting machines and broken down with FF black blasting powder. None of the coal was shipped as run-of-mine. The tipple was equipped with shaking screens 12 feet long and 5 feet wide with openings 2½ by 4 inches, screens 5 feet wide and 12 feet long, with 2-inch openings, and a revolving screen 12 feet long with ¾ and 1¼ inch openings. About 50 per cent of the coal passed through the larger screens. The screenings were conveyed by an aerial tramway to the dirt dump. There were no pickers employed. The lumps were large and appeared to be clean coal. There were two loading tracks with a capacity of 20 empty and 20 loaded cars. The daily output of the mine was about 800 tons. The intent was to increase the output to 1,000 or 1,200 tons when the No. 3 seam is opened.

BEAR CREEK. NORTH SIDE AND SOUTH SIDE MINES.

Sample.—Subbituminous coal; Red Lodge field; analyses Nos. 15127 and 15133, 15128 and 15132, 15129 and 15131, 15143 and 15144, and 15130 and 15134 (p. 53).

Mine.—The North Side and South Side mines are opened by drifts in sections 5 and 6, T. 8 S., R. 21 E. The tipple is 1 mile west of Bear Creek, on the Montana, Wyoming & Southern Railroad.

Coal bed.—No. 3, Tertiary age, Fort Union formation. Average thickness, 6 feet 3 inches; dip, about 4° southwest. The cleat is poorly defined. The coal bed has two irregular bone partings. The roof is a dark blue shale which occasionally comes down with the coal; above the shale is a sandstone cap rock. The floor is a soft smooth fire clay. There is a workable coal bed 80 feet above and one 60 feet below this bed. At the time of sampling, particles of the roof and floor mixed with the coal in mining. Cover at the points of sampling, 75 to 130 feet.

The bed was measured and sampled at three points in the South Side mine on November 25, 1912, and at one point in the North Side mine on December 1, 1912, by O. U. Bradley, as described below:

Sections of coal bed in South Side mine.

Section	A		B		C	
Laboratory No.	15127 15133		15128 15132		15129 15131	
Roof, blue shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	5	6	5	7	5	1
Clay and bone	[a]0	8	[a]0	4	[a]0	5
Coal	0	7	..	..	0	7
Bone	[a]0	4	..	..	[a]0	2
Coal	0	10	1	9	0	11
Floor, clay.						
Thickness of bed	7	11	7	8	7	2
Thickness of coal sampled	6	11	7	4	6	7

[a] Not included in sample.

Section A (samples 15127 and 15133) was measured at the face of room 2, off north entry 1, 50 feet in from entry.

Section B (samples 15128 and 15132) was measured in the crosscut from the main entry to the back entry, 350 feet from the mouth of the main entry and 75 feet from the face of the back entry.

Section C (samples 15129 and 15131) was measured at the face of the first south back entry, 150 feet from the main slope.

A composite sample was made by mixing samples 15127, 15128, and 15129. The results of an ultimate analysis of this sample are given under laboratory No. 15130.

A composite sample was made by mixing samples 15131, 15132, and 15133. The results of an ultimate analysis of this sample are given under laboratory No. 15134.

Section of coal bed in North Side mine.

Laboratory No.	15143 15144
	Ft. in.
Roof, blue shale.	
Coal	1 11
Clay and bone	0 1
Coal	0 6
Bone	0 1
Coal	2 11
Floor, clay.	
Thickness of bed	5 6
Thickness of coal sampled	5 6

The above section (samples 15143 and 15144) was measured on the last pillar on the second west main entry, 500 feet from the drift mouth.

Notes.—At the time of sampling the North Side mine was almost worked out; advance work being completed. It was estimated that all the pillars would be withdrawn within one and one-half years. Twenty-seven inches or more of bottom coal had been left in the mine.

The South Side mine was a new mine on the opposite side of the gulch from the North Side mine. The coal was delivered to the North Side tipple over a wooden trestle 1,600 feet long, with a 4 per cent grade. At the time of sampling all the workings in the South Side mine were narrow. The coal was cut in the 4 to 8-inch parting with machines and the top coal mined three cuts ahead of the bottom coal. This was the first mine in the district to mine the bottom coal of the No. 3 bed; the lower bench averages 27 inches in thickness. The coal was broken down with FF black blasting powder. Dynamite was sometimes used for brushing the roof and floor.

The screening equipment consisted of shaking screens 16 feet long and 6 feet wide, in three sections: an 8-foot section with 2-inch openings, a 4-foot section with 4-inch openings, and a 4-foot section with 6-inch openings; a small shaking screen 14 feet long and 6 feet wide, in two sections: an 8-foot section with 1¼-inch openings and a 6-foot section with 1½-inch openings; and a revolving screen 12 feet long with ⅜-inch openings. Four per cent of the coal was shipped as run-of-mine. Sixty-five per cent of the coal going to the screens passed through the 6-inch openings. Pickers were employed on picking tables. The lumps were large and had a good appearance. There were three loading tracks with a capacity of 25 empty and 25 loaded cars. The daily output was 300 tons. There was about 600 acres of unmined coal tributary to the tipple.

CHOUTEAU COUNTY.

BIG SANDY. MACKTON MINE.

Sample.—Subbituminous coal; Big Sandy field; analyses Nos. 14613 and 14615 (p. 54).

Mine.—Mackton; NW. ¼ SW. ¼ sec. 18, T. 28 N., R. 14 E., 6 miles northeast of Big Sandy.

Coal bed.—Mackton. Tertiary age, Fort Union formation. Thickness about 7½ feet; dip 40° east.

The bed was measured and sampled at two points in the mine by C. F. Bowen and Harvey Bassler on August 12, 1912, as shown below:

Sections of coal bed in Mackton mine..

Section	A	B
Laboratory No.	14613	14615
Roof, sandstone.	*Ft. in.*	*Ft. in.*
Coal	2 5	1 5
Bone	[a]0 2	[a]0 2
Coal	0 6	1 3
Bone and coal	[a]1 9	[a]1 3
Coal	2 7	3 5
Floor, sandstone.		
Thickness of bed	7 5	7 6
Thickness of coal sampled	5 6	6 1

[a] Not included in sample.

Section A (sample 14613) was measured at the face of the north entry on the second level, 335 feet below the mouth of the slope and 140 feet from the mouth of the entry.

Section B (sample 14615) was measured at the face of the south entry on the first level, 150 feet below the mouth of the slope and 250 feet south of the mouth of the entry.

Sample 14613 represents perfectly fresh coal. Sample 14615 represents coal on which no mining had been done for nearly a year and was probably somewhat weathered.

Notes.—The dip and thickness of the bed are fairly uniform in the mine. The output was about 100 tons a day during the winter months but was less in the summer season.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 128, 595.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 541, 1914. (In press.)

CUSTER COUNTY.

TERRY. CAMERON MINE.

Sample.—Lignite; Terry field; analysis No. 10677 (p. 54).

Mine.—Cameron, an open-pit mine in sec. 24, T. 12 N., R. 50 E., about 4 miles west of Terry, which is the nearest station on the Northern Pacific Railway and the Chicago, Milwaukee & St. Paul Railway. The mine is about one-fourth of a mile south of the railroads.

Lignite bed.—Unnamed. Tertiary age, Fort Union formation. Thickness, 7 feet 5 inches. Roof, soft white sandstone; floor, shale. The bed was measured and sampled at one point in the pit by F. A. Herald on July 15, 1910, as described below:

Section of lignite bed in Cameron mine.

Laboratory No.	10677
Roof, soft sandstone.	*Ft. in.*
Lignite	[a]0 8
Parting	[a]0 8
Lignite	[a]3 4
Lignite	3 5
Floor, shale.	
Thickness of bed	8 1
Thickness of coal sampled	3 5

[a] Not included in sample.

The 3 foot 4 inch bench above the part sampled was apparently as good lignite as the lower part of the bed, although it was reported at the mine that it burns quickly and does not give out as much heat. At the time of sampling the mine was worked as an open pit and the output was only one or two wagon loads a day.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 246.

WESTMORE. PROSPECT.

Sample.—Lignite; Baker field; analysis No. 10910 (p. 54).

Mine.—Prospect opening in sec. 34, T. 6 N., R. 57 E., about 17 miles south of Westmore, which is the nearest railroad station on the Chicago, Milwaukee & St. Paul Railway.

Lignite bed.—Unnamed. Tertiary age, Fort Union formation. Thickness, 4 feet 3 inches. Above the lignite is 2 inches of bituminous shale, overlain by a top clay. On account of the excessive amount of water in the opening the character of the floor could not be determined. The bed was sampled by C. F. Bowen on September 13, 1910. At the time this sample was taken the prospect was not being worked.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 471, 1912, p. 220.

DAWSON COUNTY.

GLENDIVE. PROSPECT.

Sample.—Lignite; Sidney field; analysis No. 10914 (p. 54).

Mine.—Prospect in SW. ¼ sec. 31, T. 18 N., R. 57 E., 15 miles northeast of Glendive, which is the nearest station on the Northern Pacific Railway.

Lignite bed.—B. Tertiary age, Fort Union formation. Thickness, 5 feet. Roof, white clay; floor, grayish clay. The bed was sampled by Eugene Stebinger on July 25, 1910. The sample was taken on the right wall, 25 feet from the outcrop. The entry was caved and evidently had not been worked for some years.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull, 471, 1912, p. 297.

GLENDIVE. SNYDER MINE.

Sample.—Lignite; Glendive field; analysis No. 11045 (p. 54).

Mine.—Snyder, NW ¼ sec. 27, T. 17 N., R. 55 E., about 6½ miles north of Glendive, which is the nearest station on the Northern Pacific Railway. The mine is about 6 miles from the railroad.

Lignite bed.—Second or middle bed, not named. Tertiary age, Fort Union formation. Thickness, about 8 feet. The bed was sampled at one point in the mine by J. H. Hance on October 4, 1910. The sample was taken at the working face, about 80 feet from the mine mouth.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 130, 610.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 471, 1912, p. 278.

FERGUS COUNTY.

HILGER. STONE MINE.

Sample.—Subbituminous coal, Lewistown field; analysis No. 14495 (p. 54).

Mine.—Stone; sec. 8, T. 19 N., R. 19 E., about 13 miles north of Hilger.

Coal bed.—Not named. Cretaceous age, Judith River formation. The bed was measured and sampled at one point in the mine by C. F. Bowen and Harvey Bassler on July 26, 1912, as shown below:

Section of coal bed in Stone mine.

Laboratory No.	14495	
Roof, shale.	*Ft.*	*in.*
Coal	2	0
Clay	[a]0	7
Coal	0	8
Shale	[a]0	4
Coal	0	6
Shale	[a]0	2
Coal	0	9
Sandstone	[a]0	2
Bone	[a]0	5
Coal	0	5+
Floor, not stated.		
Thickness of bed	6+	0
Thickness of coal sampled	4	4

[a] Not included in sample.

The sample was taken at the face of the entry, 140 feet from the mine mouth and represents fresh coal.

Notes.—At the time of sampling the mine was not a regular producer. The coal was used chiefly as fuel for traction engines.

For a description of the geologic relations of this bed see U. S. Geol. Survey Bull. 541.

FLATHEAD COUNTY.

FLATHEAD RIVER. OUTCROP.

Sample.—Subbituminous coal; Glacier Park field; analysis No. 12786 (p. 55).

Location.—Natural outcrop in bank of North Fork of Flathead River, in sec. 33, T. 34 N., R. 20 W.

Coal bed.—Not named. Tertiary age. The bed was measured and sampled below water level by M. R. Campbell on October 2, 1911, as follows:

Section of coal bed in outcrop in sec. 33, T. 34 N., R. 20 W.

Laboratory No.	12786	
	Ft.	*in.*
Coal	2	6
Clay	[a]0	8
Coal	1	0
Clay	[a]0	2
Coal	[a]0	10
Clay	[a]0	8
Coal	[a]0	11
Thickness of bed	6	9
Thickness of coal sampled	3	6

[a] Not included in sample.

Notes.—The coal represented by the sample was weathered. At the time the sample was taken, no coal had been mined in this valley.

MISSOULA COUNTY.

MISSOULA. HELL GATE MINE.

Sample.—Subbituminous coal; analyses Nos. 13541 and 13542.

Mine.—Hell Gate, a slope mine in sec. 4, T. 13 N., R. 19 W., 2 miles north of Missoula.

Coal bed.—Unnamed; in Tertiary lake beds, probably the White River formation of Oligocene age.

Sample 13541 was taken by J. T. Pardee on September 30, 1911, from coal in the bunker at the mouth of the mine and represented run-of-mine coal from a bed 6 feet thick in the east entry.

Sample 13542, as reported, was taken by H. L. Shapard in the east entry, 300 feet from the slope, and represented 6 feet 10 inches of coal.

Notes.—At the time the mine was visited the coal was used for local purposes. It slacks on exposure and will not stand storage nor transportation to any considerable distance.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 531, p. 241.

MUSSELSHELL COUNTY.

PAINTED ROCK. SHOLZ MINE.

Sample.—Subbituminous coal; analysis No. 15114 (p. 55).

Mine.—Sholz, a drift mine near the Great Northern Railway, NW. ¼ sec. 14, T. 5. N., R. 22 E., near Painted Rock.

Coal bed.—Unnamed. Cretaceous age; Judith River formation; lenticular, maximum thickness 44 inches.

The bed was measured and sampled at one place in the mine by E. G. Woodruff on September 10, 1912.

The mine consisted of two drifts, a north one 110 feet long, and a south one 80 feet long. The sample was taken from a working face at the end of the longer drift, and represents 2 feet 11 inches of coal. There was shale interlaminated with coal above the part sampled, and sandy shale below it.

ROSEBUD COUNTY.

BIGHORN. PROSPECT.

Sample.—Subbituminous coal; Tullock Creek field; analysis No. 14755 (p. 55).

Location.—Prospect in the NW. ¼ NE. ¼ sec. 2, T. 4 N., R. 35 E., on Unknown Creek, 6½ miles east of Bighorn. No railroad connection.

Coal bed.—Not named. Cretaceous or Tertiary age; Lance formation. Character, variable and lenticular.

The bed was measured and sampled by C. E. Lesher and G. S. Rogers on September 18, 1912.

Section of coal bed in prospect, 6½ miles east of Bighorn.

Laboratory No.	14755	
	Ft.	*In.*
Roof, shale and thin sandstone.		
Coal	[a]0	1
Sandstone	[a]0	½
Shale	[a]0	1¾
Coal	0	8
Bone	[a]0	1¾
Coal	0	8
Coal, dirty	0	3
Coal, hard	1	2¼
Floor, shale.		
Thickness of bed	3	2¼
Thickness of coal sampled	2	9¼

[a] Not included in sample.

The sample was taken 10 feet from the surface at a freshly exposed face, but probably represented coal from the weathered part of the bed. No mining had been done at this prospect.

TETON COUNTY.

BROWNING. J. M. STONE PROSPECT.

Sample.—Bituminous coal; Blackfoot field; analysis No. 12602 (p. 55).

Mine.—J. M. Stone prospect, SE. ¼ sec. 35, T. 36 N., R. 12 W., on middle fork of Milk River, 22 miles northwest of Browning.

Coal bed.—The bed occurs at the top of a coarse gray sandstone formation of Upper Cretaceous age. Roof, clay-shale 2 feet thick, overlain by gray sandstone; floor, clay. Thickness, 2 feet 8 inches; dip, 40° northeast. The bed is in a folded and faulted belt of rocks. The coal is without a parting, and has a fine cubic grain, a graphite black color, and a jointed structure suggesting torsion due to folding. The bed was measured and sampled at one point in the mine by Eugene Stebinger on September 10, 1911.

Notes.—The sample was cut from the face of the entry, 50 feet in. The coal was probably slightly weathered. Mined only occasionally for domestic use.

CUT BANK. ALLISON PROSPECT.

Sample.—Bituminous coal; analysis No. 12494 (p. 55).

Mine.—Prospect of A. E. Allison, NE. ¼ sec. 29, T. 34 N., R. 6 W., on Cutbank Creek, 5 miles northwest of Cut Bank.

Coal bed.—An unnamed coal bed occurring 225 feet above the base of the Two Medicine formation, Upper Cretaceous age. Lies nearly flat, dipping only about 40 feet per mile westward. Roof and floor are both gray clay, the coal lays on a persistent massive gray sandstone bedrock 30 to 50 feet thick.

The coal is 1 foot 10 inches thick and is very dirty, containing occasional streaks of clean hard coal. The bed was measured and sampled by Eugene Stebinger on July 24, 1911.

Notes.—The sample represents partly weathered coal from the prospect entry, under 25 feet of cover. Attempts to sell this coal for local use were not successful.

CUT BANK. PROSPECT.

Sample.—Bituminous coal; analysis No. 14109 (p. 55).

Mine.—Prospect open cut; SW. ¼ sec. 26, T. 33 N., R. 6 W.; on bank of Spring Creek, 4 miles southwest of Cut Bank.

Coal bed.—An unnamed coal bed occurring 225 feet above the base of the Two Medicine formation, Upper Cretaceous age. Thickness, 11 inches. Bed lies almost flat, dipping only 40 feet to the mile to the west. The roof and floor are both of gray clay. The coal is clean, with well-marked joint planes. The bed was measured and sampled by Eugene Stebinger on June 13, 1912.

The sample was taken in a fresh open cut under 4 feet of cover, and represents partly weathered coal.

VALIER. BLAIR MINE.

Sample.—Bituminous coal; Valier field; analysis No. 12427 (p. 55).

Mine.—A. G. Blair, a drift mine in the NW. ¼ sec. 32, T. 31 N., R. 5 W., on Birch Creek, 8 miles northwest of Valier.

Coal bed.—The only workable bed occurring at the top of the Eagle sandstone, of Upper Cretaceous age. The bed lies nearly flat, dipping westward only 20 to 30 feet to the mile. Roof, dark gray clay-shale, containing a few lenses of massive gray sandstone lying directly on the coal; floor, clay. The bed was measured and sampled at one place in the mine by Eugene Stebinger on July 12, 1911, as described below.

Section of coal bed in Blair mine.

Laboratory No.	12427	
	Ft.	*in.*
Coal	0	5
Bone	[a]0	2
Coal	1	10
Thickness of bed	2	5
Thickness of coal sampled	2	3

[a] Not included in sample.

Notes.—The sample was taken at the face of a new entry in 150 feet, in freshly mined coal. The upper 5 inches was hard coal with shiny black faces and showing some whitish dirt. The bone parting was persistent and regular. The lower bench, 1 foot 10 inches thick, was hard and fairly clean, with prominent joint planes, causing the coal to break in large blocks. At the time of sampling the mine was worked intermittently, employing up to 10 men, supplying a local trade in the vicinity of Valier and competing successfully with coal from the Great Falls field and Canadian coal from Lethbridge, Alta.

VALLEY COUNTY.

ANTELOPE. RICHARDSON MINE.

Sample.—Lignite; Plentywood field; analysis No. 14614 (p. 55).

Mine.—Richardson; a drift mine in NW. ¼ NE. ¼ sec. 21, T. 34 N., R. 55 E., 3 miles west of Antelope and 12 miles southeast of Plentywood.

Lignite bed.—Designated as "Richardson bed" by the United States Geological Survey. The bed lies about 800 feet above the base of the Fort Union formation of Tertiary age. Quality and thickness irregular. Dip, about 3½ feet per thousand southeastward.

The bed was measured and sampled at one point in the mine by C. M. Bauer, on August 12, 1912, as described below:

Section of lignite bed in Richardson mine.

Laboratory No.	14614	
Roof, carbonaceous shale.	*Ft.*	*in.*
Lignite	3	1
Shale, blue	[a]0	5
Lignite	5	2
Floor, brown shale.		
Thickness of bed	8	8
Thickness of lignite sampled	8	3

[a] Not included in sample.

The sample was collected 200 feet north of the opening on the east side of the entry, under 75 feet of cover.

Notes.—The lignite of this mine is representative of lignite mined in this part of Montana.

For a description of the geologic relations of this lignite bed, see U. S. Geol. Survey Bull. 541.

BAINVILLE. RED BANK MINE.

Sample.—Lignite; Culbertson field; analysis No. 10898 (p. 55).

Mine.—Red Bank, an open-pit mine in sec. 10, T. 28 N., R. 59 E., 8 miles northeast of Bainville, which is the nearest station on the Great Northern Railway.

Lignite bed.—G, of the United States Geological Survey. Early Eocene age, Fort Union formation. Thickness, 5 feet 6 inches. The lignite is mined by stripping the clay and soil from the surface. The bed was sampled at one place in the pit by A. L. Beekly on September 1, 1910. The upper 5 feet of the bed was comparatively dry; the lower 6 inches was wet. The sample, which represents the full thickness of the bed, was taken at the face.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 347.

CULBERTSON. BRUEGGER MINE.

Sample.—Lignite; Culbertson field; analysis No. 10724. (p. 56.)

Mine.—Bruegger; sec. 8, T. 28 N., R. 56 E.; 3 miles north of Culbertson, on the Great Northern Railway.

Lignite bed.—E bed of the United States Geological Survey. Tertiary age, Fort Union formation. Thickness, 7 feet. Roof, a gray clay shale; floor, blue clay.

The lignite bed was measured and sampled at one point in the mine by A. L. Beekly on July 26, 1910. The sample was taken about 75 feet north of the mine opening and represents 7 feet of lignite, the thickness of the bed. The lower 2½ feet was hard lignite, the next 2½ feet was somewhat slacked and broken, and the upper 2 feet was hard lignite. The roof was a shaly clay, and the floor blue clay.

Notes.—In 1910 the mine was worked only in the winter and the output was small.

For analyses and descriptions of other samples of lignite from this mine, see Bull. 22, Bureau of Mines, 1913, pp. 135, 630.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 471, 1912, p. 345.

CULBERTSON. BUTTERFIELD MINE.

Sample.—Lignite; Culbertson field; analysis No. 10727 (p. 56).

Mine.—Butterfield, in sec. 3, T. 27 N., R. 56 E., 3¾ miles southeast of Culbertson, which is the nearest station on the Great Northern Railway.

Lignite bed.—DD bed of the United States Geological Survey. Tertiary age, Fort Union formation. Thickness, 6 feet. Roof, sandy shale; floor, clay.

The bed was sampled by A. L. Beekly on July 18, 1910. The sample was taken about 150 feet from the mouth of the entry, the upper 18 inches being left to support the roof, and represents only the lower 4½ feet of the bed.

Notes.—The lignite is hard and of fair quality. The woody structure is so evident that it is often difficult to determine whether it is "wood" or coal. The lignite is tough and elastic and requires a strong blasting powder. At the time of sampling the mine was small and produced only 5 or 6 wagon loads a day during the winter season.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 340.

CULBERTSON. DEMPSEY MINE.

Sample.—Lignite; Culbertson field; analysis No. 10725 (p. 56).

Mine.—Dempsey, in sec. 34, T. 28 N., R. 56 E., 3 miles southeast of Culbertson, which is the nearest station on the Great Northern Railway.

Lignite bed.—E bed of the United States Geological Survey. Tertiary age, Fort Union formation. Thickness, 6 feet 6 inches. Roof, sandy clay; floor, shaly clay. The lignite is free from partings and consists of alternating hard and soft layers; part of the bed is brown.

The bed was sampled at one point in the mine by A. L. Beekly on July 21, 1910. Sample 10725 was taken at the face, about 60 feet northeast of the mouth.

Notes.—The mine was not being worked at the time the sample was taken.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 345.

CULBERTSON. PROSPECT.

Sample.—Lignite; Culbertson field; analysis No. 10726 (p. 56).

Mine.—Prospect, in sec. 34, T. 28 N., R. 56 E., 3 miles southeast of Culbertson, on the Northern Pacific Railway.

Lignite bed.—F bed of the United States Geological Survey. Tertiary age, Fort Union formation. Thickness, 7 feet. Roof, sandy shale; floor, blue clay. The lignite is soft brown.

The bed was sampled at one point in the mine by A. L. Beekly on July 21, 1910. The sample was taken in the entry, about 50 feet from the mouth of the entry, and represents 7 feet of lignite, the thickness of the bed.

Notes.—This prospect is about one-half mile from the Peck mine. It was not being worked at the time the sample was taken.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 345.

FROID. ASTROPE MINE.

Sample.—Lignite; Culbertson field; analysis No. 11005 (p. 56).

Mine.—The Astrope, a local mine in sec. 15, T. 29 N., R. 55 E., 5½ miles southwest of Froid, which is the nearest station on the Bainville and Plentywood branch of the Great Northern Railway.

Lignite bed.—F of the United States Geological Survey, early Eocene age, Fort Union formation. Thickness, 6 feet. The upper 1 foot is soft, dirty lignite. The bed was sampled at one point in the mine by A. L. Beekly on September 14, 1910. The sample was taken in the mine 115 feet north of the mine mouth, and represents the lower 5 feet of the bed.

Notes.—At the time of sampling the output was 1 to 3 tons a day.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 348.

FROID. PROSPECT.

Sample.—Lignite; Culbertson field; analysis No. 11004 (p. 56).

Location.—A well in sec. 11, T. 29 N., R. 55 E., 3½ miles southwest of Froid, which is the nearest station on the Bainville and Plentywood branch of the Great Northern Railway.

Lignite bed.—F of the United States Geological Survey, early Eocene age, Fort Union formation. Thickness, 7 feet, 10 inches. Roof, clay; floor, clay. There were no partings in the coal.

The bed was sampled in the well by A. L. Beekly on September 14, 1910. The well, which was 6 feet in diameter, penetrated the bed at a depth of 30 feet below the surface.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 348.

MEDICINE LAKE. BELGON MINE.

Sample.—Lignite; Culbertson field; analysis No. 10899 (p. 56).

Mine.—Belgon, an open-pit mine in sec. 4, T. 31 N., R. 59 E., 15 miles east of Medicine Lake, which is the nearest station on the Bainville and Plentywood branch of the Great Northern Railway.

Lignite bed.—No name. Early Eocene age, Fort Union formation. Thickness, 10 feet.

The bed was measured and sampled at one point in the pit by A. L. Beekly on September 6, 1910.

The sample was taken from the fresh working face where the bed was 10 feet thick, and represents 7 feet 6 inches of hard, tough lignite, with a woody grain. There was 1 foot 6 inches of soft, brownish-black lignite above the part sampled and 1 foot of wet, muddy lignite below it. The roof was shale.

For a description of the geologic relations of the lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 350.

MEDICINE LAKE. COAL RIDGE MINE.

Sample.—Lignite; Culbertson field; analysis No. 11002 (p. 56).

Mine.—Coal Ridge, a drift mine in sec. 8, T. 34 N., R. 58 E., about 22 miles northeast of Medicine Lake, which is the nearest station on the Bainville and Plentywood branch of the Great Northern Railway.

Lignite bed.—Known as the Coal Ridge, early Eocene age, Fort Union formation. Thickness, 10 to 16 feet. The bed becomes thinner toward the west.

The bed was measured and sampled at one point in the mine by A. L. Beckly on September 23, 1910.

The sample was taken at the face of the entry, about 400 feet from the mine mouth, where the bed was 10 feet thick. The sample represents 6 feet 6 inches of lignite. The upper 2 feet are left for roof, and the lower 1 foot 6 inches for floor.

Notes.—The outcrop lignite was badly slacked and contained thin streaks of shale. The lignite at the outcrop was 14 feet thick.

At the time of sampling the mine was small, producing only about 3 tons a day.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 352.

MEDICINE LAKE. JONES MINE.

Sample.—Lignite; Culbertson field; analysis No. 11006 (p. 56).

Mine.—Jones, in sec. 30, T. 33 N., R. 56 E., 6 miles northeast of Medicine Lake, which is the nearest station on the Bainville and Plentywood branch of the Great Northern Railway.

Lignite bed.—Locally known as Jones. Early Eocene age, Fort Union formation. Thickness, about 7 feet. Above the coal is a sandy clay; in mining, 1 foot 6 inches of coal is left for roof.

The bed was sampled at one point in the mine by A. L. Beekly on September 19, 1910. The sample was taken 250 feet south of the entry mouth and represents 5 feet 6 inches of lignite. At the time of sampling the output was 3 to 5 tons a day.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 351.

MEDICINE LAKE. TED YOUNG MINE.

Sample.—Lignite; Culbertson field; analysis No. 11003 (p. 56).

Mine.—Ted Young, a drift mine in sec. 6, T. 31 N., R. 56 E., 2½ miles southeast of Medicine Lake, which is the nearest station on the Bainville and Plentywood branch of the Great Northern Railway.

Lignite bed.—Locally known as Young. Early Eocene age, Fort Union formation. Thickness, 3 feet 6 inches. The bed was measured and sampled at one point in the mine by A. L. Beekly on September 17, 1910. The sample was taken about 300 feet southeast of the entry mouth and represents 3 feet 6 inches, the thickness of the bed, of hard, black, clean lignite.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 349.

MONDAK. OPEN PIT MINE.

Sample.—Lignite; Culbertson field; analysis No. 10900 (p. 57).

Mine.—Open pit in sec. 19, T. 27 N., R. 59 E., 7 miles northwest of Mondak, on the Great Northern Railway, and 1½ miles from the railway.

Lignite bed.—G of the United States Geological Survey. Early Eocene age, Fort Union formation. Thickness, 4 feet 6 inches. The lignite is brownish black. The clay overburden, 10 feet thick, is removed in mining.

The bed was sampled at one place in the pit by A. L. Beekly on August 23, 1910. The upper 2 feet of the coal is more or less weathered and soft, and the sample does not include this part of the bed.

For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 471, 1912, p. 342.

PLENTYWOOD. PIERCE MINE.

Sample.—Lignite; Plentywood field; analysis No. 14670 (p. 57).

Mine.—Pierce; in SE. ¼ sec. 10, T. 35 N., R. 55 E., 2½ miles northeast of Plentywood.

Lignite bed.—Designated as "Richardson bed" by the United States Geological Survey. The bed lies about 800 feet above the base of the Fort Union formation of Tertiary age. At the point of sampling it is overlain by about 30 feet of Fort Union cover and 30 feet of glacial drift. Quality and thickness, irregular. Dip, about 3 feet per thousand southeastward.

The bed was measured and sampled at one point in the mine by C. M. Bauer on August 20, 1912, as described below:

Section of lignite bed in Pierce mine.

Laboratory No.	14670	
Roof, carbonaceous shale.	*Ft.*	*in.*
Lignite	0	3
Shale	[a]0	8
Lignite	0	6
Shale	[a]0	3
Lignite, containing gypsum	2	0
Clay	[a]0	7
Lignite	3	9
Floor shale.		
Thickness of bed	8	0
Thickness of lignite sampled	6	6

[a] Excluded from sample.

The above section (sample 14670) was measured in the mine, 250 feet northwest of the opening.

Notes.—For a description of the geologic relations of this lignite bed see U. S. Geol. Survey Bull. 541.

NEVADA.

ESMERALDA COUNTY.

COALDALE. DARMS MINE.

Sample.—Bituminous coal; Coaldale field; analysis No. 13978 (p. 57).

Mine.—Darms, a slope mine in NE. ¼ sec. 33, T. 2 N., R. 37 E.; 3¼ miles southeast of Coaldale, on the Tonopah & Goldfield Railroad.

Coal bed.—C. Tertiary (Eocene or Miocene) age. Thickness, variable. Dip variable, being 60° NE. at the outcrop, 77° at 245 feet down the slope, and 10° at 280 feet. Strike, N. 53° W.

The bed was measured and sampled by J. H. Hance on March 7, 1912. Sample 13978 was taken in a drift off the slope, 300 feet from the mouth of the slope and 20 feet from the slope. The sample represents 7 feet of impure coal, the thickness of the bed.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 531, 1913, p. 321.

COALDALE. NEVADA MINE.

Sample.—Bituminous coal; Coaldale field; analyses Nos. 13979, 13980, and 14409 (p. 57).

Mine.—Nevada, a slope mine in S. ½ of SE. ¼ sec. 29, T. 2 N., R. 37 E., 3½ miles southeast of Coaldale, on the Tonopah & Goldfield Railroad.

Coal bed.—C. Tertiary (Eocene or Miocene) age; dip variable. The bed is cut off to the south by a fault.

The bed was measured and sampled at two points in the mine by J. H. Hance on March 8, 1912.

Notes.—Sample 13979 was taken at a point in the slope 150 feet from the mouth and represents 9 feet of impure coal, the entire thickness of the bed. Sample dry.

Sample 13980 was taken at a place in the mine 100 feet from the mouth and represents 8 feet of impure coal, the entire thickness of the bed.

Sample 14409 was taken from a block of coal representing a bench 12 inches thick in the mine. This block was sent by freight to Washington and the sample was cut by M. R. Campbell on July 17, 1912. The sample is supposed to represent the best coal in the bed.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 531, 1913, p. 320.

NEW MEXICO.

BERNALILLO COUNTY.

ALBUQUERQUE. HOLMES MINE.

Sample.—Bituminous coal; Tijeras field; analysis No. 12650 (p. 57).

Mine.—Holmes; a prospect drift entry in SW. ¼ sec. 6, T. 10 N., R. 6 E., about 25 miles east of Albuquerque.

Coal bed.—Not named. Cretaceous age; Mesaverde formation. Thickness, 6 inches to 2 feet 3 inches. There is shale above and below the coal. The coal is impure and badly crushed.

The bed was measured and sampled by W. T. Lee on July 17, 1911.

The sample was taken 300 feet from the mouth of the entry from a fresh face where the coal was 2 feet 3 inches thick. The same bed in other parts of the entry was only 6 inches thick. The differences were probably due to crushing.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 471, 1912, p. 576.

COLFAX COUNTY.

DAWSON. DAWSON NO. 2 MINE.

Sample.—Bituminous (coking) coal; Raton field; analyses Nos. 12233, 12234, and 12235 (p. 57).

Mine.—Dawson No. 2, a drift mine in the Dawson district. The Dawson mines are in secs. 1, 2, 11, and 12, T. 29 N., and secs. 35 and 36, T. 28 N., R. 20 E., at Dawson, on the El Paso & Southwestern Railroad.

Coal bed.—"Raton." Cretaceous age, Vermejo formation. Average thickness, 6 feet; dip, 1° northwest; cleat, east and west. Roof, sandstone; floor, usually smooth clay and shale. Cover at points of sampling, 250 feet.

The bed was measured and sampled at two points in the mine by J. C. Roberts on May 17, 1911, as described below:

Sections of coal bed in Dawson No. 2 mine.

Section	A		B	
Laboratory No.	12233		12234	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	[a]1	6	..	..
Coal	2	4	0	6
Coal, bony	0	2	0	5
Coal	1	2	7	4
Coal, bony	0	½	..	..
Coal	2	9½	..	..
Coal, bony	[a]1	6	..	..
Floor, bony coal and shale.				
Thickness of bed	9	6	8	3
Thickness of coal sampled	6	6	8	3

[a] Not included in sample.

Section A (sample 12233) was measured at the face of the fifth north, 6,700 feet northeast of the "high line" entrance and represents a full face sample.

Section B (sample 12234) was measured at the face of the thirteenth east, 6,500 feet from the "high line" entrance.

Notes.—There are 5 mines at Dawson. At the time of sampling the mines had the following capacities: No. 1, 1,500 tons a day; No. 2, 2,000; No. 4, 1,400; and No. 5, 2,000 tons. The room-and-pillar method of mining is employed. The coal was shot down with "forty per cent" dynamite. None of the coal was shipped as run-of-mine. About 2,000 tons a day was sent to the screens and about 40 per cent passed through a shaking screen with 2-inch and 4-inch openings; pickers were employed. The screenings were coked; the coke ovens had a daily capacity of 1,600 tons. There were 5 loading tracks with a capacity of 200 empty and 200 loaded cars.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 137, 640.

GARDINER. GARDINER NO. 1 MINE.

Sample.—Bituminous coal; Raton field; analysis No. 14795 (p. 57).

Mine.—Gardiner No. 1, a (new) drift mine in the Blossburg district, at Gardiner, one-half mile southeast of Blossburg.

Coal bed.—Raton. Cretaceous age, Vermejo formation. The bed is regular in thickness, but its value is diminished in some places by crushing and other movements of the inclosing rocks. Roof, shale; floor, shale.

The bed was measured and sampled at one point in the mine by W. T. Lee on September 27, 1912, as described below:

Section of coal bed in Gardiner No. 1 mine.

Laboratory No.	14795	
Roof, shale.	*Ft.*	*in.*
Coal, bony	[a]0	7
Coal	5	5
Floor, shale.		
Thickness of bed	6	0
Thickness of coal sampled	5	5

[a] Not included in sample.

Notes.—The sample was collected from a fresh face in room 6 off first west entry off the main entry, 400 feet from the outcrop. At the time of sampling the mine was doing development work and was shipping only 100 tons a day.

KOEHLER. KOEHLER NO. 1 MINE.

Sample.—Bituminous (coking) coal; Raton field; analyses Nos. 12335, 12336, 12337, and 14796 (p. 58).

Mine.—Koehler No. 1, a drift mine in the Koehler district, T. 29 N., R. 22 E., at Koehler, on the St. Louis, Rocky Mountain & Pacific Railway.

Coal bed.—Raton. Cretaceous age, Vermejo formation. Average thickness, 9 feet, regular; dip 1° west. The roof is shale, bony coal, or sandstone, according to the locality. In most places where conglomeratic sandstone rests on the coal the bed has been partly eroded and is too thin to be mined. Floor, shale; cover, 450 to 500 feet.

The bed was measured and sampled at three points in the mine by J. C. Roberts on May 28, 1911, and at one point by W. T. Lee on September 27, 1912, as described below:

Sections of coal bed in Koehler No. 1 mine.

Section	A	B	C	D
Laboratory No.	12335	12336	12337	14796
Roof, shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal, bony	[a] 2 0	[a] 2 0	.. ..	[a] 2 0
Coal	6 0	6 2	5 9	6 3
Coal (poor)	[a] 1 0	.. ..	.. ..	.. ..
Floor, shale.				
Thickness of bed	9 0	8 2	5 9	8 3
Thickness of coal sampled	6 0	6 2	5 9	6 3

[a] Not included in sample.

Section A (sample 12335) was measured as the face of room 20, off seventh east entry, 4,500 feet southeast of the mouth of the main entry.

Section B (sample 12336) was measured at the face of eighth west entry, 4,300 feet southeast of the mouth of the main entry.

Section C (sample 12337) was measured in room 20, off seventh east entry, 4,200 feet southeast of the mouth of the main entry.

Section D (sample 14796) was measured at the face of ninth west entry, 4,900 feet southwest of the main opening in Prairie Crow Canyon.

Notes.—There are three mines at Koehler. The room-and-pillar method of mining is employed. At the time of sampling the mines had the following capacity: No. 1, 1,000 tons a day; No. 2, 1,400 tons; and No. 3, 400 tons. From 300 to 350 men were employed. There was enough coal to maintain the above output for many years. Ten per cent of the coal was shipped as run-of-mine. The washer had a capacity of 1,200 tons a day, the coke ovens 700 tons, and the storage bins 800 tons. There were four loading tracks with a capacity of 60 empty and 80 loaded cars.

SUGARITE. SUGARITE NO. 1 AND NO. 2 MINES.

Sample.—Bituminous coal; Raton field; analyses Nos. 14791 and 14792 (p. 58).

Mines.—Sugarite No. 1 in NE. ¼ NE. ¼ sec. 9, T. 31 N., R. 24 E.; and Sugarite No. 2 in NE. ¼ SW. ¼ sec. 3, T. 31 N., R. 24 E. (private survey); drift mines at Sugarite.

Coal bed.—Sugarite. Cretaceous or Tertiary age, Raton formation. Roof, shale, with thin seams of coal; floor, shale.

The bed was measured and sampled at one point in No. 1 mine by W. T. Lee on September 27, 1912, as described below.

Section of coal bed in Sugarite No. 1 mine.

Laboratory No.	14791
Roof, shale, with thin seams of coal.	*Ft.* *in.*
Coal	[a]0 4
Shale	[a]0 1½
Coal	5 1
Floor, shale.	
Thickness of bed	5 6½
Thickness of coal sampled	5 1

[a] Not included in sample.

The above section (sample 14791) was measured in the entry, 1,500 feet southwest of the mine mouth.

The bed was measured and sampled at one point in No. 2 mine by W. T. Lee on September 27, 1912, as described below:

Section of coal bed in Sugarite No. 2 mine.

Laboratory No.	14792
Shale, with thin seams of coal.	*Ft.* *in.*
Coal	[a]0 6
Shale	[a]0 2
Coal	3 5
Bone	[a]0 2
Coal	1 4
Shale.	
Thickness of bed	5 7
Thickness of coal sampled	4 9

[a] Not included in sample.

The above section (sample 14792) was measured in the entry, 1,200 feet northeast of the mine mouth.

Notes.—At the time the samples were taken the mines were new, and all the coal produced was from the main entries.

YANKEE. YANKEE No. 3 MINE.

Sample.—Bituminous (noncoking) coal; Raton field; analyses Nos. 13365, 13366, and 13367 (p. 58).

Mine.—Yankee No. 3, a slope mine at Yankee, on the Santa Fe, Raton & Eastern Railroad with connections to the Atchison, Topeka & Santa Fe Railway and the St. Louis, Rocky Mountain & Pacific Railway.

Coal bed.—Yankee. Cretaceous or Tertiary age, Raton formation. Average thickness, 5 feet 6 inches; dip, 4 per cent northwest; cleat, east and west. The roof is a soft black shale which is treacherous; above the shale is a sandstone cap rock. Floor, soft smooth fire clay. Particles of the roof did not become mixed with the coal in mining but particles of the floor did. Cover at points of sampling, 250 to 300 feet.

The bed was measured and sampled at two points in the mine by J. C. Roberts on February 8, 1912, as described below:

Sections of coal bed in Yankee No. 3 mine, near Yankee.

Section	A	B
Laboratory No.	13365	13366
Roof, shale.	*Ft.* *in.*	*Ft.* *in.*
Coal	0 5	0 5
Shale	[a]0 1½	[a]0 4
Coal	1 0	1 0
Shale	[a]0 1	[a]0 1½
Coal	0 7	0 7
Shale	[a]1 0	[a]1 3
Coal	2 7	2 7
Floor, shale.		
Thickness of bed	5 9½	6 3½
Thickness of coal sampled	4 7	4 7

[a] Not included in sample.

Section A (sample 13365) was measured at the face of room 2, off tenth west entry 10.

Section B (sample 13366) was measured at the face of fifth south entry.

A composite sample was made by mixing face samples 13365 and 13366. The results of an ultimate analysis of this sample are shown under laboratory No. 13367.

Notes.—At the time of sampling, the coal was undercut by hand; the cuttings were not loaded with the coal. Permissible explosives were used for breaking down the coal and for brushing the roof and floor. All the output was shipped as run-of-mine. The tipple was equipped with a bar screen 16 feet long and 3 feet wide with 3-inch openings between the bars; a shaking screen 10 feet long by 8 feet wide, with ¾-inch, 2-inch, and 3-inch holes; and a revolving screen 16 feet long with ¾-inch openings. Pickers were employed at the conveyor belt and on the car. The lumps were medium large and of good appearance. There were four loading tracks with a capacity of 35 empty and 35 loaded cars. The daily output of the mine was 200 tons; the intent was to increase it to 2,500 tons. The mine had a probable life of 10 years.

For analysis and description of another sample of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 139, 645.

LINCOLN COUNTY.

White Oaks. Old Abe Mine.

Sample.—Bituminous coal; White Oaks field; analysis No. 15053 (p. 59).

Mine.—Old Abe, a slope mine in NW. ¼ sec. 5, T. 7 S., R. 13 E., 2½ miles southeast of White Oaks.

Coal bed.—Old Abe. Post-Laramie (?) age. Thickness, variable; dip, 20° west. The bed was measured and sampled at one point in the mine by C. H. Wegemann on November 3, 1912, as described below:

Section of coal bed in Old Abe mine.

Laboratory No.	15053	
Roof, sandstone.	*Ft.*	*in.*
Coal	0	2
Bone	0	1
Coal	1	½
Bone	0	¼
Coal	0	10¾
Bone	0	¾
Coal	0	4
Coal, poor	0	1
Coal	1	4
Floor, shale.		
Thickness of bed	4	½
Thickness of coal sampled	4	¼

The above section was measured in the main slope, 300 feet from the slope mouth. At the time of sampling a slope had been run on the dip of the bed (20° west) and side entries driven from it, the rooms being driven up the rise from the entries.

For analyses and descriptions of another sample of coal from this mine, see Bull. 22, Bureau of Mines, 1913, pp. 139, 646.

White Oaks. Wild Cat Mine.

Sample.—Bituminous coal; White Oaks field; analysis No. 15054 (p. 59).

Mine.—Wild Cat, a slope mine in SW. ¼ sec. 32, T. 6 S., R. 13 E., 2 miles southeast of White Oaks.

Coal bed.—Not named. Post-Laramie (?) age. Thickness, variable; dip, 25° west. The bed was measured and sampled at one point in the mine by C. H. Wegemann, on November 3, 1912, as shown below:

Section of bed in Wild Cat mine.

Laboratory No.	15054
Roof, black shale.	*Ft. in.*
Bone	[a]0 3½
Coal	0 8¼
Bone	0 1¼
Coal	0 11
Coal, bony	0 1
Floor, black shale.	
Thickness of bed	2 1
Thickness of coal sampled	1 9½

[a] Not included in sample.

The above section (sample 15054) was measured near the foot of the main slope. At the time of sampling a slope about 240 feet in length had been run on the dip of the bed (25° west) and entries driven from it, the rooms being driven up the rise.

M'KINLEY COUNTY.

BLACKROCK. ZUNI INDIAN SCHOOL MINE.

Sample.—Subbituminous coal; Durango-Gallup field; analysis No. 15032 (p. 59).

Mine.—Zuni Indian School, a drift mine in sec. 6, T. 11 N., R. 17 W., 10 miles northeast of Blackrock.

Coal bed.—Not named. Cretaceous age, in the Benton group, below the Mesaverde formation. The bed was measured and sampled at one point in the mine by D. E. Winchester, on October 11, 1912, as described below:

Section of coal bed in Zuni Indian School mine.

Laboratory No.	15032
Roof, shale.	*Ft. in.*
Bone	[a]0 2
Coal	1 10
Bone	[a]0 2
Coal	1 0
Floor, shale.	
Thickness of bed	3 2
Thickness of coal sampled	2 10

[a] Not included in sample.

Note.—The sample was taken in room 2, west of the main entry.

SANTA FE COUNTY.

MADRID. ANTHRACITE NO. 4 MINE.

Sample.—Anthracite coal; Cerrillos field; analysis No. 14886 (p. 59).

Mine.—Anthracite No. 4, a slope mine at Madrid, on the Ortiz land grant.

Coal bed.—White Ash. Cretaceous age; Mesaverde formation. Thickness, relatively uniform; dip, about 15° east. The roof is shale in most places, in some places it is formed by an overlying sill of intrusive rock, which has locally metamorphosed the coal. The floor is shale.

The bed was measured and sampled at one point in the mine by W. T. Lee on October 10, 1912.

The sample was collected from a working face 1,500 feet southeast of the mouth of the main entry, at a point where the coal was 2 feet 9 inches thick.

Note.—The output at the time the sample was taken was 150 tons a day. The coal was used mainly for domestic purposes.

For a description of the geologic relations of this coal bed see U. S. Geological Survey Bull. 531, 1913, p. 286.

Madrid. Blacksmith Mine.

Sample.—Bituminous coal; Cerrillos field; analysis No. 14884 (p. 59).

Mine.—Blacksmith, a drift mine at Madrid, on the Ortiz land grant.

Coal bed.—Peacock (?). Cretaceous age; Mesaverde formation.

The bed was measured and sampled at one point in the mine by W. T. Lee on October 10, 1912, as described below:

Section of coal bed in Blacksmith mine.

Laboratory No.	14884	
	Ft.	*in.*
Roof, shale.		
Coal	[a]0	6
Shale	[a]2	2
Coal	0	9
Shale	[a]0	3
Coal	1	7
Floor, shale.		
Thickness of bed	5	3
Thickness of coal sampled	2	4

[a] Not included in sample.

Notes.—The sample was collected from a fresh face in the mine, 300 feet from the mouth of the mine. The coal at this point, as in general throughout the mine, is crushed by movements of the containing rocks.

For a description of the geologic relations of this coal bed see U. S. Geological Survey Bull. 531, 1913, p. 297.

Madrid. Holen Mine.

Sample.—Bituminous coal; Cerrillos field; analysis No. 14885 (p. 59).

Mine.—Holen, a slope mine at Madrid, on the Ortiz land grant.

Coal bed.—Cook & White. Cretaceous age; Mesaverde formation. The roof and floor are shale.

The bed was measured and sampled at one point in the mine by W. T. Lee on October 9, 1912, as described below:

Section of coal bed in Holen mine.

Laboratory No.	14885	
	Ft.	*in.*
Roof, shale.		
Bone	[a]0	2
Coal	3	3
Floor, shale.		
Thickness of bed	3	5
Thickness of coal sampled	3	3

[a] Not included in sample.

The sample was taken from a working face 1,300 feet east of the mine mouth.

Notes.—The coal near the surface does not coke, but a few hundred feet from the surface it changes to a coking coal. At the time of sampling the output of the mine was 100 tons a day. The coal was used mainly for domestic fuel.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 531, 1913, p. 306.

Madrid. Peacock Prospect.

Sample.—Bituminous coal; Cerrillos field; analysis No. 14887 (p. 59).

Mine.—Peacock, a prospect slope entry at Madrid, on the Ortiz land grant.

Coal bed.—Peacock. Cretaceous age, Mesaverde formation. Both roof and floor are bony coal.

The bed was measured and sampled at one point in the mine by W. T. Lee on October 10, 1912.

The sample was taken in the entry 400 feet northeast of the mouth of the entry from a face freshly cleared for the purpose of sampling. The sample represents 2 feet 4 inches of coal. There was bony coal above and below the part sampled.

Notes.—The prospect was not being worked at the time of sampling.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 531, 1913, p. 297.

NORTH DAKOTA.

ADAMS COUNTY.

HAYNES. NIPPER & MONROE MINE.

Sample.—Lignite; Cannonball River field; analysis No. 14542 (p. 59).

Mine.—Nipper & Monroe, a drift mine in sec. 16, T. 129 N., R. 94 W., 3½ miles northeast of Haynes.

Lignite bed.—Haynes. Tertiary (Eocene) age, Fort Union formation. Average thickness reported, 12 feet.

The bed was sampled at one point in the mine by E. R. Lloyd on July 30, 1912. The part sampled was 8 feet 3 inches thick. The top of the bed had been left for roof because the overlying shale was not strong.

The sample was collected 630 feet east of the mouth of the entry.

For a description of the geologic relations of this bed see U. S. Geol. Survey Bull. 541.

HAYNES. WILLIAM PINKHAM MINE.

Sample.—Lignite; Cannonball River field; analysis No. 14544 (p. 59).

Mine.—William Pinkham, in sec. 36, T. 130 N., R. 94 W., about 9 miles northwest of Haynes.

Lignite bed.—Haynes. Tertiary (Eocene) age, Fort Union formation. Thickness, about 12 feet.

The bed was sampled at one point in the mine by E. R. Lloyd on July 30, 1912. The part sampled was 8 feet 10 inches thick and uniform in quality. The top of the bed had been left for roof because the overlying shale was weak.

The sample was taken 225 feet south of the mouth of the mine.

For a description of the geologic relations of this bed see U. S. Geol. Survey Bull. 541.

BOWMAN COUNTY.

AMOR. DURKIN PROSPECT.

Sample.—Lignite; Bowman County field; analysis No. 14857 (p. 59).

Location.—Open pit, known as the Durkin prospect, 2 miles southeast of Amor post office in SW. ¼ sec. 34, T. 130 N., R. 103 W., and 12 miles southwest of Bowman.

Lignite bed.—This bed is included in the Ludlow lignitic member of the Lance formation, Cretaceous or Tertiary age, and is designated as the T Cross bed from its best exposure, being at the T Cross ranch on Bacon Creek in sec. 20, T. 133 N., R. 104 W., in Billings County, where it is 24 feet thick. Its average thickness in Bowman County is about 5 feet, ranging from 3 feet 2 inches to 10 feet. The bed lies nearly flat, dipping only slightly to the northeast.

The bed was measured and sampled July 31, 1912, by C. J. Hares, as shown below:

Section of lignite in outcrop 2 miles southeast of Amor.

Laboratory No.	14857
Roof, sandstone.	*Ft. in.*
Lignite, dirty and badly weathered	[a]3 7
Lignite, dirty	[a]0 7
Lignite, fair	0 9
Lignite, good, but with considerable marcasite	[a]0 2
Lignite, excellent	1 10½
Lignite, good, under water	[a]1 8
Floor, not stated.	
Thickness of bed	8 7½
Thickness of lignite sampled	2 7½

[a] Not included in sample.

Notes.—There was 1 foot 6 inches of soft sandstone, overlain by 5 feet of shale and soil, above the lignite. The lignite is mined by stripping. At the time the sample was collected no lignite had been removed for some time, a face was cleaned and a fairly fresh sample obtained. The lignite was used wholly for local purposes.

SCRANTON. SCRANTON MINE.

Sample.—Lignite; Bowman County field; analysis No. 14485 (p. 59).

Mine.—Scranton, a drift mine in SW. ¼ sec. 24, T. 131 N., R. 100 W., at Scranton, on the Chicago, Milwaukee & St. Paul Railroad.

Lignite bed.—Probably the Harmon bed of Smith and Leonard (see U. S. Geol. Survey Bull. 341, p. 24) and the same one as sampled on Sand Creek (see U. S. Geol. Survey Bull. 285, p. 324) where it is 31 feet thick. About 200 feet above the base of the Fort Union formation, Tertiary age. Thickness, 19 to 23 feet, average 20½ feet. The bed lies nearly flat. It is by far the most important, thickest, and most extensive bed in Bowman and Billings counties.

The bed was measured and sampled at one point in the mine on June 30, 1912, by E. G. Woodruff and C. J. Hares, as shown below:

Section of lignite bed in Scranton mine.

Laboratory No.	14485
Clay.	*Ft. in.*
Lignite	[a]5 0
Lignite, impure	[a]0 7
Lignite	6 0
Shale, brown, carbonaceous	[a]0 1½
Lignite	[a]0 7
Lignite	[a]8 0
Thickness of bed	20 3½
Thickness of lignite sampled	6 0

[a] Not included in sample.

The sample was taken at the face of the main east entry, off the main entry, about 500 feet from the main entry and 1,000 feet from the mouth of the main entry. The upper 5 feet of the bed is left for roof and the lower 8 feet for floor.

Notes.—The lignite from this mine is tough and woody but readily slacks upon exposure to air and sunlight. For that reason the lignite will not withstand long shipment. At the time of sampling it was used at points along the Chicago, Milwaukee & St. Paul Railroad, in the brick yards at Scranton, and by farmers.

For results of briquetting tests of this lignite see Bull. 14, Bureau of Mines, 1912, pp. 31, 32.

For analyses and descriptions of other samples of lignite from this mine see Bull. 22, Bureau of Mines, 1913, pp. 142, 658.

MORTON COUNTY.

LEITH. JONES MINE.

Sample.—Lignite; Cannonball River field; analysis No. 14729 (p. 60).

Mine.—Jones, a shaft mine in sec. 12, T. 133 N., R. 88 W., 1½ miles west of Leith. Cover, 42 feet.

Lignite bed.—Haynes (?). Cretaceous age, Fort Union formation. The bed is reported to be 8 feet 6 inches thick and to have a shale roof. Only 5 feet 6 inches of the bottom of the bed is being mined, the upper 3 feet being left for roof. The lignite is uniform in quality, from dull to bright luster, and contains thin partings of mineral charcoal. There is also a considerable amount of pyrite in nodules which are easily separable from the lignite.

The bed was sampled at one point in the mine by E. R. Lloyd and C. E. Lesher on September 7, 1912. The sample was taken about 75 feet southwest of the foot of the shaft.

For a description of the geologic relations of the lignite bed see U. S. Geol. Survey Bull. 541.

WARD COUNTY.

PLAZA. PROSPECT.

Sample.—Lignite; analysis No. 11294 (p. 60).

Mine.—Prospect, in sec. 17, T. 152 N., R. 92 W., 25 miles west of Plaza, on the Minneapolis, St. Paul & Sault Ste. Marie Railroad.

Lignite bed.—Known as No. 3. Tertiary age, Fort Union formation. Thickness, 4 feet 11 inches. Roof, shale 6 inches thick; floor, clay. The bed was measured and sampled by M. A. Pishel on October 27, 1910. The bed was 4 feet 11 inches thick. The sample represented the lower 3 feet 8 inches, the upper 1 foot 8 inches being dirty coal. There was shale above the coal and clay below it.

WILLIAMS COUNTY.

AVOCA. BRUEGGER MINE.

Sample.—Lignite; Williston field; analysis No. 12587 (p. 60).

Mine.—Bruegger; in the NW. ¼ SE. ¼ sec. 3, T. 154 N., R. 100 W., one-third mile northeast of Avoca, on the Great Northern Railway.

Lignite bed.—C bed. Tertiary (early Eocene) age, Fort Union formation.

The bed was measured and sampled at one point in the mine by F. A. Herald on September 5, 1911, as described below:

Section of lignite bed in the Bruegger mine.

Laboratory No.	12587	
	Ft.	*in.*
Lignite	[a] 3	4
Lignite	[a] 5	6
Clay	[a] 0	4
Lignite	1	10
Thickness of bed	11	0
Thickness of lignite sampled	5	6

[a] Not included in sample.

Note.—The section was measured 50 feet west of the end of the main entry and 1,225 feet from the mine mouth. The sample was wet.

For a description of the geologic relations of this bed, see U. S. Geol. Survey Bull. 531, 1913, p. 122.

Trenton. Geltz Mine.

Sample.—Lignite; Williston field; analysis No. 12411 (p. 60).

Mine.—Geltz, in the NW. ¼ sec. 27, T. 154 N., R. 102 W., 6 miles northeast of Trenton, on the Great Northern Railway.

Lignite bed.—No name. Tertiary (early Eocene) age, Fort Union formation. The bed was measured and sampled by F. A. Herald on June 30, 1911, as described below:

Section of lignite bed in the Geltz mine.

Laboratory No	12411	
Roof, yellow clay and shale.	*Ft.*	*in.*
Lignite, hard, clean	3	4
Clay, gray	[a]0	5
Lignite	1	9
Floor, not stated.		
Thickness of bed	5	6
Thickness of lignite sampled	5	1

[a] Not included in sample.

Notes.—The sample was collected at the end of the main entry, 150 feet from the mine mouth. The sample represents dry lignite. The mine was worked intermittently; the lignite was used for local purposes.

For a description of the geologic relations of this bed, see U. S. Geol. Survey Bull. 531, 1913, p. 128.

Williston. Powell Mine.

Sample.—Lignite; Williston field; analysis No. 12588 (p. 60).

Mine.—Powell, in NW. ¼ of SE. ¼ sec. 33, T. 154 N., R. 100 W., 3½ miles southeast of Williston, on the Great Northern Railway.

Lignite bed.—C. Tertiary (early Eocene) age, Fort Union formation.

The bed was measured and sampled by F. A. Herald on September 6, 1911.

The sample represents the lower 6 feet 6 inches of the bed, which at this point was 10 feet thick.

Notes.—The sample was taken 50 feet north of the main entry and 900 feet northeast of the mine mouth. The lignite was dry. The daily output was 30 tons in the summer and 50 tons in the winter.

For a description of the geologic relations of this bed, see U. S. Geol. Survey Bull. 531, 1913, p. 122.

Williston. U. S. Reclamation Service Mine.

Sample.—Lignite; Williston field; analysis No. 12533 (p. 60).

Mine.—U. S. Reclamation Service; SE. ¼ NE. ¼ sec. 7, T. 154 N., R. 100 W.; 3 miles northeast of Williston, on the Great Northern Railway.

Lignite bed.—B. Tertiary (early Eocene) age, Fort Union formation. Thickness, 10 feet to 10 feet 6 inches. The bed was measured and sampled by F. A. Herald on August 16, 1911, the sample representing the bottom 8 feet of the bed, which is here 10 feet 3 inches thick.

Notes.—The sample was taken from William Spangler's room, 50 feet north of the main entry and 1,225 feet east of the mine mouth, and represents dry lignite. In 1911 the mine was operated only during the irrigation season. The daily output averaged 90 tons.

For a description of the geologic relations of this bed, see U. S. Geol. Survey Bull. 531, 1913, p. 121.

OHIO.

JEFFERSON COUNTY.

AMSTERDAM. AMSTERDAM MINE.

Sample.—Bituminous coal; Ohio field; analyses Nos. 10533, 15388, 15389, and 15390 (p. 61).

Mine.—Amsterdam, a shaft mine in sec. 18, T. 11 N., R. 4 W., near Amsterdam, on the Lake Erie, Alliance & Wheeling Railroad.

Coal bed.—Lower Freeport (?). Carboniferous age, Allegheny formation. Thickness, excluding bony coal at top of bed, 4 to 4½ feet; fairly uniform. The bed is overlain by shale 12 feet thick; above the shale is another coal bed of variable thickness. The shale roof is very poor; most of the rooms fall in before being worked out. The floor is an underclay, which is taken up in the haulage ways.

The bed was measured and sampled at three points in the mine by D. D. Condit on December 27, 1912, as described below:

Sections of coal bed in Amsterdam mine.

Section	A		B		C	
Laboratory No	15388		15389		15390	
Roof, shale or bony coal.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	[a] 1	3	[a] 1	2	[a] 1	2
Coal	1	11	1	7½	1	6½
Bone	[a] 0	2½	[a] 0	3½	[a] 0	3
Coal	1	10	1	8½	1	11
Coal, impure	[a] 0	5	[a] 0	5	[a] 0	5
Floor, shale.						
Thickness of bed	5	7½	5	2½	5	3½
Thickness of coal sampled	3	9	3	4	3	5½

[a] Not included in sample.

Section A (sample 15388) was measured in room 4, off entry 4, east section, 1,000 feet northeast of the shaft.

Section B (sample 15389) was measured in room 15, off west entry 9, section 2 north, 4,000 feet northwest of the shaft.

Section C (sample 15390) was measured in section 3 south, west entry 8, 4,500 feet southwest of the shaft.

The bed was measured and sampled at one point in the mine by J. W. Paul on May 3, 1910.

The sample (No. 10533) was taken at the center of the face of the face entry, where the bed was 5 feet 2 inches thick, and represented the lower 4 feet 1 inch, all clean coal. There was 13 inches of bony coal above the part sampled.

Notes.—The shaft is 270 feet deep to the top of the coal. The coal is mined by the room-and-pillar method. At the time of sampling the coal was undercut by electric cutting machines.

AMSTERDAM. EASTERN OHIO MINE.

Sample.—Bituminous coal; Ohio field; analyses Nos. 15391, 15392, and 15393 (p. 61).

Mine.—Eastern Ohio, a slope mine in NE. ¼ sec. 8, T. 12 N., R. 4 W., on the Lake Erie, Alliance & Wheeling Railroad, halfway between Amsterdam and Bergholz.

Coal bed.—Lower Freeport (?). Carboniferous age, Allegheny formation. Roof, dark sandy shale, fairly firm. A little below the middle of the bed is a thin, persistent layer of bony coal. The coal is hard and tough.

The bed was measured and sampled at three points in the mine by D. D. Condit on December 28, 1912, as described below:

Sections of coal bed in Eastern Ohio mine.

Section	A		B		C	
Laboratory No	15391		15392		15393	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, impure	[a] 0	4	[a] 0	4	[a] 0	3
Coal	2	5	2	9	3	1
Bone	[a] 0	2	[a] 0	3	[a] 0	2½
Coal	2	3	2	0	1	10
Coal, impure	[a] 0	2	..	..	..	..
Floor, clay and shale.						
Thickness of bed	5	4	5	4	5	4½
Thickness of coal sampled	4	8	4	9	4	11

[a] Not included in sample.

Section A (sample 15391) was measured in room 11, off west butt entry 12, 3,000 feet west of the mouth of the slope.

Section B (sample 15392) was measured at west entry 13, off the south face entry.

Section C (sample 15393) was measured in left butt entry 12, off the north face entry, 2,700 feet northeast of the mouth of the slope.

AMSTERDAM. ELIZABETH MINE.

Sample.—Bituminous coal; Ohio field; analyses numbers 15440, 15441, and 15443 (p. 61).

Mine.—Elizabeth, a shaft mine in SE. ¼ sec. 7, T. 12 N., R. 4 W., 2 miles east of Amsterdam, on a branch of the Lake Erie, Alliance & Wheeling Railroad.

Coal bed.—Probably the Lower Freeport. Carboniferous age, Allegheny formation. Roof, shale; floor, clay.

The bed was measured and sampled at three places in the mine by D. D. Condit on December 30, 1912, as described below:

Sections of coal bed in Elizabeth mine.

Section	A		B		C	
Laboratory No	15440		15441		15443	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	..	..	[a] 0	7	[a] 0	4½
Coal, impure	0	5	0	7	0	6
Bone	[a] 0	2	[a] 0	2½	[a] 0	1
Coal	1	9	1	9	1	7
Bone	[a] 0	3½	[a] 0	3	[a] 0	3½
Coal	1	7	1	9	1	9½
Coal, impure	[a] 0	3	[a] 0	4	..	..
Floor, clay.						
Thickness of bed	4	5½	5	5½	4	7½
Thickness of coal sampled	3	9	4	1	3	10½

[a] Not included in sample.

Section A (sample 15440) was measured in room 22, off west entry 3, 2,600 feet northwest of the shaft.

Section B (sample 15441) was measured in room 33, off the second south face entry, 2,400 feet west of the shaft.

Section C (sample 15443) was measured in room 6, off west entry 9, 2,600 feet northeast of the shaft.

Notes.—The coal is hard and tough and breaks with a conchoidal fracture. The impure bony coal at the top and the two bone layers in the body of the bed are persistent and vary considerably in thickness from place to place. Pyrite bands are plentiful. The shale roof requires little timbering. Most of the mine workings were unusually dry. The equipment of the mine was modern. The daily output of the mine (run-of-mine) was reported at about 1,000 tons at the time of sampling.

BRILLIANT. GOUCHER NO. 2 MINE.

Sample.—Bituminous coal; Ohio field; analyses Nos. 15563 and 15564 (p. 62).

Mine.—Goucher No. 2, a drift mine, in SE. ¼ sec. 6, T. 5 N., R. 2 W., 1 mile west of Brilliant, on Cleveland & Pittsburg division of the Pennsylvania Railroad.

Coal bed.—Pittsburgh or No. 8. Carboniferous age, Monongahela formation. Thickness fairly uniform; at this mine about 5 feet. The bed lies nearly flat.

The bed was measured and sampled at two points in the mine by D. D. Condit on January 4, 1913, as described below:

Sections of coal bed in Goucher No. 2 mine.

Section	A		B	
Laboratory No.	15563		15564	
Roof, clay ("soapstone").	*Ft.*	*in.*	*Ft.*	*in.*
Bone	..	..	[a]0	3
Coal	2	3	2	3
"Sulphur" band	[a]0	¼	[a]0	¼
Coal	[a]0	3½	[a]0	3
"Sulphur" band	[a]0	¼	[a]0	¼
Coal	1	3	2	5
"Sulphur" band	0	⅛	..	..
Coal	0	11	..	..
Floor, clay and shale.				
Thickness of bed	4	9⅛	5	3½
Thickness of coal sampled	4	5⅛	4	8

[a] Not included in sample.

Section A (sample 15563) was measured in face entry 2, north side of mine, 500 feet northwest of the mine mouth. Cover, about 100 feet.

Section B (sample 15564) was measured in room 1, right entry 1, south side, 500 feet northwest of the mine mouth and about 1,200 feet from point where sample 15563 was taken.

Notes.—The output (run-of-mine) was reported at the time of sampling to be 700 tons a day.

HOPEDALE. PARLETT MINE.

Sample.—Bituminous coal; Ohio field; analyses Nos. 15444 and 15445 (p. 62).

Mine.—Parlett, a slope mine in SE. ¼ sec. 32, T. 9 N., R. 3. W, 2 miles southeast of Hopedale, on the Wabash-Pittsburg Terminal Railway.

Coal bed.—Pittsburgh or No. 8 of Ohio Geological Survey. Carboniferous age, Monongahela formation. Average thickness, 4 feet 9 inches. The bed lies nearly flat. The roof is clay and the upper bench of coal characteristic of the Pittsburgh seam. The cover at most places is less than 100 feet.

The bed was measured and sampled at two points in the mine by D. D. Condit on December 31, 1912, as described below:

Sections of coal bed in Parlett mine.

Section	A		B	
Laboratory No.	15444		15445	
Roof, clay or coal.	*Ft.*	*in.*	*Ft.*	*in.*
Coal, impure	..	..	[a]0	11
Clay	..	..	[a]0	3
Coal	..	..	[a]1	1
Clay	[a]1	2	[a]1	0
Coal	2	5½	2	2
Coal with several shale bands	[a]0	3½	[a]0	3
Coal	2	½	2	3
Floor, clay.				
Thickness of bed	5	11½	7	11
Thickness of coal sampled	4	6	4	5

[a] Not included in sample.

Section A (sample 15444) was measured in room 9, off west entry 2, off south entry, 2,200 feet north of the bottom of the slope. Cover, about 30 feet.

Section B (sample 15445) was measured in north entry 2; 180 feet beyond rooms 9 and 10, off west entry, 400 feet northwest of the bottom of the slope. Cover, 28 feet.

Notes.—The mine is very wet. At the time of sampling the output was reported to be 150 tons a day.

PINEY FORK. PINEY FORK NO. 1 AND NO. 2 MINES, AND CABBAGE RUN MINE.

Sample.—Bituminous coal; Ohio field; analyses Nos. 15489, 15490, and 15491 (p. 62).

Mines.—Piney Fork No. 1 and No. 2 mines and Cabbage Run mine, secs. 21, 17, and 16, T. 8 N., R. 3 W., 2 miles southwest of Smithfield, on Lake Erie, Alliance & Wheeling Railroad.

Coal bed.—Pittsburgh or No. 8 of Ohio Geological Survey. Carboniferous age, Monongahela formation. Thickness fairly uniform, about 4½ feet. The bed lies nearly flat. The roof of the main bed of coal is a clay band about 1 foot thick which is taken down in most places. Above this is the "roof" coal, an impure bony bed of variable thickness which has not been mined in this district.

The bed was measured and sampled at one point in No. 1 mine, at one point in No. 2 mine, and at one point in Cabbage Run mine by D. D. Condit on January 3, 1913, as described below:

Sections of coal bed in Piney Fork mines.

Section	A		B		C	
Laboratory No.	15489		15490		15491	
Roof, coal or clay.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Bone	[a]0	2	..	..	..	..
Coal	2	7	2	2¼	2	3½
Clay	[a]0	⅛	[a]0	¼	[a]0	⅛
Coal	[a]0	3	[a]0	3½	[a]0	3½
Clay	[a]0	⅛	[a]0	¼	[a]0	¼
Coal	1	1	1	8	1	3
Clay band	0	⅛	..	..	0	⅛
Coal	0	5	..	..	0	6½
Coal, high in sulphur	[a]0	3	[a]0	3	[a]0	3
Floor, clay and shale.						
Thickness of bed	4	9⅜	4	5¼	4	8
Thickness of coal sampled	4	1⅛	3	10¼	4	1⅛

[a] Not included in sample.

Section A (sample 15489) was measured in mine No. 1, room 14, off left entry 11 off the main east entry, 4,000 feet southwest of the mine mouth. Cover, 125 feet.

Section B (sample 15490) was measured in mine No. 2, room 28, off east entry 21, 5,600 feet northeast of the mine mouth. Cover, 100 feet.

Section C (sample 15491) was measured in Cabbage Run mine, room 26 off left entry 3, 1,050 feet northeast of the mine mouth. Cover, 50 feet.

Notes.—At the time of sampling there were three drift mines in operation at Piney Fork, No. 1 and No. 2, and Cabbage Run mine. The daily output of screened coal from the three was reported to be about 1,250 tons. Sulphur bands and lenses were rather plentiful, but these were carefully sorted out of the coal.

SMITHFIELD. PLUM RUN NO. 1, NO. 4, AND NO. 5 MINES.

Sample.—Bituminous coal; Ohio field; analyses Nos. 15486, 15487, and 15488 (p. 63).

Mines.—Plum Run mines Nos. 1, 4, and 5, in SW. ¼ and NW. ¼ sec. 29 and sec. 30, T. 5 N., R. 2 W., 2 miles east of Smithfield, on branch of Lake Erie, Alliance & Wheeling Railroad.

Coal bed.—Pittsburgh or No. 8 of Ohio Geological Survey. Carboniferous age, Monongahela formation. Thickness, 4½ to 5 feet. The bed lies nearly flat. The coal is overlain by a layer of clay 1 foot thick; above this clay is the "roof coal," which varies in thickness and was not mined. In the middle of the bed is a "bearing in" layer, 3 to 4 inches thick, which was rejected in mining. The coal above this layer varies considerably in thickness from place to place. "Sulphur" in the form of small scattered lenses and bands was common.

The bed was measured and sampled at one point in No. 1, at one point in No. 4, and at one point in No. 5 mine by D. D. Condit on January 2, 1913, as described below:

Sections of coal bed in Plum Run mines.

Section	A		B		C	
Laboratory No	15486		15487		15488	
Roof, clay.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	3	1	10	2	1
"Sulphur" and clay	[a]0	¼	[a]0	½	[a]0	¼
Coal	[a]0	2½	[a]0	2¾	[a]0	2½
"Sulphur" and clay	[a]0	¼	[a]0	¼	[a]0	¼
Coal	2	3	2	3	2	2¼
Floor, clay and shale.						
Thickness of bed	4	9	4	4½	4	6¼
Thickness of coal sampled	4	6	4	1	4	3¼

[a] Not included in sample.

Section A (sample 15486) was measured in mine No. 1, room 7, off first south face entry, off seventh east entry, 2,500 feet northeast of the mine mouth. Cover, about 50 feet.

Section B (sample 15487) was measured in mine No. 4, room 14, off fifteenth right entry, 4,400 feet northeast of the mine mouth. Cover, 225 feet.

Section C (sample 15488) was measured in mine No. 5, room 3, off second right entry, off first right face entry, 1,800 feet southeast of the mine mouth. Cover, 200 feet.

Notes.—At the time of sampling there were four drift mines in operation here. The combined output of screened coal was reported at 1,350 tons a day.

STEUBENVILLE. LA BELLE MINE.

Sample.—Bituminous coal; Ohio field; analyses Nos. 15675 and 15676 (p. 63).

Mine.—La Belle, a shaft mine in SW ¼, sec. 29, T. 2 N., R. 1 W., on the banks of Ohio River in the city of Steubenville.

Coal bed.—Lower Freeport. Carboniferous age, Allegheny formation. The thickness varies greatly from place to place. The bed lies about 100 feet below river level.

The bed was measured and sampled at two points in the mine by D. D. Condit on January 9, 1913, as described below:

Sections of coal bed in La Belle mine.

Section	A		B	
Laboratory No	15675		15676	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	9	1	9
Bone	0	¼	0	1
Coal	0	9	0	8½
Shale	[a]0	1½	[a]0	3
Coal	0	6¼	0	7
Floor, clay, shale.				
Thickness of bed	3	2	3	4½
Thickness of coal sampled	3	½	3	1½

[a] Not included in sample.

Section A (sample 15675) was measured in room 9, off left entry, off second south butt entry, 5,200 feet southeast of the shaft.

Section B (sample 15676) was measured in fifth south butt entry, off southeast entry, 4,400 feet southeast of the shaft.

Notes.—At the time of sampling nearly all the output was from West Virginia territory. The coal was used in iron and steel mills. The coal at the bottom of the bed is somewhat high in sulphur and for that reason was kept separate from the remainder of the bed and used only for steam purposes.

NOBLE COUNTY.

BELLE VALLEY. NOBLE MINE.

Sample.—Bituminous coal; Ohio field; analyses Nos. 17213, 17214, and 17215 (p. 63).

Mine.—Noble, a shaft mine 1 mile northwest of Belle Valley on the Marietta division of the Pennsylvania Railroad.

Coal bed.—Known as No. 7, Carboniferous age, Allegheny formation. Average thickness, about 4 feet 6 inches. Dip, northeast. Rock faults are of frequent occurrence. The roof is shale, roof coal, shale, and sandstone in the order mentioned. The floor is a hard smooth clay. Particles of the roof, but not of the floor, occasionally get mixed with the coal in loading. There are two other workable beds of coal at the Noble mine, one about 250 feet above and the other 100 feet below the No. 7 bed.

The bed was measured and sampled at two points in the mine by H. D. Mason, jr., and G. W. Salisbury on May 21, 1913, as described below.

Sections of coal bed in Noble mine.

Section	A		B	
Laboratory No.	17213		17214	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Top coal laminated with mother coal	0	6½	0	5
Coal	1	9	0	10¾
Mother coal	..	..	0	1
Coal	..	..	1	3
Bony coal	..	..	0	2¼
"Sulphur"	0	1	..	..
Coal	0	8	1	6
"Slate" binder	[a]0	3	[a]0	2
Coal	0	10	0	10
Bottom coal	0	2	..	..
Floor, underclay.				
Thickness of bed	4	3½	5	4
Thickness of coal sampled	4	½	5	2

[a] Not included in sample.

Section A (sample 17213) was measured at the face of eighth south entry (motor road), 5,700 feet southwest of the shaft bottom.

Section B (sample 17214) was measured at the face of room 5 off seventh south entry, 5,600 feet southwest of the shaft bottom.

A composite sample was made by mixing samples 17213 and 17214. The results of an ultimate analysis of this sample are given under laboratory No. 17215.

Note.—The shaft is 185 feet deep. The mine is worked by the room-and-pillar method. At the time of sampling the coal was shot down with FF black blasting powder. Dynamite was used for brushing the floor and roof. About 80 per cent of the coal was shipped as run-of-mine. About 33 per cent of the coal going to the screens passed through. The screens consisted of bars 72 feet long with 1½ and ¾ inch openings. Pickers were employed on the cars in loading. There were three loading tracks with a capacity of 50 empty cars and 50 loaded cars. The average daily output was 600 tons, with a capacity of 1,500 tons. There was about 200 acres to be mined from this opening.

OKLAHOMA.

HASKELL COUNTY.

McCURTAIN. SAN BOIS NO. 2 MINE.

Sample.—Bituminous (coking) coal; McAlester field; analyses Nos. 13688, 13689, 13690, 13691, 13692, and 13693 (p. 64).

Mine.—San Bois No. 2, a slope mine, 1 mile west of McCurtain, on the Fort Smith & Western Railway.

Coal bed.—McCurtain (Hartshorne) bed. Carboniferous (Allegheny) age, McAlester shale. Average thickness, 7 feet. Roof, shale of medium quality; floor, hard clay with smooth surface. Dip, 7° S. Cover at points of sampling, about 225 feet.

The bed was measured and sampled at five points in the mine by R. Y. Williams on March 30 and April 2, 1912, as described below:

Sections of coal bed in the San Bois No. 2 mine.

Section	A		B		C		D		E	
Laboratory No	13688		13689		13690		13691		13692	
Roof, dark gray limey shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, soft, bright, granular	2	2½	2	4	2	3⅞	2	6¼	2	1
Bone	[a]2	3	[a]1	7⅞	[a]1	4	[a]3	2¾	[a]1	2
Coal, soft and bright	0	10½	1	7¼	0	11⅞	1	⅛	0	6
"Sulphur" and bony coal	[a]0	½	0	¼	0	¼	[a]0	½	0	¼
Coal, soft and bright	1	½	1	8¼	2	2	2	2	2	8¾
Coal, containing 3 bone streaks	0	1	..	..	..	..	..	..	..	..
Coal, soft and bright	1	6⅜	..	..	..	..	..	..	..	..
Floor, hard shale.										
Thickness of bed	8	⅜	7	3⅝	6	10	8	11⅝	6	6
Thickness of coal sampled	5	8⅞	5	7¾	5	6	5	8⅜	5	4

[a] Not included in sample.

Section A (sample 13688) was measured at the face of thirteenth south entry, 5,000 feet west of the slope mouth.

Section B (sample 13689) was measured at the face of room 3, off thirteenth north entry, top side, 5,000 feet west of the slope mouth.

Section C (sample 13690) was measured at the face of eleventh south entry, 50 feet south of the slope and 5,000 feet west of the slope mouth.

Section D (sample 13691) was measured at the face of twelfth south entry in the top side, 5,500 feet west of the slope mouth.

Section E (sample 13692) was measured at the neck of room 6, twelfth north entry, top side, 5,300 feet west of the slope mouth.

Notes.—The mine is worked on the room-and-pillar system. At the time of sampling the coal was shot off the solid with dynamite. The lower bench was mined first, then the "dirt" band was removed and the upper 3 feet of coal was blasted. The tipple was provided with bar screens and the entire output was screened. The screenings were washed and coked, there being 210 ovens at this plant. Pickers were employed on the cars. The average daily output was about 600 tons.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 149, 675.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 260, 1905, p. 382.

PITTSBURG COUNTY.

ADAMSON. ECLIPSE NO. 1 MINE.

Sample.—Bituminous coal; McAlester field; analyses Nos. 16143, 16144, and 16145 (p. 64).

Mine.—Eclipse No. 1, a slope mine in sec. 9, T. 5 N., R. 17 E., 1½ miles east of Adamson, on the Missouri, Kansas & Texas Railway.

Coal bed.—Lower Hartshorne, Carboniferous (Allegheny) age, McAlester shale. Average thickness, 4 feet 6 inches. Dip, 38° northwest. Cleat, east and west. No faults, rolls, or horsebacks are encountered. The immediate roof is a "draw slate" 1 to 8 inches thick which comes down with the coal; above the "slate" is a gray shale. The floor is a hard smooth shale. Particles from both the floor and roof get mixed with the coal in mining. Cover at points of coupling, about 200 feet. The bed was measured and sampled at two points in the mine by W. T. Burgess on January 30, 1913.

Section A (sample 16143) was measured at the face of the third east entry, 600 feet northeast of the mouth of the slope. The sample represents 4 feet 6 inches of clean coal, the thickness of the bed.

Section B (sample 16144) was measured at the face of the third west entry, 600 feet northwest of the mouth of the slope. The sample represents 4 feet of clean coal, the thickness of the bed.

A composite sample was made by combining samples 16143 and 16144. The ultimate analysis of this sample is given under laboratory No. 16145.

Notes.—The mine is worked by the room-and-pillar system. At the time of sampling the coal was shot off the solid with black powder and "40 per cent" dynamite. These explosives were also used in brushing the floor and roof. The coal was all shipped as run-of-mine. Pickers were employed on the cars. The daily output was about 100 tons, all from advance workings. The mine had a probable life of about five years.

ALDERSON. ALDERSON NO. 5 MINE.

Sample.—Bituminous coal; McAlester field; analyses Nos. 11915, 11916, 11917, and 11918 (p. 65).

Mine.—Alderson No. 5, a shaft mine in sec. 15, T. 5 N., R. 15 E., 1½ miles southwest of Alderson, on the Chicago, Rock Island & Pacific Railway.

Coal bed.—McAlester. Carboniferous (Allegheny) age, McAlester shale. Average thickness, 3 feet 6 inches; dip, 9° southwest. Roof, smooth bluish-gray shale; floor, hard, rough fire clay. Cover, 320 to 625 feet. The bed was measured and sampled at four points in the mine by H. I. Smith on March 6, 1911, as described below:

Sections of coal bed in Alderson No. 5 mine.

Section	A		B		C		D	
Laboratory No	11915		11916		11917		11918	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	..	..	..	..	[a]0	4	..	..
Shale and coal	..	..	..	..	[a]0	6	..	..
Coal, soft	..	..	..	..	0	6	0	7
Coal, uniform	3	7	3	5	2	0	3	0
Floor, clay.								
Thickness of bed	3	7	3	5	3	4	3	7
Thickness of coal sampled	3	7	3	5	2	6	3	7

[a] Not included in sample.

Section A (sample 11915) was measured at the face of room 1, off the fourth west back entry, off the west slope, 2,200 feet west of the shaft.

Section B (sample 11916) was measured at the face of room 7, off seventh west entry, off the west slope, about 2,800 feet northwest of the shaft.

Section C (sample 11917) was measured at the face of room 66, off eleventh east entry, off the east slope, about 4,000 feet east of the shaft.

Section D (sample 11918) was measured at the face of second west entry, off the east slope, about 2,000 feet southeast of the shaft. This section contained a "sulphur" band one-eighth inch thick about 7 inches from the top.

Notes.—The mine is worked by the room-and-pillar system; the shaft is 555 feet deep. At the time of sampling the coal was undercut and was shot down with FF black powder and a permissible explosive. The coal was friable, about 75 per cent passing through a 4-inch screen. Pickers were employed on the cars in loading. The four loading tracks had a capacity of 25 empty cars and 100 loaded cars. The daily output was 400 tons, all from advance workings. The pillars were not robbed.

ALDERSON. ALDERSON NO. 38 MINE.

Sample.—Bituminous coal; McAlester field; analyses Nos. 11592, 11594, 11670, 15188, and 15189 (p. 65).

Mine.—Alderson No. 38, a slope mine in T. 5 N., R. 15 E., 2 miles east of Alderson, on the Chicago, Rock Island & Pacific Railway.

Coal bed.—McAlester. Carboniferous (Allegheny) age, McAlester shale. Average thickness, 3 feet 5 inches; dip, 15° to 17° south. Roof, bluish-gray shale; floor, hard, brittle clay. Cover at points of sampling, 425 to 750 feet.

The bed was measured and sampled at three points in the mine by H. I. Smith on January 17 and 30, 1911, and at two points by C. S. Stevenson and W. T. Burgess on December 1, 1912.

Sample 11592 was taken in room 1, off seventh west entry, about 2,200 feet down the main slope. The sample represents 2 feet 9 inches of clear coal.

Sample 11594 was taken in room 4, off fifth east entry, about 2,300 feet down the main slope. The sample represents 3 feet 3 inches of clear coal.

Sample 11670 was taken in room 26, off fourth east entry, about 1,900 feet down the main slope. The sample represents 3 feet 2 inches of coal that contained a few thin seams (less than one-eighth inch thick) of bony coal.

Sample 15188 was taken at the face of sixth east entry, and represents 3 feet 6 inches of coal.

Sample 15189 was taken at the face of seventh west air course, and represents 3 feet 9 inches of coal.

Each sample represents the thickness of the bed at that point.

Notes.—The mine is worked by the room-and-pillar system. At the time of sampling the coal was shot from the solid with FF black powder. None of the coal was shipped as run-of-mine during 1910. About 33 per cent of the coal would pass through a 2½-inch screen. The coal was not washed. There were 5 loading tracks with a capacity of 14 empty and 14 loaded cars. The capacity was 200 tons a day.

CRAIG. BOLEN-DARNALL NO. 4 MINE.

Sample.—Bituminous coal; McAlester field; analyses Nos. 15104 and 15105 (p. 65).

Mine.—Bolen-Darnall No. 4, a slope mine in Craig, on the Ardmore branch of the Chicago, Rock Island & Pacific Railway.

Coal bed.—Lower Hartshorne (?). Carboniferous (Allegheny) age, McAlester shale. Thickness, fairly uniform, average about 3 feet. Roof, bluish sandy shale, soft in places. Floor, hard rough clay, which becomes soft when exposed to the action of water. Cover at points of sampling, about 700 feet.

The bed was measured and sampled at two points in the mine by W. T. Burgess and C. S. Stevenson on November 12, 1912, as described below:

Sections of coal bed in Bolen-Darnall No. 4 mine.

Section	A		B	
Laboratory No	15104		15105	
Roof, gray shale.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	5½	3	½
"Sulphur" band	[a]0	1	..	..
Coal	1	4¾	..	..
Floor, fire clay.				
Thickness of bed	2	11¼	3	½
Thickness of coal sampled	2	10¼	3	½

[a] Not included in sample.

Section A (sample 15104) was measured at the face of eighth west entry, 3,000 feet northwest of the mouth of the slope.

Section B (sample 15105) was measured at the face of eighth east entry, 3,000 feet from the mouth of the slope.

Notes.—The mine is opened by a slope and is worked on the room-and-pillar system. The coal from this mine, like many others in this field, breaks easily into small pieces. In 1910 about 55 per cent of the coal passed through the screens. Only a small proportion was shipped as run-of-mine. The average daily output was about 150 tons.

McALESTER. BUSBY NO. 5 MINE.

Samples.—Bituminous coal; McAlester field; analyses Nos. 11569, 11586, 11587, 11588, 11589, 11590, and 11591 (p. 65).

Mine.—Busby No. 5, a shaft mine in sec. 32, T. 6 N., R. 15 E., 2 miles east of McAlester on the Chicago, Rock Island & Pacific Railway and the Missouri, Kansas & Texas Railway.

Coal bed.—McAlester. Carboniferous (Allegheny) age, McAlester shale. Average thickness, 4 feet; dip, 10° S. 15° W.; cleat, S. 30° E. Roof, bluish-gray shale 30 feet thick; floor, hard fire clay. Cover at points of sampling, 550 to 720 feet.

The bed was measured and sampled at six places in the mine by H. I. Smith on January 11 and 19, 1911.

Sample 11569 was taken in room 11, off third west entry, off the east slope, about 1,600 feet from the shaft. The sample represents 4 feet 2 inches of clear coal, the thickness of the bed.

Sample 11586 was taken at the face of second west entry, off the west slope, 300 feet beyond room 26. The sample represents 4 feet of coal that contained a ⅛-inch band of "sulphur" about 4 inches from the top.

Sample 11587 was taken at the face of third east entry, off the east slope, 120 feet beyond room 22. The sample represents the thickness of the bed, 4 feet 2 inches; the upper three-fourths of an inch was bony coal.

Sample 11588 was taken at the face of fourth east entry, off the west slope. The bed was 4 feet thick and contained ⅛-inch band of "sulphur," 8½ inches from the top. The top inch was bony coal. The sample represents the entire thickness of the bed.

Sample 11589 was taken at the face of fifth east entry, off the east slope, where the bed was 4 feet 3 inches thick. The sample represents 4 feet 1½ inches of coal. There was 1½ inches of bony coal above the part sampled.

Sample 11590 was taken at the face of third east entry, off the west slope, 200 feet beyond room 8. The sample represents the entire thickness of the bed. The bed was 4 feet thick, the top inch was bony coal, and there was a band of "sulphur" ⅛ inch thick, 10 inches from the top.

A composite sample was made by mixing face samples 11586 to 11590. The results of an ultimate analysis of this sample are given under Laboratory No. 11591.

Notes.—The mine is worked by the room-and-pillar system. At the time of sampling the coal was shot from the solid with FF black powder. Only a small proportion of the coal was sold as run-of-mine. About 65 per cent of the coal going to the screens would pass through a 2-inch screen. Pickers were employed on the cars and at the shaking screens. There was no washery. The four loading tracks had a capacity of 20 empty and 20 loaded cars. The output was 600 tons a day; the unmined area tributary to this opening, 200 acres.

OREGON.

COOS COUNTY.

WEST FORK. FLANIGAN PROSPECT.

Sample.—Bituminous coal; Eden Ridge field; analysis No. 14828 (p. 66).

Mine.—Flanigan prospect, in NE. ¼ sec. 28, T. 32 S., R. 11 W., 36 miles southwest of West Fork (Dothan post office), on the Southern Pacific Railway.

Coal bed.—Anderson. Tertiary (Eocene) age, Arago formation.

The bed was measured and sampled by M. R. Campbell on September 24, 1912, as described below:

Section of coal bed in Flanigan prospect.

Laboratory No.	14828	
Roof, shale.	*Ft.*	*in.*
Coal	0	3½
Bone	0	2
Coal	1	6
Coal, bony	0	5
Clay, white	[a] 0	2
Coal	0	8
Bone	[a] 0	2½
Coal	0	6
Bone	0	3
Coal, soft, weathered (not mined)	1	4
Floor, clay.		
Thickness of bed	5	6
Thickness of coal sampled	5	1½

[a] Not included in sample.

Notes.—The sample was taken in the entry, 125 feet from the mouth of the entry, and represents wet coal.

WEST FORK. HAMMOND PROSPECT.

Sample.—Bituminous coal; Eden Ridge field; analysis No. 14831 (p. 66).

Mine.—Hammond prospect, in SW. ¼ NE. ¼ sec. 29, T. 32 S., R. 11 W., 37 miles southwest of West Fork (Dothan post office), on the Southern Pacific Railway.

Coal bed.—Carter. Tertiary (Eocene) age, Arago formation.

The bed was measured and sampled at one point in the mine by M. R. Campbell on September 26, 1912, as described below:

Section of coal bed in Hammond prospect.

Laboratory No.	14831	
Roof, shale and coal.	*Ft.*	*in.*
Coal	0	10½
Bone and coal	[a]0	8
Coal	0	7
Bone and coal	[a]0	11
Coal	1	1
Bone	[a]0	2
Coal	1	8½
Floor, clay.		
Thickness of bed	6	0
Thickness of coal sampled	4	3

[a] Not included in sample.

Notes.—The sample was collected in the entry, 100 feet from the drift mouth, and represents dry coal.

WEST FORK. HILLIS PROSPECT.

Sample.—Bituminous coal; Eden Ridge field; analysis No. 14827 and 14830 (p. 66).

Mine.—Hillis prospect, in SE. ¼ NE. ¼ sec. 32, T. 32 S., R. 11 W., 36 miles southwest of West Fork (Dothan post office), on the Southern Pacific Railway.

Coal bed.—Anderson. Tertiary (Eocene) age, Arago formation.

The bed was measured and sampled at two points in the prospect by M. R. Campbell on September 24, 1912, as described below:

Sections of coal bed in Hillis prospect.

Laboratory No.	14827		14830	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	[a]0	1½	0	1½
Bone	[a]0	¼	0	¼
Coal	[a]0	4¾	0	4¾
Bone	[a]0	¼	0	¼
Coal	[a]2	2¾	2	2¾
Clay, white	[a]0	3	[a]0	3
Coal	[a]0	2	[a]0	2
Bone and bony coal	[a]1	5	[a]1	5
Coal	1	3½	[a]1	3½
Coal, soft	[a]0	3	[a]0	3
Bone	[a]0	8	[a]0	8
Floor, clay.				
Thickness of bed	6	10	6	10
Thickness of coal sampled	1	3½	2	9½

[a] Not included in sample.

Notes.—Both samples were taken in the entry, 90 feet from the mouth. Sample 14827 represents wet coal and sample 14830 represents dry coal.

WEST FORK. VANDERPOOL PROSPECT.

Sample.—Bituminous coal; Eden Ridge field; analysis No. 14829 (p. 67).

Mine.—Vanderpool prospect, in NW. ¼ NE. ¼ sec. 30, T. 32 S., R. 11 W., 38 miles southwest of West Fork (Dothan post office), on the Southern Pacific Railway.

Coal bed.—Anderson. Tertiary (Eocene) age, Arago formation.

The bed was measured and sampled at one point in the mine by M. R. Campbell on September 27, 1912, as described below:

Section of coal bed in Vanderpool prospect.

Laboratory No	14829	
Roof, shale.	*Ft.*	*in.*
Coal	[a]0	3
Bone	[a]0	7
Coal	0	2
Bone	0	1
Coal, good	1	7
Bone	[a]0	3
Clay, white	[a]0	5
Coal, hard	[a]0	9
Coal, soft, laminated	[a]1	9½
Floor, clay.		
Thickness of bed	5	10½
Thickness of coal sampled	1	10

[a] Not included in sample.

Notes.—The sample was taken in the entry, 217 feet from the mouth of the entry, and represents dry coal.

MALHEUR COUNTY.

NYSSA. HONOLULU PROSPECT.

Sample.—Lignite; analysis No. 12585 (p. 67).

Mine.—Honolulu prospect, in NE. ¼ sec. 2, T. 22 S., R. 44 E. (unsurveyed), about 25 miles southwest of Nyssa, on the Oregon Short Line Railroad.

Lignite bed.—Not named. Tertiary (Miocene?) age, Payette formation. Thickness, 2 feet 4 inches; variable. Dip, about 6° SW.; strike, N. 37° W. The bed is irregular in character. Roof, shale; floor, clay. At an opening about 1,000 feet north of the point of sampling the bed is cut by a sill of basalt and in places is metamorphosed to high-grade coal, which occurs in small lenses.

The bed was measured and sampled at one point in the mine by C. F. Bowen on July 24, 1911, as shown below:

Section of lignite bed at Honolulu prospect.

Laboratory No	12585	
Roof, shale.	*Ft.*	*in.*
Lignite, fair	0	8
Lignite, bony	[a]1	8
Floor, clay, sandy.		
Thickness of bed	2	4
Thickness of lignite sampled	0	8

[a] Not included in sample.

The sample was taken at the mouth of the slope; the lignite was more or less weathered.

PENNSYLVANIA.

ALLEGHENY COUNTY.

BRUCETON. EXPERIMENTAL MINE.

Sample.—Bituminous coal; Pittsburgh field; analyses Nos. 11983, 11984, 11985, 11986, 12251, 12262, 12264, 12274, 12557, 12569, 12572, 12577, 14797, and 14798 (p. 67).

Mine.—The Experimental mine of the United States Bureau of Mines is a drift opening in the Pittsburgh district, about 1 mile south of Bruceton, near the Baltimore & Ohio Railroad siding at Wallace Station.

Coal bed.—Pittsburgh. Carboniferous age, Monongahela formation. Average thickness at mine, about 5 feet 3 inches. The roof is a "draw slate" 10 to 12 inches thick, above which there are thin strata of coal and shale, overlain by shale. The coal above the "draw slate" adheres to the roof shale. The floor is a fire clay. The bed is practically horizontal and the cover at the points where the samples were taken is 15 to 140 feet thick.

The bed was measured and sampled at three points in the mine by L. M. Jones on April 4, 1911, and at nine points by H. I. Smith between May, 1911, and October, 1912, as described below:

Sections of coal bed at Experimental mine.

[SECTIONS A TO F.]

Section	A	B	C	D	E	F
Laboratory No.	11983	11984	11985	12251	12262	12264
Roof, "draw slate."	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	1 8	3 1½	2 3	3 3	2 8	2 7
Shale	.. ..	a 0 ¼	.. ..	.. ..	.. ..	.. ..
Soft dirt and "sulphur"	0 ¼	.. ..	.. ..	.. ..	.. ..	a 0 ½
Coal	0 ¾	0 2¾	.. ..	.. ..	.. ..	.. ..
Fire clay	.. ..	.. ..	.. ..	a 0 ½	.. ..	.. ..
Soft dirt and "sulphur"	0 ¼	.. ..	0 ⅛	.. ..	Streak.	.. ..
Coal	1 5¼	.. ..	0 6⅝	0 3	0 6½	0 3
Shale	0 ⅛	a 0 1	a 0 ½	a 0 ½	a 0 ¾	a 0 ½
Coal	0 3¼	1 1½	0 2½	0 5⅝	0 3	0 3
Shale	a 0 ⅝	.. ..	a 0 1¼	.. ..	a 0 ¾	a 0 ¾
Coal	.. ..	.. ..	0 3½	.. ..	.. ..	0 3
"Sulphur" band	.. ..	a 0 1/16	a 0 ½	.. ..	.. ..	a 0 ¾
Coal	2 2	0 9¼	1 9	1 11	2 5	1 10½
Dirty coal and shale	a 0 2½	a 0 3	.. ..	.. ..	.. ..	.. ..
Floor, fire clay.						
Thickness of bed	5 11	5 7 1/16	5 3	5 11⅝	6 0	5 5
Thickness of coal sampled	5 7⅞	5 3	5 ¾	5 10⅝	5 10½	5 2½

a Not included in sample.

[SECTIONS G TO L.]

Section	G	H	I	J	K	L
Laboratory No.	12557	12569	12572	12577	14797	14798
Roof, "draw slate."	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	1 5	1 4½	1 10½	1 3	0 8½	0 4½
"Sulphur" or bone	0 ¼	0 ½	0 ½	0 ½	a 0 1¼	0 ¼
Coal	1 5¾	1 6	0 1	0 2½	0 7½	0 5½
"Sulphur" and shale	.. ..	.. ..	0 ¼	Streak.	a 0 ½	a 0 ½
Coal	.. ..	.. ..	0 10½	0 7½	0 11	1 5
"Sulphur"	.. ..	.. ..	.. ..	Streak.	0 ¼	Streak.
Coal	.. ..	.. ..	.. ..	0 8	0 10	0 8
Shale binder	0 ½	a 0 ½	Streak.	0 ½	a 0 ¾	Streak.
Coal	0 3	0 2½	0 2¾	0 3	.. ..	0 3¼
Shale binder	.. ..	a 0 ¾	a 0 1	.. ..	.. ..	a 0 1
Bone	0 1	.. ..	.. ..	0 1	.. ..	.. ..
Coal	0 3	.. ..	0 1	0 1	0 10¼	0 3
"Sulphur" balls	0 1	.. ..	0 1½	0 1	.. ..	0 1¼
Coal	1 4½	0 11	0 7½	0 10	.. ..	.. ..
Mother coal	0 ½	0 ½	.. ..	0 ¼	0 ¼	.. ..
Coal	0 1	.. ..	.. ..	0 2½	1 3¼	1 8
Bony coal	0 ½	.. ..	0 ½	Streak.	.. ..	.. ..
Coal	0 2	0 8½	0 10	0 8	.. ..	.. ..
Dirty coal	.. ..	a 0 4	a 0 3	a 0 2	.. ..	a 0 3
Floor, underclay.						
Thickness of bed	5 4	5 2¾	5 2	5 2¾	5 5½	5 7¼
Thickness of coal sampled	5 4	4 9½	4 10	5 ¾	5 3	5 2¾

a Not included in sample.

Section A (sample 11983) was measured on the rib of the first crosscut, about 200 feet from the mouth of the air course.

Section B (sample 11984) was measured on the rib near the face of the air course, 390 feet from the mouth of the air course.

Section C (sample 11985) was measured at the face of the main entry, 345 feet from the mouth of the main entry.

A composite sample was made by mixing samples 11983, 11984, and 11985. The results of an analysis of this sample are shown under laboratory No. 11986.

Section D (sample 12251) was measured in the air course, about 75 feet from the outcrop at the mouth of the air course.

Section E (sample 12262) was measured on the left side of the air course, about 150 feet from the mouth of the air course.

Section F (sample 12264) was measured on the left side of the air course, about 430 feet from the outcrop at the mouth of the air course.

Section G (sample 12557) was measured on the right side of the main entry, 680 feet in by the outcrop.

Section H (sample 12569) was measured on the right side of the main entry, 530 feet in by the outcrop.

Section I (sample 12572) was measured on the right side of the main entry, 580 feet in by the outcrop.

Section J (sample 12577) was measured on the right side of the main entry, 630 feet in by the outcrop.

Section K (sample 14797) was measured in the crosscut off the main entry, at station E–1250, 3 feet from the main entry.

Section L (sample 14798) was measured in the main entry at station E–1270.

Notes.—The Experimental mine is opened by parallel air course and entry, which at the time the last samples were taken, were in about 1,270 feet. The mine is used for experiments made in connection with investigations of the Bureau of Mines. No attempt is made to produce coal to supply a regular demand. All the coal is shipped as run-of-mine.

CREIGHTON. HITE MINE.

Sample.—Bituminous coal; Beaver field; analyses Nos. 11130, 11132, and 11133 (p. 68).

Mine.—Hite; a drift mine in the Pittsburgh district, at Creighton, on the Western Pennsylvania branch of the Pennsylvania Railroad. The mine opening is about one-eighth of a mile from the Allegheny River but none of the coal is shipped by water.

Coal beds.—Upper Freeport or E and Lower Freeport or D, of the State Geological Survey. Carboniferous age, Allegheny formation. At this mine the beds are separated by less than 1 foot of bony coal. Average thickness about 5 feet 10 inches. The coal outcrops on a hillside and lies practically level; slight variations of dip are due to local rolls. The face cleavage strikes approximately northeast and the butt cleavage about southwest. The coal is of good quality. The immediate roof, a hard gray shale, is overlain by about 1 foot 6 inches of cannel coal. The floor is a smooth hard fire clay. The bed was measured and sampled at two points in the mine by L. M. Jones and J. J. Rutledge on October 28, 1910, as described below:

Sections of coal bed in Hite mine.

Section	A		B	
Laboratory No.	11130		11132	
Roof, cannel coal.	*Ft.*	*in.*	*Ft.*	*in.*
Coal, clean and hard	..	..	0	4
Mother coal	..	..	0	½
Coal	2	4	2	2
Coal, bony	[a] 0	11	[a] 0	8
Coal	2	7	2	8½
Floor, fire clay.				
Thickness of bed	5	10	5	11
Thickness of coal sampled	4	11	5	3

[a] Not included in sample.

Section A (sample 11130) was taken at the face of room 17, off first butt entry, about 8,000 feet west of the mine mouth.

Section B (sample 11132) was taken at the face of room 18, off third butt entry 9,000 feet from the mine mouth.

A grab sample of the cannel coal, 1 foot 3 inches thick, from above the coal mined was taken at the intersection of the line entry and main entry 1, for approximate analysis of which see Laboratory No. 11133.

Notes.—The coal is mined by the room-and-pillar system. At the time of sampling the tipple had screens, and there was trackage for 25 or 30 cars. The daily output was about 350 tons. The mine is old, having been in operation about 30 or 40 years.

OAK STATION. OAK MINE.

Sample.—Bituminous coal; Pittsburgh field; analysis No. 10845 (p. 68).

Mine.—Oak, a drift mine at Oak Station in the Pittsburgh district, on the Castle Shannon (narrow gage) railroad, a short distance south of Pittsburgh.

Coal bed.—Pittsburgh. Carboniferous age, Allegheny formation. Average thickness, 5 feet 4 inches. The roof, a "draw slate" about 12 inches thick, is overlain with 12 inches of bony coal with a sandstone cap rock. The floor at the point of sampling is limestone. The coal lies practically level. Cover at point of sampling, 85 feet. The bed was measured and sampled at one point in the mine by L. M. Jones on September 9, 1910, as described below:

Section of coal bed in the Oak mine.

Laboratory No.	10845	
	Ft.	*in.*
Roof, "draw slate."		
Coal	0	7
"Sulphur" band	[a] 0	½
Coal	2	11
"Sulphur" band	[a] 0	½
Coal	0	1½
"Sulphur" band	[a] 0	½
Coal	1	7½
Floor, limestone.		
Thickness of bed	5	4½
Thickness of coal sampled	5	3

[a] Not included in sample.

This section (sample 10845) was measured in room 10, off left entry 13, 5,300 feet north of the mine mouth.

Notes.—The coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut with punching machines and shot down with FF black powder. The mine was equipped for producing 400 tons a day; about 60 per cent of the output was from advance workings, the remainder from pillars. When the pillars were robbed the recovery amounted to about 90 per cent of the coal. All of the coal was screened over 1½-inch bar screens; the screenings amounted to about 30 per cent of the output.

BEDFORD COUNTY.

HOPEWELL. CAMBRIA NO. 3 MINE.

Sample.—Semibituminous coal; Broadtop field; analyses Nos. 15055, 15056, 15057, 15058, 15059, and 15060 (pp. 68, 69).

Mine.—Cambria No. 3, a shaft mine 2 miles east of Hopewell, on the Huntingdon & Broadtop Railroad.

Coal bed.—Upper Freeport, locally known as the Kelley. Carboniferous age, Allegheny formation. Thickness, 3 to 5 feet, average 4 feet. There is about 6 inches of "draw slate" above the coal. Roof, smooth shale; floor, smooth brittle underclay, particles of which sometimes were mixed with the coal in loading. Cover at points of sampling, 225 feet.

The bed was measured and sampled at five points in the mine by H. I. Smith on November 8 and 9, 1912, as described below.

Sections of coal bed in Cambria No. 3 mine.

Section	A	B	C	D	E
Laboratory No.	15055	15056	15057	15058	15059
Roof, shale and "draw slate."	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	0 4½	[a]0 4½	1 4	1 4	1 4½
Shale	0 ¼	[a]0 ½	.. ..	.. ..	.. ..
Coal	0 6	1 7	.. ..	.. ..	.. ..
Mother coal and "sulphur"	0 ¼	.. ..	.. ..	.. ..	.. ..
Coal	0 2¼	.. ..	.. ..	.. ..	.. ..
Coal, hard, gray	0 3½	0 2	0 4	0 3½	.. ..
Coal, broken	.. ..	.. ..	.. ..	.. ..	0 6½
Coal	0 4	0 8	0 5	0 3½	.. ..
Coal and broken shale mixed	0 2½	.. ..	0 3½	0 5	.. ..
Coal, bony	.. ..	0 3	.. ..	.. ..	.. ..
Coal, soft, brittle	.. ..	0 2¼	.. ..	.. ..	.. ..
Coal	0 8½	.. ..	0 8½	1 2	0 8½
"Sulphur"	0 ⅛	.. ..	Streak.	.. ..	Streak.
Coal, bony	.. ..	.. ..	.. ..	0 ½	.. ..
Coal	0 9½	1 5	0 4	0 5	1 0
Shale	[a]0 1	.. ..	0 2	.. ..	.. ..
Coal	0 4	.. ..	0 4½	0 0	0 0
Floor, clay.					
Thickness of bed	3 10⅜	4 8¼	3 11½	3 11½	3 7½
Thickness of coal sampled	3 9⅜	4 3¼	3 11½	3 11½	3 7½

[a] Not included in sample.

Section A (sample 15055) was measured at the face of room 7 off second heading, 7,000 feet northeast of the shaft.

Section B (sample 15056) was measured near the face, 600 feet to the right from top of third plane, 5,100 feet northeast of the shaft.

Section C (sample 15057) was measured at the face of second heading, 7,300 feet northeast of the shaft.

Section D (sample 15058) was measured at the face of third heading, 7,400 feet northeast of the shaft.

Section E (sample 15059) was measured at a point 200 feet from face of fourth heading, 7,000 feet northeast from shaft.

A composite sample was made by combining face samples 15055, 15056, 15058, and 15059. The results of an ultimate analysis of this sample is given under laboratory No. 15060.

Notes.—The room-and-pillar system of mining is employed. At the time of sampling the coal was undercut by hand and shot down by FFF black powder. All coal was shipped as run-of-mine. Pickers were employed on the cars in loading. There were two tracks with a capacity of 32 empty cars and 32 loaded cars.

CAMBRIA COUNTY.

BARNESBORO. ALLPORT MINE.

Sample.—Bituminous coal; Windber field; analyses Nos. 12221, W19738, W19739, W19740, W19741, W19742, and W19743 (p. 69).

Mine.—Allport, a drift mine in the Clearfield district, 1 mile east of Barnesboro, on the Pennsylvania Railroad.

Coal beds.—Lower Freeport or D and Upper Freeport or E. Carboniferous age, Allegheny formation. Average thickness of D bed, 4 feet 5 inches; of E bed, 3 feet 7 inches. Each bed has a roof of smooth shale and a floor of smooth hard fire clay.

The D bed was measured and sampled at three points in the mine and the E bed at three points by P. M. Riefkin on April 26, 1911, as described below:

Sections of D bed in Allport mine.

Section	A		B		C	
Laboratory No.	W19740		W19741		W19742	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Cannel coal	[a]0	1¼	0	1½	0	2¼
Coal	0	10¼	1	4	0	10
Mother coal	0	¼	0	¼	..	..
Coal	0	10¼	0	9	..	..
"Sulphur" and mother coal	0	⅛	0	1	Streak.	
Coal	0	6¼	0	4½	1	4½
Hard coal	0	1	0	¾	0	1
Coal	0	9¼	1	1¼	1	1
Shale	[a]0	1	[a]0	¾	[a]0	1½
Coal	1	½	0	11½	0	11¾
Floor, hard fire clay ("bastard slate").						
Thickness of bed	4	4⅛	4	10½	4	8
Thickness of coal sampled	4	1⅞	4	9¾	4	6½

[a] Not included in sample.

Section A (sample W19740) was measured at the face of left entry2, off heading 9, off the main heading, 6,250 feet southeast of the mine mouth.

Section B (sample W19741) was measured on the rib in room 7, left entry 2, off right entry 9, off the main heading, 6,250 feet southeast of the mine mouth.

Section C (sample W19742) was measured at the face of right entry 9, off the main heading, 6,950 feet southwest of the mine mouth.

Sections of E bed in Allport mine.

Section	A		B		C	
Laboratory No.	W19738		W19739		W19743	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	[a]0	½	..	..	..	..
Coal	0	11	1	1	1	2
"Sulphur" and mother coal	[a]0	⅝	0	½	0	¼
Coal	1	3½	0	6	1	3½
Mother coal	Streak.		Streak.		..	..
Coal, hard	0	5	1	1	0	3
Coal, bony	[a]0	1½	[a]0	1	0	¼
Coal	0	10½	0	9½	0	10½
Floor, hard fire clay ("bastard slate").						
Thickness of bed	3	8⅝	3	7	3	7½
Thickness of coal sampled	3	6	3	6	3	7½

[a] Not included in sample.

Section A (sample W19738) was measured at the face of third right entry, off the main heading, 3,700 feet southeast of the mine mouth.

Section B (sample W19739) was measured at the face of the main slant off the main entry, 3,700 feet southeast of the mine mouth.

Section C (sample W19743) was measured 600 feet in on the second panel from the main slant off the main entry, 2,800 feet south of the mine mouth.

A composite sample was made by combining face samples W19740, W19741, and W19742. The results of an ultimate analysis of this sample are shown under laboratory No. 12221.

Notes.—At the time of sampling the coal was undercut with compressed air punchers and broken down with FFFF black powder. All the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track at the tipple with a capacity of 18 empty and 19 loaded cars. The mine had a capacity of 900 tons a day with an average daily output of 700 tons, 60 per cent of which was from advance workings. It was expected that the output would be increased to 900 tons a day. There was approximately 235 acres of unmined coal in the D bed and 370 acres in the E bed. The coal from both beds was loaded over the same tipple.

BARNESBORO. CYMBRIA MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. W20394, W20395, W20444, W20445, W20446, and 12372, (p. 69).

Mine.—Cymbria, a shaft mine in the Clearfield district, 1½ miles east of Barnesboro, on the Walnut Run branch of the Pennsylvania Railroad.

Coal bed.—Lower Kittanning, Miller, or B. Carboniferous age, Allegheny formation. Average thickness, 2 feet 10 inches. Roof, shale 3 to 18 feet thick, overlain by bony coal and sandstone; floor, smooth hard fire clay.

The bed was measured and sampled at five points in the mine by E. W. Miller on June 8, 1911, as described below:

Sections of coal bed in Cymbria mine.

Section	A		B		C		D		E	
Laboratory No	W20394		W20395		W20444		W20445		W20446	
Roof, sandstone, bony coal, and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	2½	2	0	0	4½	1	0	2	11
Mother coal, soft	..	..	0	½	..	..	..	..	..	..
"Sulphur"	0	¼	..	..	*a* 0	⅛	0	¼	..	..
Coal	2	½	0	7	0	6¼	0	7¼	..	..
"Sulphur"	*a* 0	½	..	..	0	¼	0	¼	..	..
Coal	0	6½	..	..	0	7¼	0	9¼	..	..
"Sulphur"	..	..	..	..	0	¼	..	..	..	..
Coal	..	..	..	..	0	10	..	..	..	..
Floor, fire clay.										
Thickness of bed	2	10¼	2	7½	2	4⅝	2	5	2	11
Thickness of coal sampled.	2	9¾	2	7½	2	4½	2	5	2	11

a Not included in sample.

Section A (sample W20394) was measured at the face of second right entry, about 1,000 feet southwest of the shaft.

Section B (sample W20395) was measured near the face of room 18, off second left entry, about 1,200 feet northeast of the shaft.

Section C (sample W20444) was measured at the face of the main heading, 1,900 feet from the shaft.

Section D (sample W20445) was measured at the face of third left entry, about 1,400 feet northeast of the shaft.

Section E (sample W20446) was measured at the face of third right entry, about 1,200 feet southeast of the shaft.

A composite sample was made by combining face samples W20394, W20395, W20444, W20445, and W20446. The results of an ultimate analysis of this sample are given under laboratory No. 12372.

Notes.—At the time of sampling the coal was undercut with machines and shot down with FFFF black powder. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track at the tipple with a capacity of 48 cars. The mine had a capacity of 300 tons a day and an average daily output of 150 tons, all from advance workings. There were between 600 and 700 acres of unmined coal.

BARNESBORO. CYMBRIA NO. 1 MINE.

Sample.—Bituminous coal; Windber field; analyses Nos. 12162, 12163, 12164, 12165, 12170, and 12171 (p. 70).

Mine.—Cymbria No. 1, a slope mine in the Clearfield district, 2 miles east of Barnesboro, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Lower Freeport or D; Carboniferous age, Allegheny formation. Average thickness, 3 feet 6 inches. Roof, smooth dark shale; floor, hard smooth clay. Material from the roof and floor did not become mixed with the coal in mining.

The bed was measured and sampled at three points in the mine on April 28, 1911, and at two points on May 1, 1911, by H. I. Smith, as described below:

Sections of coal bed in Cymbria No. 1 mine.

Section	A		B		C		D		E	
Laboratory No.	12162		12163		12164		12170		12171	
Roof, sandstone and dark shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Bony shale	[a]0	2	[a]0	1	[a]0	2½	[a]0	2	[a]0	2
Coal	1	0	0	8½	2	0	1	10	3	2¼
Mother coal and "sulphur"	..	..	[a]0	¾	..	..	0	½	..	..
Coal	0	10	1	0	..	..	..	..	..	..
"Sulphur" and mother coal	[a]0	½	[a]0	¼	..	..	..	..	..	..
Coal	0	5	1	3	1	4½	0	3	..	..
Coal, bony	0	½	..	..	..	..	0	1	..	..
Coal	0	10½	..	..	..	..	1	0	..	..
Coal	..	..	[a]0	1½	..	..	..	..	..	..
"Mining slate"	[a]0	1	[a]0	1¾	[a]0	¾	[a]0	1½	[a]0	¾
Coal	[a]0	½	..	..	..	..	..	..	..	..
Coal	0	11	0	7	0	11⅜	0	8	0	10¼
Floor, hard clay.										
Thickness of bed	4	5	3	11¾	4	7⅛	4	1⅞	4	3¼
Thickness of coal sampled	4	1	3	6½	4	3⅞	3	10⅜	4	½

[a] Not included in sample.

Section A (sample 12162) was measured at the face of the first crosscut off left entry 2, 5,500 feet from the mine mouth.

Section B (sample 12163) was measured at the face of the eighth right off left entry 2, about 4,000 feet from the mine mouth.

Section C (sample 12164) was measured at the face of the first left off left entry 2, 4,000 feet from the mine mouth.

Section D (sample 12170) was measured at the face of the third right off the second main entry, about 3,200 feet northeast of the mine mouth.

Section E (sample 12171) was measured at the face of the second left off the main heading, 2,900 feet northeast of the drift mouth.

A composite sample was made by combining face samples 12162, 12163, 12164, 12170, and 12171. The results of an ultimate analysis of this sample are given under laboratory No. 12165.

Notes.—At the time of sampling the coal was undercut by chain machines and also by hand. The undercut coal was broken down with FFFF black powder. A permissible explosive was used for brushing the roof and floor. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. The tipple had two loading tracks, with a capacity of 25 empty and 35 loaded cars. The coal from Cymbria No. 1 and Cymbria No. 2½ mines went to the same tipple, and the combined capacity of the two mines was 800 tons a day with an average daily output of 600 tons, 66 per cent of which was from advance workings and the remainder from pillars. There was approximately 175 acres of unmined coal.

BARNESBORO. CYMBRIA NO. 2½ MINE.

Sample.—Bituminous coal; Windber field; analyses Nos. 12158, 12159, 12160, 12161, 12172, and 12173 (pp. 70, 71).

Mine.—Cymbria No. 2½, a slope mine in the Clearfield district, 2 miles east of Barnesboro, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Upper Freeport or E. Carboniferous age, Allegheny formation. Average thickness, 3 feet 6 inches. Roof, a dark, smooth shale, above which is a sandstone cap rock; floor, a smooth, hard clay.

The bed was measured and sampled at three points in the mine by H. I. Smith on April 28, 1911, and at two points on May 1, 1911, as described on the following page.

Sections of coal bed in Cymbria No. 2½ mine.

Section	A	B	C	D	E
Laboratory No.	12158	12159	12160	12172	12173
Roof, sandstone, shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	.. ..	.. ..	0 2	.. ..	.. ..
Coal, bony	.. ..	.. ..	0 ⅛	.. ..	.. ..
Coal	.. ..	.. ..	0 4	.. ..	.. ..
Coal, bony	[a]0 ½	.. ..	0 ⅛	[a]0 ½	[a]0 ¼
Coal (brittle)	1 5	1 9	0 10	1 6	0 3¾
"Sulphur" and mother coal	0 ½	.. ..	[a]0 ¼	0 ⅛	0 ¼
Coal (firm)	1 2¼	0 11	1 1½	1 ½	1 7¼
Mother coal	.. ..	.. ..	.. ..	.. ..	0 ¼
Coal	.. ..	.. ..	.. ..	.. ..	0 10
Coal	.. ..	[a]0 1	[a]0 ½	.. ..	.. ..
"Slate binder"	[a]0 1	[a]0 ½	[a]0 1¼	[a]0 2	[a]0 1
Coal	[a]0 1½	[a]0 1	[a]0 ¾	.. ..	.. ..
Coal	0 9	0 9	0 11	0 10	0 10½
Floor, hard fire clay.					
Thickness of bed	3 7¾	3 7½	3 7¾	3 7⅛	3 9¼
Thickness of coal sampled	3 4¾	3 5	3 4¾	3 4⅝	3 8

[a] Not included in sample.

Section A (sample 12158) was measured at the face of left entry 2, about 2,000 feet from the mine mouth.

Section B (sample 12159) was measured at the face of room 12 off the main heading, 2,200 feet from the mine mouth.

Section C (sample 12160) was measured at the face of the third left main heading.

Section D (sample 12172) was measured at the face of room 6 off the main heading, 1,500 feet southwest of the mine mouth.

Section E (sample 12173) was measured at the face of room 8 off the second left heading, 1,200 feet southwest of the mine mouth.

A composite sample was made by combining face samples 12158, 12159, 12160, 12172, and 12173. The results of an ultimate analysis of this sample are shown under laboratory No. 12161.

Notes.—As Cymbria No. 1 mine and Cymbria No. 2½ mine use the same tipple, the notes on Cymbria No. 1 are applicable to Cymbria No. 2½.

BARNESBORO. EMPIRE A MINE.

Sample.—Bituminous coal; Windber field; analyses Nos. 12212, W19643, W19644, W19645, W19646, W19647, and W19648 (p. 71).

Mine.—Empire A, a drift mine in the Clearfield district, three-fourths of a mile west of Barnesboro, on the New York Central and the Pennsylvania Railroad.

Coal bed.—Lower Freeport or D. Carboniferous age, Allegheny formation. Average thickness, 4 feet 4 inches. Roof, shale; between the shale and the coal is a stratum of cannel coal about 2 inches thick. Floor, hard smooth fire clay. Material from the roof and floor did not become mixed with the coal in mining.

The bed was measured and sampled at six points in the mine by P. M. Riefkin on April 28, 1911, as described below:

Sections of coal bed in Empire A mine.

Section	A	B	C	D	E	F
Laboratory No.	W19643	W19644	W19645	W19646	W19647	W19648
Roof, shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Cannel coal	[a]0 2	[a]0 2	[a]0 1¼	[a]0 2	[a]0 1	[a]0 2
Coal	0 7	0 3½	0 7	0 8	0 6¼	0 7
Coal, bony, and shale	[a]0 1½	.. ..	.. ..	[a]0 ½	[a]0 ⅜	[a]0 1
Coal, hard, gray	.. ..	0 1	.. ..	.. ..	.. ..	.. ..
Coal	1 2	1 10	.. ..	1 1¾	0 5¼	1 1
Mother coal	0 ¼	0 ½	Streak.	0 ¼	Streak.	0 ½
Coal	0 8½	1 1	2 2	0 10	1 7½	1 10½
Mother coal	0 ¼	.. ..	0 ¼	0 ¼	0 ⅜	.. ..
Coal	0 9½	.. ..	0 7	0 7¾	0 2¾	.. ..
Shale	[a]0 1	[a]0 1¾	[a]0 ¾	[a]0 ⅞	[a]0 1	[a]0 1
Coal	0 8½	0 11	0 10	0 8½	0 8	0 11½
Floor, hard fire clay.						
Thickness of bed	4 4¾	4 6¾	4 4¼	4 3⅞	3 9⅛	4 10½
Thickness of coal sampled	4 ¼	4 3	4 2¼	4 ½	3 6⅜	4 6½

[a] Not included in sample.

Section A (sample W19643) was measured in room 1 off left entry 5 off new parallel heading, 14,000 feet northwest of the drift mouth.

Section B (sample W19644) was measured near the face of room 4 off old first left heading off the main entry, 2,000 feet northwest of the drift mouth.

Section C (sample W19645) was measured at the corner of the back heading on right entry 3 off the old parallel heading, 3,000 feet northeast of the drift mouth.

Section D (sample W19646) was measured 40 feet from the face of the parallel heading off the main entry, 9,000 feet northeast of the drift mouth.

Section E (sample W19647) was measured at the face of room 24 off left entry 14 off the main entry, 11,000 feet northwest of the drift mouth.

Section F (sample W19648) was measured at the face of room 17 off left entry 3 off the new parallel heading, 14,000 feet northwest of the drift mouth.

A composite sample was made by combining face samples W19643 to W19648, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 12212.

Notes.—At the time of sampling the coal was undercut by compressed-air punchers and broken down with FFFF black powder. There was one loading track at the tipple, and the entire production was shipped as run-of-mine. Pickers were employed on the car in loading. The mine had a capacity of about 1,000 tons a day, with an average daily output of 625 tons, 75 per cent of which was from advance workings. There were approximately 750 acres of unmined coal.

BEAVERDALE. PENNSYLVANIA NO. 15 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12174, 12175, 12176, 12177, 12178, and 12179 (p. 71).

Mine.—Pennsylvania No. 15, a shaft mine in the Clearfield district, at Beaverdale, on the South Fork branch of the Pennsylvania Railroad.

Coal bed.—Lower Kittanning or B; Carboniferous age, Allegheny formation. Thickness, 3 feet to 3 feet 8 inches; dip, 3° north. Roof, bony coal, 5 inches thick, with sandstone cap rock; floor, hard smooth fire clay. Material from the roof and floor did not mix with the coal in mining.

The bed was measured and sampled at five points in the mine by E. G. Borden on May 1, 1911, as described below:

Sections of coal bed in Pennsylvania No. 15 mine.

Section	A		B		C		D		E	
Laboratory No	12174		12175		12176		12177		12179	
Roof, sandstone, and bony coal.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in*
Coal	0	2	..	..	..	..	0	3¼	0	6½
Mother coal	0	½	..	..	..	..	..	..	..	..
Coal	1	7	..	..	..	..	..	..	..	..
Coal, gray	..	..	..	..	..	..	..	..	0	1½
"Sulphur"	Streak.		..	..	..	..	Streak.		..	..
Coal	1	1¼	2	9	3	4	3	½	2	5
Mother coal	0	¼	0	½	..	..	..	..	..	..
Coal	0	4	0	7	..	..	..	..	..	..
Floor, hard fire clay.										
Thickness of bed	3	3	3	4½	3	4	3	3¾	3	1
Thickness of coal sampled	3	3	3	4½	3	4	3	3¾	3	1

Section A (sample 12174) was measured near the face of the first slant off right heading 5 off the first dip off the main entry.

Section B (sample 12175) was measured at the face of the sump for the main dip, off left heading 4 off the first dip off the main entry.

Section C (sample 12176) was measured at the face of the first slant off right heading 4 off the first dip off the main entry.

Section D (sample 12177) was measured at the face of the first slant off right heading 3 off the first dip off the main entry.

Section E (sample 12179) is a pillar sample taken in room 23 off right heading 3 off the first dip off the main entry.

A composite sample was made by combining face samples 12174 to 12177, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 12178.

Notes.—The shaft is 624 feet deep. At the time of sampling the coal was undercut with compressed-air punching machines and broken down with black powder. Dynamite was used for brushing the roof and floor. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. There were two loading tracks at the tipple with a capacity of 25 empty and 22 loaded cars. The daily output was 450 tons, about 80 per cent of which was from advance workings.

BENS CREEK. BENS CREEK NO. 1 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12211, W19697, W19700, W19702, W19736, and W19737 (p. 72).

Mine.—Bens Creek No. 1, a shaft mine in Portage township, 1 mile east of Bens Creek, on the Bens Creek branch of the Pennsylvania Railroad.

Coal bed.—Lower Kittanning, Miller, or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet 6 inches. Roof, hard gray smooth shale; floor, smooth hard shale, underlain by fire clay.

The bed was measured and sampled at five points in the mine by E. W. Miller on May 2, 1911, as described below:

Sections of coal bed in Bens Creek No. 1 mine.

Section	A		B		C		D		E	
Laboratory No	W19697		W19700		W19702		W19736		W19737	
	Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Roof, shale and bony coal.										
Coal	2	0	1	9¾	..	..	..	..	0	5¼
"Sulphur" streak	0	⅛	0	⅛	..	..	..	..	0	⅛
Coal	1	6½	0	7½	3	9	3	4	3	2⅜
"Sulphur" streak	..	..	0	⅛	..	..	..	..	..	..
Coal	..	..	0	8	..	..	..	..	..	..
Floor, shale and fire clay.										
Thickness of bed	3	6⅝	3	1½	3	9	3	4	3	7¾
Thickness of coal sampled	3	6⅝	3	1½	3	9	3	4	3	7¾

Section A (sample W19697) was measured on the barrier pillar on No. 2 sidetrack, 100 feet from room 21 and 3,200 feet southwest of the old main heading.

Section B (sample W19700) was measured on the pillar on No. 1 sidetrack between the manway and room 1, 1,200 feet southwest of the shaft.

Section C (sample W19702) was measured on the main pillar of the old main heading, 900 feet southwest of the shaft.

Section D (sample W19736) was measured at the face of heading 6, off the first dip, 700 feet southeast of the shaft.

Section E (sample W19737) was measured on the pillar of heading 1, between rooms 27 and 28, 2,700 feet southwest of the shaft.

A composite sample was made from face samples W19697, W19700, W19702, and W19737. The results of an ultimate analysis of this sample are shown under laboratory No. 12211.

Notes.—At the time of sampling the coal was undercut by hand and shot down with black powder. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track at the tipple with a capacity of 24 cars. The mine had a capacity of 230 tons a day and an average daily output of about 225 tons, all from pillar workings.

BENS CREEK. PENNSYLVANIA NO. 2 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. W19695, W19696, W19698, W19699, W19701, and 12222 (pp. 72, 73).

Mine.—Pennsylvania No. 2, a slope mine 1 mile south of Bens Creek, on the Bens Creek branch of the Pennsylvania Railroad.

Coal bed.—Lower Kittanning, Miller, or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet 6 inches. Roof, shale overlain by sandstone cap rock; floor, smooth shale underlain by fire clay.

The bed was measured and sampled at five points in the mine by E. W. Miller on May 1, 1911, as described below:

Sections of coal bed in Pennsylvania No. 2 mine.

Section	A		B		C		D		E	
Laboratory No	W19695		W19696		W19698		W19699		W19701	
Roof, sandstone, shale, and bony coal.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	6	1	¾	1	3	1	0	1	2½
Mother coal, hard	0	¾	0	¾	0	¾	0	½	0	½
Coal	0	6¾	0	9½	2	4	2	4½	0	6
"Sulphur" streak	[a]0	½	0	¼	..	..	..	..	0	¼
Coal	0	8½	0	7¾	..	..	..	..	0	3¾
"Sulphur" streak	[a]0	¼	0	¼	..	..	..	..	0	½
Coal	0	8¾	0	5	..	..	..	..	1	5
"Sulphur" ball	[a]0	1	..	..	..	..	..	..	..	..
Coal	0	1	..	..	..	..	..	..	..	..
Shale	..	..	[a]0	½	..	..	..	..	..	..
Coal	..	..	0	3	..	..	..	..	..	..
Floor, shale and fire clay.										
Thickness of bed	3	9½	3	3¾	3	7¾	3	5	3	6½
Thickness of coal sampled	3	7¾	3	3¼	3	7¾	3	5	3	6½

[a] Not included in sample.

Section A (sample W19695) was measured at the face of heading 19, 4,100 feet northeast of the drift mouth.

Section B (sample W19696) was measured on the pillar between rooms 14 and 15 off main heading 4, 2,900 feet northwest of the drift mouth.

Section C (sample W19698) was measured on the pillar between rooms 2 and 3 off butt entry 17, 3,900 feet north of the drift mouth.

Section D (sample W19699) was measured on the pillar between rooms 1 and 2 off butt entry 16, 3,900 feet north of the drift mouth.

Section E (sample W19701) was measured on the pillar between rooms 21 and 22 off heading 5, 3,200 feet north of the drift mouth.

Notes.—At the time of sampling the coal was undercut by hand and shot down with black powder. About 90 per cent of the coal was shipped as run-of-mine; the remaining 10 per cent was passed over 8-foot bar screens with 4-inch openings. Pickers were employed on the cars in loading. There were two loading tracks with a capacity of 25 cars. The mine was equipped to produce 200 tons a day and had an average daily output of 175 tons, practically all of which was from pillar workings. The intent was to increase the output to about 200 tons.

CARROLLTOWN ROAD. LOGAN NO. 5 MINE.

Sample.—Bituminous coal; Windber field; analyses Nos. 12370, W20364, W20365, W20366, W20367, and W20368 (p. 73).

Mine.—Logan No. 5, a drift mine one-fourth of a mile west of Carrolltown Road, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Miller. Carboniferous age, Allegheny formation. Average thickness, 2 feet 8 inches. Roof, hard shale overlain by bony coal and sandstone; floor, smooth, hard fire clay.

The bed was measured and sampled at five points in the mine by E. W. Miller on June 7, 1911, as described on the following page.

Sections of coal bed in Logan No. 5 mine.

Section	A	B	C	D	E
Laboratory No.	W20364	W20365	W20366	W20367	W20368
Roof, sandstone, bony coal, and shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	0 6	0 6¼	1 10	0 6½	0 5
"Sulphur" streak	0 ⅛	0 $\frac{1}{16}$	0 ⅛	0 ¼	0 ⅛
Coal	0 7	1 ¾	0 8	1 9¾	0 7½
"Sulphur" streak	0 ⅛	0 ⅛	.. ..	.. ..	0 ⅛
Coal	1 5¼	0 10	.. ..	.. ..	1 5¾
Floor, fire clay.					
Thickness of bed	2 6½	2 5$\frac{3}{16}$	2 6⅛	2 4½	2 6½
Thickness of coal sampled	2 6½	2 5$\frac{3}{16}$	2 6⅛	2 4½	2 6½

Section A (sample W20364) was measured in the first main heading, 4,300 feet south of the mine mouth.

Section B (sample W20365) was measured at the face of right entry 2; 250 feet from the second main heading and 3,750 feet south of the mine mouth.

Section C (sample W20366) was measured at the face of room 8 off the air course of right entry 1, off the second main entry, 4,300 feet south of the mine mouth.

Section D (sample W20367) was measured at the face of the second main heading, 3,600 feet southeast of the mine mouth.

Section E (sample W20368) was measured at the face of the fourth left heading off the second main heading, 3,200 feet southeast of the mine mouth.

A composite sample was made by combining face samples W20364 to W20368, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 12370.

Notes.—At the time of sampling the coal was undercut both by hand and with machines, and shot down with black powder. There were no screens, the output being shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track at the tipple with a capacity of 50 cars. The mine had a capacity of 600 tons a day with an average daily output of about 350 tons, all of which was from advance workings. The intent was to increase the production to about 700 tons a day. There were approximately 700 acres of unmined coal.

COLVER. COLVER MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 13634 to 13643, inclusive (p. 73).

Mine.—Colver, a drift mine, in the Ebensburg district, at Colver, the terminus of the Cambria & Indiana Railroad.

Coal bed.—Known as the Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 4 feet. Roof, hard gray shale, with smooth surface; floor, shaly underclay with a smooth surface.

The bed was measured and sampled at nine points in the mine by L. M. Jones, on March 28 and 29, 1912, as described below:

Sections of coal bed in Colver mine.

Section	A	B	C	D	E	F
Laboratory No.	13634	13635	13636	13637 13643	13638 13642	13639 13641
Roof, gray shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal, bony	[a]0 3	[a]0 3	[a]0 3	[a]0 5	[a]0 4	[a]0 5
Coal, soft	1 1	2 8½	2 1¾	3 10	2 4½	0 9
Coal, dull	.. ..	.. ..	.. ..	.. ..	.. ..	0 ½
Shale	0 ¼	.. ..	.. ..	.. ..	0 ½	.. ..
Coal, soft	1 0	.. ..	.. ..	.. ..	.. ..	1 ¼
"Sulphur" band	[a]0 ¼	[a]0 ½	[a]0 ½	.. ..	.. ..	[a]0 ½
Coal	0 5¼	1 1½	1 8¾	.. ..	1 1½	1 8¾
Shale	0 ¼	.. ..	.. ..	.. ..	.. ..	.. ..
Coal	1 1	.. ..	.. ..	.. ..	.. ..	.. ..
Floor, shaly underclay.						
Thickness of bed	3 11	4 1½	4 2	4 3	3 10	4 1
Thickness of coal sampled	3 7¾	3 10	3 10½	3 10	3 6	3 7½

[a] Not included in sample.

Section A (sample 13634) was cut from the face of main entry A, 2,400 feet southwest of bore hole 25.

Section B (sample 13635) was cut from the face of entry A-2, 1,175 feet southeast from bore hole 25.

Section C (sample 13636) was cut from the face of entry A-3, 1,400 feet south of bore hole 25.

Section D (duplicate samples 13637 and 13643) was cut from the face of the main entry, 1,200 feet southeast of bore hole 25.

Section E (duplicate samples 13638 and 13642) was cut from the face of entry B-1), 725 feet southeast of bore hole 25, at the inner end of the rock tunnel.

Section F (duplicate samples 13639 and 13641) was cut from the face of entry A-1, 950 feet southeast of bore hole 25.

A composite sample was made by mixing samples 13634 to 13639 inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 13640.

Notes.—This mine was originally opened by a slope and shaft but at the time of sampling all the coal shipped was hauled to the surface through a rock tunnel 2,050 feet long.

The steel tipple was designed to handle 4,000 tons of coal a day. There were two loading tracks with a capacity of 80 empty cars. The tipple was equipped with a shaking screen having perforations 1¼ inches in diameter, but the entire output was shipped as run-of-mine.

The coal was undercut with chain cutting machines and with punching machines and shot down with FFFF black powder. The impurities that might be loaded with the coal were a bony coal at the top of the seam, which in places stuck to the roof, and a "sulphur" band below the middle. The roof and floor were hard and generally smooth and were not likely to get mixed with the coal.

The average daily output, the mine working double shifts, was 1,000 tons. There was 8,000 acres of coal land at this mine, the development of which was still in an early stage. The maximum day's run was 1,040 tons

DUNLO. HENRIETTE MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 14238 to 14246, inclusive (p. 74).

Mine.—Henriette, a shaft mine 1 mile south of Dunlo, on the South Fork branch of the Pennsylvania Railroad, with connections to the main line at South Fork.

Coal bed.—Miller or B. Carboniferous age, Allegheny formation. Thickness, about 3 feet 8 inches; dip, 7½° northwest. Roof, smooth shale 1 to 10 feet thick, above which is a sandstone cap rock. Floor, hard, smooth fire clay. The parting between the floor and the coal and the roof is distinct, so that neither the floor or the roof material got mixed with the coal in mining. Cover at points of sampling, 130 to 400 feet.

The bed was measured and sampled at five points in the mine by A. H. Fay, on June 27, 1912, as described below:

Sections of coal bed in Henriette mine.

Section	A		B		C		D		E	
Laboratory No.	14245 14238		14246 14239		14244 14240		14241		14242	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Top coal or bone	0	4	0	3	0	2	0	3½	..	..
Coal	3	5½	3	7	3	4½	3	4	3	8
Floor, shale and fire clay.										
Thickness of bed	3	9½	3	10	3	6½	3	7½	3	8

Section A (sample 14238 is the good coal and sample 14245 is the bone or top coal) was measured on the rib of room 2, right heading 1 off slope 2, about 700 feet north of shaft No. 2.

Section B (sample 14239 is the good coal and sample 14246 is the bone or top coal) was measured at the junction of the sixth slant and main heading, about 3,050 feet north of shaft No. 2.

Section C (sample 14240 is the good coal and sample 14244 is the bone or top coal) was measured on the pillar of sixth left entry near room 13, north of the rock fault, 4,350 feet northwest of shaft No. 2.

Section D (sample 14241) was measured on the pillar of sixth left heading, 450 feet north of the back heading of slope 2 and 4,150 feet northwest of shaft No. 2.

Section E (sample 14242) was measured on the rib between rooms 1 and 2, first level above the main entry, about 1,500 feet northwest of shaft No. 2.

A composite sample was made by combining samples 14238 to 14242, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14243.

Notes.—The mine is opened by a shaft 90 feet deep, from the bottom of which a slope extended a distance of about 7,000 feet. The coal is mined by the room-and-pillar system. That part of the mine from which these samples were taken had been practically abandoned for a number of years, and at the time of sampling no mining was being done there. It was proposed to pull the pillars later. Other property tributary to this slope was producing about 700 tons a day. All of the coal was shipped as run-of-mine. Pickers were employed on the car in loading. The coal is somewhat friable and does not yield extra large lumps. There were two loading tracks with a capacity of 25 to 30 cars.

EL MORA. PEERLESS NO. 1 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12134, 12135, 12136, 12137, 12138, and 12139 (p. 75).

Mine.—Peerless No. 1, a drift mine in Carroll Township, near El Mora on the Pennsylvania Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, about 3 feet; dip, about 3°. The immediate roof is a "draw slate" about 18 inches thick, above which there is a sandstone cap rock. Floor, hard rough fire clay. The bed was measured and sampled at five points in the mine by P. M. Riefkin on April 27, 1911, as described below:

Sections of coal bed in Peerless No. 1 mine.

Section	A		B		C		D		E	
Laboratory No.	12134		12135		12136		12137		12138	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, hard	0	4	0	5	..	..	0	5½	0	5½
Coal	0	4	..	..	..	..	0	8½	..	..
Bone	0	⅛	0	⅛	..	..	..	..	..	..
Mother coal	..	..	..	..	..	..	0	½	..	..
Coal	2	5	2	7	2	9	0	9	..	..
"Sulphur"	0	¼	..	..	[a] 0	½	0	¼	0	¼
Coal	0	2½	..	..	0	2	1	0	2	5½
Floor, fire clay.										
Thickness of bed	3	3⅞	3	⅛	2	11½	2	11¾	2	11¼
Thickness of coal sampled	3	3⅞	3	⅛	2	11	2	11¾	2	11¼

[a] Not included in sample.

Section A (sample 12134) was measured on a heading stump on left heading 3 off the main entry, about 1,700 feet northeast of the drift mouth.

Section B (sample 12135) was measured at the face of room 1, off right heading 8 off the main entry, 2,300 feet east of the drift mouth.

Section C (sample 12136) was measured at the face of left heading 7 off the main entry, about 2,800 feet northeast of the drift mouth.

Section D (sample 12137) was measured on the rib of room 1, off right heading 3 off the main heading, 1,600 feet southeast of the drift mouth.

Section E (sample 12138) was measured in heading 1 off the main entry, about 425 feet from the main entry.

A composite sample was made by combining face samples 12134 to 12138, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 12139.

Notes.—The mine is opened by entry; the cover at the opening is 40 feet. The room-and-pillar method of mining is used. At the time of sampling FFFF black powder was used for breaking the coal. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. The mine had a daily output of about 125 tons a day, 75 per cent of which was from advance workings and the remainder from pillars.

FIGART. FRICK NO. 2 MINE.

Sample.—Bituminous coal; Windber field; analyses Nos. 12366, W20433, W20434, W20474, W20475, and W20476 (p. 75).

Mine.—Frick No. 2, a drift mine 1¼ miles northeast of Figart, on the Pennsylvania Railroad.

Coal bed.—C or Middle Kittanning. Carboniferous age, Allegheny formation. Average thickness, 2 feet 7 inches. Roof, dark shale; floor, medium soft fire clay with a smooth surface. The B bed, or Lower Kittanning, is also worked at this mine, and the coal is delivered to the same tipple as is the coal from the C bed.

The C bed was measured and sampled at five points in the mine by H. I. Smith on June 10, 1911, as described below.

Sections of coal bed in Frick No. 2 mine.

Section	A		B		C		D		E	
Laboratory No	W20433		W20434		W20474		W20475		W20476	
Roof, dark shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	[a]0	8	[a]0	9½	[a]0	10½	[a]0	9	[a]0	10
Coal	1	3½	1	3½	0	4½	0	6	1	3
Coal, gray	..	..	..	..	0	1	0	½	..	..
Shale	0	½	..	..	..	..	..	..	..	..
Coal	0	1	..	..	0	8¾	0	10	..	..
Cannel coal	0	1	0	1	[a]0	3	0	½	[a]0	2
Coal, bony	[a]0	1	..	..	..	..	..	..	..	..
Coal	0	9	0	9	0	10½	0	11	0	8½
Coal streaked with "sulphur"	[a]0	2½	([b])		..	..	..	..	..	..
Coal	..	..	0	2	..	..	..	..	..	..
Floor, fire clay.										
Thickness of bed	3	2½	3	1	3	2¼	3	1	2	11½
Thickness of coal sampled	2	2½	2	3½	2	¾	2	4	1	11½

[a] Not included in sample. [b] "Sulphur" streak.

Section A (sample W20433) was measured at the face of the second main heading, 400 feet from the mouth of the main entry.

Section B (sample W20434) was measured at the face of entry 1, off the second main heading, 400 feet from the mouth of the main entry.

Section C (sample W20474) was measured at the face of room 5, off the main left heading, 850 feet from the mouth of the main entry.

Section D (sample W20475) was measured at the face of the first left heading, 500 feet from the mouth of the main entry.

Section E (sample W20476) was measured at the face of the straight main heading, 1,200 feet from the mouth of the entry.

A composite sample was made by combining face samples W20433, W20434, W20474, W20475, and W20476. The results of an ultimate analysis of this sample are shown under laboratory No. 12366.

Notes.—At the time of sampling the coal was undercut by hand and shot down with FFFF black powder. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track with a capacity of 28 cars. The mine had a capacity of 100 tons and an average daily output of 80 tons, of which the larger part was from advance workings. There were approximately 50 acres of unmined coal tributary to this opening.

HASTINGS. MILLER RUN MINE.

Sample.—Bituminous coal; Windber field; analyses Nos. 12371, W20440, W20441, W20442, W20443, and W20447 (p. 76).

Mine.—Miller Run, a drift mine 2½ miles northeast of Hastings, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Miller or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet. Roof, shale overlain by sandstone; floor, smooth fire clay.

The bed was measured and sampled at five points in the mine by E. W. Miller on June 9, 1911, as described below:

Sections of coal bed in Miller Run mine.

Section	A		B		C		D		E	
Laboratory No.	W20440		W20441		W20442		W20443		W20447	
Roof, sandstone and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	7	0	5	..	..	0	2	0	1½
Coal, bony	[a]0	9½	..	..	..	..	..	..	..	..
"Sulphur"	..	..	..	..	..	..	0	⅛	..	..
Coal	0	3½	..	..	..	..	0	2½	..	..
Mother coal	0	¼	..	..	..	..	..	..	..	..
"Sulphur" ball	..	..	..	..	..	..	0	¼	0	½
Coal	..	..	..	..	..	..	0	1½	0	5
Coal, bony	..	..	[a]0	10	[a]1	4½	[a]0	9	[a]0	8
Coal	1	2½	1	3½	1	2	1	5½	1	4
"Sulphur"	[a]0	½	..	..	..	..	..	..	..	..
Coal, bony	..	..	0	¾	0	¼	0	¾	0	½
Coal	1	1¾	1	6½	1	6¼	1	4	1	4
Floor, fire clay.										
Thickness of bed	4	1	4	1¾	4	1	4	1⅝	3	11½
Thickness of coal sampled	3	3	3	3¾	2	8½	3	4⅝	3	3½

[a] Not included in sample.

Section A (sample W20440) was measured at the face of the main air course, 2,000 feet from the drift mouth.

Section B (sample W20441) was measured at the face of room 4, off the fifth right entry, 200 feet from the main entry.

Section C (sample W20442) was measured at the face of the first right entry, 200 feet off left entry 2.

Section D (sample W20443) was measured at the face of the second left entry, 500 feet off the main entry.

Section E (sample W20447) was measured on pillar of room 5, first left entry, 300 feet off the main entry.

A composite sample was made by mixing face samples W20440 to W20443. The results of an ultimate analysis of this sample are shown under laboratory No. 12371.

Notes.—At the time of sampling the coal was undercut by hand and shot down with FFF black powder. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track at the tipple with a capacity of 8 empty and 8 loaded cars. The mine was equipped to produce 200 tons a day and had an average daily output of 100 tons, of which the larger part was from advance workings. The intent was to increase the output to 200 tons in the near future. There were 129 acres of unmined coal tributary to this opening.

HASTINGS. PENNSYLVANIA NO. 11 AND NO. 12 MINES.

Sample.—Bituminous coal; Windber field; analyses Nos. 12150 to 12157, inclusive, and 12166 to 12169, inclusive (pp. 76, 77).

Mines.—Pennsylvania No. 11 and Pennsylvania No. 12, drift mines in the central Pennsylvania district, 1 mile south of Hastings, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Lower Freeport or D, Carboniferous age, Allegheny formation. Average thickness, 4 feet 4 inches. Roof, dark shale, overlain by sandstone; floor, hard rough fire clay. Particles of the roof or floor did not become mixed with the coal in mining.

The bed was measured and sampled at five points in the No. 11 mine and at five points in the No. 12 mine by H. I. Smith, April 29, 1911, as described below:

Sections of coal bed in Pennsylvania No. 11 mine.

Section	A		B		C		D		E	
Laboratory No	12150		12151		12166		12167		12169	
Roof, shale and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Shale	[a]0	2½	[a]0	2	[a]0	2¼	[a]0	2	[a]0	2
Coal	1	6	1	1	2	10	2	0	0	11
Mother coal and "sulphur"	0	⅛	0	¼	..	..	..	..	0	¼
Coal	2	8½	2	3	0	4	1	1	2	2¾
"Mining slate" and coal	[a]0	2½	[a]0	1½	[a]0	1	[a]0	1½	[a]0	2⅛
Coal	0	8	1	½	0	7	0	3½	0	10¾
Floor, fire clay.										
Thickness of bed	5	3⅝	4	8¼	4	¼	3	8	4	4⅞
Thickness of coal sampled	4	10⅝	4	4¾	3	9	3	4½	4	¾

[a] Not included in sample.

Section A (sample 12150) was measured at the face of fifth right entry, off the main heading.

Section B (sample 12151) was measured at the face of the old main heading.

Section C (sample 12166) was measured at the face of the second right cross heading.

Section D (sample 12167) was measured at the face of left heading 3.

Section E (sample 12169) was measured on the pillar in room 16, off third left entry.

A composite sample was made by combining face samples 12150, 12151, 12166, and 12167. The results of an ultimate analysis of this sample are shown under laboratory No. 12168.

Sections of coal bed in Pennsylvania No. 12 mine.

Section	A		B		C		D		E	
Laboratory No	12152		12153		12154		12155		12156	
Roof, shale and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Shale	..	..	[a]0	1	[a]0	1½	[a]0	1	[a]0	2
Coal	2	6	0	8	3	2¼	1	6	2	5
Coal, bony	..	..	[a]0	½	..	..	..	..	[a]0	1
Coal	..	..	1	5	..	..	..	..	0	5
Mother coal and "sulphur"	..	..	0	⅛	..	..	0	¼	0	½
Coal	..	..	1	4½	..	..	1	10½	0	5½
"Mining slate" and coal	[a]0	1½	[a]0	¾	[a]0	2	[a]0	1¼	[a]0	3
Coal	0	6	0	5	0	6	0	9½	0	5
Floor, fire clay.										
Thickness of bed	3	1½	4	⅞	3	11¾	4	4½	4	3
Thickness of coal sampled	3	0	3	10⅝	3	8¼	4	2¼	3	9

[a] Not included in sample.

Section A (sample 12152) was measured at the face of the original old main heading, 5,800 feet from the drift mouth.

Section B (sample 12153) was measured at the face of fifth entry, off the main heading.

Section C (sample 12154) was measured at the face of fourth right heading, 1½ miles from the drift mouth.

Section D (sample 12155) was measured at the face of room 1, off first entry off third right heading.

Section E (sample 12156) was measured at the face of third right entry, off the main heading.

A composite sample was made by combining face samples 12152 to 12156, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 12157.

Notes.—The system of mining in both mines is single entry, room-and-pillar. At the time of sampling the coal was undercut in the lower part of the bed by electric chain cutting machines and shot down with FFF and FFFF black powder. There was one loading track at the tipple with a capacity of 28 empty and 28 loaded cars. Pickers were employed on the cars in loading. The entire output of the mines was shipped as run-of-mine coal. The No. 11 mine was equipped to produce about 400 tons of coal a day and had a daily output of 325 tons. There was about 100 acres of coal tributary to this mine. The No. 12 mine was equipped to produce 300 tons a day. There was about 100 acres of coal tributary to this drift entry. All of the coal from these two mines was used for coking purposes. Both mines load over the same tipple.

JOHNSTOWN. VALLEY NO. 1 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12213, W19563, W19564, W19565, W19566, and W19567 (p. 77).

Mine.—Valley No. 1, a drift mine in the central Pennsylvania district, 3 miles south of Johnstown, on the Somerset & Cambria branch of the Baltimore & Ohio Railroad.

Coal bed.—C′ or Upper Kittanning. Carboniferous age, Allegheny formation. Average thickness, 4 feet 2 inches. Roof, bony coal overlain with shale; floor, rough shale.

The bed was measured and sampled at five points in the mine by E. W. Miller on April 27, 1911, as described below:

Sections of coal bed in Valley No. 1 mine.

Section	A		B		C		D		E	
Laboratory No.	W19563		W19564		W19565		W19566		W19567	
Roof, bony coal and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	4¾	1	5	0	2¼	1	3½	0	5¼
Bone, shale, and "sulphur"	[a] 0	¼	0	¼	0	⅝	[a] 0	¾	0	⅜
Coal	2	7	2	8¾	3	6½	2	9½	3	3
Bone	..	..	..	..	..	..	..	..	[a] 0	¼
Coal	..	..	..	..	..	..	..	..	0	7⅛
Floor, shale.										
Thickness of bed	4	0	4	2	3	9⅜	4	1¾	4	4⅛
Thickness of coal sampled	3	11¾	4	2	3	9⅜	4	1	4	3⅞

[a] Not included in sample.

Section A (sample W19563) was measured at the face of room 11 in Andy's heading, 1,000 feet west of tipple No. 2.

Section B (sample W19564) was measured at the face of room 21, off left heading 7, 3,400 feet southeast of the drift mouth. There are small "sulphur" concretions in this section.

Section C (sample W19565) was measured on the chain pillar, 4,200 feet northeast of the drift mouth.

Section D (sample W19566) was measured at the face of room 4, off eighth right heading 3,600 feet southeast of the drift mouth.

Section E (sample W19567) was measured on the rib of the main air course, 150 feet beyond twelfth right heading.

A composite sample was made by mixing face samples W19563, W19564, W19566, and W19567. The results of an ultimate analysis of this sample are shown under laboratory No. 12213.

Notes.—At the time of sampling the coal was undercut by hand and shot down with black powder. There were two tipples; each tipple had one loading track with a capacity of 50 cars. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. The mine had a capacity of about 1,500 tons a day with an average daily output of 1,200 tons, 90 per cent of which was from advance workings.

NANTY GLO. LINCOLN NO. 1 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12180, 12181, 12182, 12183, 12184, and 12185 (p. 78).

Mine.—Lincoln No. 1, a drift mine in the central Pennsylvania district, 1 mile north of Nanty Glo on the Pennsylvania Railroad.

Coal bed.—Lower Kittanning, Miller, or B. Carboniferous age, Allegheny formation. Average thickness, about 3 feet 5 inches. The bed lies practically flat; the cleat is irregular. Roof, hard dark shale overlain by sandstone capping; floor, soft shale. All of the samples were taken at a depth of about 600 feet. There are occasionally small soft "sulphur" lenses in this coal. The bed was measured and sampled at five points in the mine by A. J. Hazlewood on May 3, 1911, as described below:

Sections of coal bed in Lincoln No: 1 mine.

Section	A		B		C		D		E	
Laboratory No	12180		12181		12182		12183		12184	
Roof, hard shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	9	0	10	0	10	0	11	0	10
Dull-coal binder	0	1	0	1½	0	2	0	2	0	2
Coal, soft and bright	2	6	2	5½	2	4	2	5	2	5
Floor, soft shale.										
Thickness of bed	3	4	3	5	3	4	3	6	3	5
Thickness of coal sampled	3	4	3	5	3	4	3	6	3	5

Section A (sample 12180) was measured at the face of the new main heading.

Section B (sample 12181) was measured at the face of right entry 2

Section C (sample 12182) was measured at the face of right entry 4.

Section D (sample 12183) was measured at the face of left entry 7.

Section E (sample 12184) was measured at the face of left entry 9.

Notes.—The coal is mined by the room-and-pillar double-entry system. At the time of sampling the coal was undercut with electric chain-cutting machines and broken down with FFFF black powder. All of the coal was shipped as run-of-mine. The daily output was about 500 tons, practically all of which was from advance workings. The tipple is about 300 feet from the mouth of the mine. There were trackage facilities for handling about 40 cars.

NANTY GLO. SPRINGFIELD NO. 1 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12128, 12129, 12130, 12131, 12132, and 12133 (pp. 78, 79).

Mine.—Springfield No. 1, a slope mine in the central Pennsylvania district, half a mile southeast of Nanty Glo on the Black Lick branch of the Cresson division of the Pennsylvania Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet 7 inches. Roof, firm shale, overlain by a sandstone cap rock 22 feet thick. Floor, a soft shale, generally smooth. Material from the roof or floor did not become mixed with the coal in mining. There are small soft "sulphur" lenses throughout the coal bed.

The bed was measured and sampled at five points in the mine by A. J. Hazlewood on April 29, 1911, as described below:

Sections of coal bed in Springfield No. 1 mine.

Section	A		B		C		D		E	
Laboratory No	12128		12129		12130		12131		12133	
Roof, sandrock, soft shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, soft, with irregular cleavage	0	11	0	11	1	0	1	0	0	10
Coal, dull	0	1	0	1	0	1½	0	1½	0	½
Coal, very soft with a number of soft "sulphur" lenses	2	4	2	8	2	5½	2	6½	2	10½
Floor, soft shale.										
Thickness of bed	3	4	3	8	3	7	3	8	3	9
Thickness of coal sampled	3	4	3	8	3	7	3	8	3	9

Section A (sample 12128) was measured at the face of left entry 4, 2,700 feet from the mine mouth.

Section B (sample 12129) was measured at the face of the main entry, 3,500 feet from the mine mouth.

Section C (sample 12130) was measured at the face of the third crosscut off the dip heading, 2,500 feet from the mine mouth.

Section D (sample 12131) was measured at the face of left entry 1, 3,000 feet from the mine mouth.

Section E (sample 12133) was measured on the pillar in room 3, off left entry 2.

A composite sample was made by combining face samples 12128 to 12131, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 12132.

Notes.—The mine is on a syncline, and is entered by a steep slope 430 feet long. The bed is slightly rolling; there were three or four "swamps" on the main entry. The dip entry turns to the right and at the time of sampling had considerable water in it. The high parts of the mine were dry.

The coal was undercut in the bottom part of the bed with compressed air punchers and broken down with FFFF black powder. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track with a capacity of 20 empty and 20 loaded cars. The mine had a capacity of 550 tons with an average daily output of 500 tons, which was all from advance workings. There was approximately 2,000 acres of unmined coal tributary to this opening.

PORTAGE. FORGE SLOPE NO. 1 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12369, W20396, W20397, W20398, W20399, W20400, and W20401 (p. 79).

Mine.—Forge Slope No. 1, a slope mine in the central Pennsylvania district, 2 miles north of Portage, on the Martins branch of the Pennsylvania Railroad.

Coal bed.—C′ or Upper Kittanning, Carboniferous age, Allegheny formation. Average thickness, 3 feet 5 inches. The roof is bony coal of poor quality 1 foot 6 inches to 3 feet 4 inches thick, and is overlain by shale and sandstone. Floor, fire clay underlain by limestone.

The bed was measured and sampled at six points in the mine by P. M. Riefkin on June 8, 1911, as described below:

Sections of coal bed in Forge Slope No. 1 mine.

Section	A		B		C		D		E		F	
Laboratory No.	W20396		W20397		W20398		W20399		W20400		W20401	
Roof, shale and bony coal.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, soft	..	..	..	..	..	..	0	5½	0	8	..	..
"Sulphur"	..	..	..	..	..	..	0	¼	0	¼	..	..
Coal	1	3¼	1	2½	1	4	0	7	0	6	1	2½
"Sulphur"	..	..	..	..	..	..	0	⅛	..	..	..	..
Coal	..	..	..	..	..	..	0	3	..	..	..	..
Coal, hard	0	3	0	2½	0	2	0	1½	0	2½	0	2¼
Coal	1	10½	2	0	1	11½	2	2	2	1	2	2¼
Floor, fire clay.												
Thickness of bed	3	4¾	3	5	3	5½	3	7⅜	3	5¾	3	7
Thickness of coal sampled	3	4¾	3	5	3	5½	3	7⅜	3	5¾	3	7

Section A (sample W20396) was measured near the face of the main south heading, 1 mile south of the mine mouth.

Section B (sample W20397) was measured on the rib near the face of left entry 7, off the main south entry, 1 mile west of the mine mouth.

Section C (sample W20398) was measured on the rib near the face of right entry 2, off left entry 7, off the main south entry, 4,500 feet northwest from the mine mouth.

Section D (sample W20399) was measured on the rib near the face of right entry 1, off left entry 7, off the main south heading, 5,000 feet southwest of the mine mouth.

Section E (sample W20400) was measured at the face of the main north heading, 3,000 feet north from the mine mouth.

Section F (sample W20401) was measured on the rib near the face of right entry 3, off the main north entry, 2,600 feet north from the mine mouth.

A composite sample was made by combining face samples W20396 to W20401, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 12369.

Notes.—At the time of sampling the coal was undercut with puncher machines and shot down with FFFF black powder. There were two loading tracks with a capacity of 45 empty and 45 loaded cars. There were no screens at the tipple, all of the coal being loaded as run-of-mine. Pickers were employed on the cars in loading. The mine had a capacity of 800 tons and a daily average output of 700 tons, the larger part of which was from advanced workings. The intent was to increase the output to 800 tons a day. There was approximately 300 acres of unmined coal tributary to this opening.

PORTAGE. MILLER No. 1 SHAFT.

Sample.—Semibituminous coal; Windber field; analyses Nos. 11355, 11356, 11357, 11358, 11359, and 11360 (p. 79).

Mine.—Miller No. 1 shaft; central Pennsylvania district; a shaft mine 1 mile southeast of Portage, on the main line of the Western Pennsylvania Division of the Pennsylvania Railroad. There are railroad connections from the mine to the main line.

Coal bed.—Lower Kittanning, Miller, or B. Carboniferous age, Allegheny formation. Average thickness, about 3 feet 6 inches. Dip, 5 to 15 per cent, west. Roof, hard black shale about 18 inches thick, above which is a sandstone cap rock; floor, medium-hard smooth clay. Cover at points of sampling, 200 to 425 feet. The bed was measured and sampled at five points in the mine by Charles Enzian on December 7, 1910, as described below:

Sections of coal bed in Miller No. 1 shaft.

Section	A		B		C		D		E	
Laboratory No.	11355		11356		11357		11358		11359	
Roof, hard shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Bone	[a]0	4½	[a]0	3½	[a]0	2¾	[a]0	5¾	[a]0	5½
Coal	2	9½	1	3¾	1	2	1	1	1	0
"Sulphur" band	..	..	0	⅛	..	..	..	..	..	..
Shale, hard	..	..	..	..	0	¼	0	⅛	0	¼
Coal	..	..	1	4½	1	8½	2	3¾	2	¼
Mother coal	..	..	0	¼	0	¼	..	..	..	..
Coal	..	..	0	11½	0	3½	..	..	..	..
Floor, soft shale.										
Thickness of bed	3	2	3	11¼	3	5¼	3	10⅝	3	6
Thickness of coal sampled	2	9½	3	7¾	3	2½	3	4⅞	3	½

[a] Not included in sample.

Section A (sample 11355) was measured at the face of the second right dip, 2,800 feet southwest of the shaft.

Section B (sample 11356) was measured at the face of third left heading, 4,100 feet southeast of the shaft.

Section C (sample 11357) was measured on the pillar of the A-1 heading, 2,000 feet from the foot of the shaft.

Section D (sample 11358) was measured at the face of the first right dip, 1,500 feet from the foot of the shaft.

Section E (sample 11359) was measured at the face of first left heading, 3,000 feet from the foot of the shaft.

A composite sample was made by combining face samples 11355 to 11359, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 11360.

Notes.—The mine is opened by a shaft about 425 feet deep. The plan of working is triple entry, room-and-pillar. At the time of sampling the coal was undercut by hand. Black powder was used for breaking down the coal and dynamite for brushing the roof. The tipple was steel. There were two loading tracks with a capacity of 80 cars. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. The mine was comparatively new and had an average daily output of 700 tons, all of which was from advance workings. The intent was to increase the output to about 1,500 tons a day. When the pillars were pulled, the total recovery was about 90 per cent.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 160, 712, 713.

PORTAGE. PLYMOUTH No. 1 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 11705, W17110, W17111, and W17112 (p. 80).

Mine.—Plymouth No. 1, a drift mine in the central Pensylvania district, 2½ miles east of Portage, on the Pennsylvania Railroad.

Coal bed.—C′ or Upper Kittanning. Carboniferous age, Allegheny formation. Average thickness, 3 feet 8 inches. Roof, bony coal, overlain by shale. Floor, hard, smooth fire clay which softens readily when in contact with the air.

The bed was measured and sampled at three points in the mine by G. S. Pope on January 27, 1911, as described below:

Sections of coal bed in Plymouth No. 1 mine.

Section	A		B		C	
Laboratory No	W17110		W17111		W17112	
Roof, bone and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	3	1	3	1	1
Bone	[a] 0	½	..	..	[a] 0	1½
Bone, soft	..	..	0	1	..	..
Coal	..	..	0	8	..	..
"Sulphur"	..	..	[a] 0	½	..	..
Coal	2	4	1	8	1	11¼
Floor, fire clay.						
Thickness of bed	3	7½	3	8½	3	1¾
Thickness of coal sampled	3	7	3	8	3	¼

[a] Not included in sample.

Section A (sample W17110) was measured at the face of No. 5 dip, 300 feet from the main heading.

Section B (sample W17111) was measured at the face of fourth left entry, 2,000 feet from the main heading.

Section C (sample W17112) was measured at the face of the main heading, 250 feet from eighth left entry.

Notes.—At the time of sampling the coal was undercut by hand and shot down with FFFF black powder. The tipple had one loading track with a capacity of 15 cars. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. The mine had a capacity of 300 tons a day and an average daily output of 250 tons. Plans were under way for installing rope haulage and doubling the production of the mine. There were approximately 750 acres of unmined coal tributary to this opening.

PORTAGE. SONMAN No. 2 SHAFT.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12205, 12217, W19688, W19689, W19690, W19691, W19692, W19693, and W19694 (p. 80).

Mine.—Sonman No. 2 shaft, in the central Pennsylvania district, 1 mile east of Portage, on the Sonman branch of the Pennsylvania Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Thickness, 3 feet 8 inches to 4 feet 6 inches, fairly uniform. Dip, 11½° N., 68° W. Roof, smooth shale; above the shale is a sandstone cap rock; floor, rough fire clay. Particles of the roof or floor did not become mixed with the coal in mining.

The bed was measured and sampled at seven points in the mine by P. M. Riefkin on May 1, 1911, as described below.

Sections of coal bed in Sonman No. 2 shaft.

Section	A		B		C		D		E		F		G	
Laboratory No	W19688		W19689		W19690		W19691		W19692		W19693		W19694	
Roof, sandstone and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	[a]0	2½	[a]0	4	[a]0	6½	[a]0	8	[a]0	7	[a]0	7½	[a]0	5½
Coal	..	..	..	..	..	..	1	2½	..	..	1	5½	1	8½
Coal, bony	..	..	..	..	..	..	0	¼	..	..	0	¼	0	1
Coal	3	½	2	5	3	10	2	5	2	8½	1	8	0	7
"Sulphur" and mother coal	..	..	..	..	..	..	..	..	0	¼	0	¼	Streak.	
Coal, hard	..	..	1	2	..	..	..	..	..	..	..	..	..	..
Coal	..	..	..	..	..	..	..	..	0	8	0	7½	1	6
Floor, fire clay.														
Thickness of bed	3	3	3	11	4	4½	4	3¾	3	11¾	4	5	4	4
Thickness of coal sampled	3	½	3	7	3	10	3	7¾	3	4¾	3	9½	3	10½

[a] Not included in sample.

Section A (sample W19688) was measured at the face of room 8, left heading 2 off dip off the main south heading.

Section B (sample W19689) was measured on a pillar near the face of room 23, first right off plane, 3,000 feet southeast of the shaft.

Section C (sample W19690) was measured at the face of room 13, left heading 1 off dip off the main south heading.

Section D (sample W19691) was measured at the face of "No. 32 crook" off the main north heading.

Section E (sample W19692) was measured at the face of "No. 28 crook," left heading 1 off dip off main south heading.

Section F (sample W19693) was measured at the face of room 10, first right off plane 17 off main north heading.

Section G (sample W19694) was measured at the face of right heading 3 off main dip off main north heading.

A composite sample was made by combining face samples W19689, W19691, and W19692. The results of an ultimate analysis of this sample are shown under laboratory No. 12217.

A composite sample was made by combining face samples W19688, W19690, W19693, and W19694. The results of an ultimate analysis of this sample are given under laboratory No. 12205.

Notes.—The mine is opened by a shaft 320 feet deep. At the time of sampling the coal was undercut by hand and broken down with black powder. The entire output was shipped as run-of-mine. Pickers were employed on the car in loading. There were two loading tracks with a capacity of 55 empty and 55 loaded cars. The daily output was 1,100 tons, about 75 per cent of which was from advance workings. There was approximately 1,500 acres of unmined coal tributary to this opening.

SPANGLER. PENNSYLVANIA NO. 21 AND NO. 22 MINES.

Sample.—Bituminous coal; Windber field; analyses Nos. 12126, 12127, 12140 to 12148, inclusive (p. 81).

Mine.—Pennsylvania No. 21 and Pennsylvania No. 22, slope mines, 2 miles northwest of Spangler, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Lower Freeport or D, Carboniferous age, Allegheny formation. Average thickness, 3 feet 10 inches. Roof, bony coal overlain by "draw slate" 6 inches thick, which comes down with the coal. Floor, hard fire clay with rough surface. In mining care was taken that the material from the roof and floor did not become mixed with the coal.

The bed was measured and sampled at five points in No. 22 mine and at four points in No. 21 mine by H. I. Smith, on April 26 and 27, 1911, as described below:

Sections of coal bed in Pennsylvania No. 21 mine.

Section	A		B		C		D	
Laboratory No.	12144		12145		12146		12147	
Roof, bony coal, and "draw slate."	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	4½	..	..	0	4	..	..
Coal and "shale"	0	¼	a 0	1	0	¼	..	..
Coal	1	0	0	8½	0	5	1	3
Coal and "sulphur"	..	..	a 0	1¼	a 0	2	a 0	½
Coal	..	..	0	4½	0	3½	0	6
Coal	..	..	a 0	1	a 0	2¼	..	..
Bony coal	0	½	a 0	1	a 0	1¼	0	1
Coal	..	..	a 0	½	a 0	¾	..	..
Coal	1	4	1	1	1	0	0	10
Coal	a 0	½	a 0	¾	a 0	1	a 0	½
"Mining slate"	a 0	½	a 0	1¼	a 0	1¼	a 0	1
Coal	a 0	1	a 0	1	a 0	¾	a 0	½
Coal	0	10	0	8	0	8	0	9
Floor, hard fire clay.								
Thickness of bed	3	9	3	5¾	3	6	3	7½
Thickness of coal sampled	3	7	2	10	2	8¾	3	5

a Not included in sample.

Section A (sample 12144) was measured at the face of the main heading, 5,000 feet southwest of the mine mouth.

Section B (sample 12145) was measured at the face of the second main entry, 6,500 feet from the mine mouth.

Section C (sample 12146) was measured at the face of back heading 17. The point of sampling was 5,900 feet west of the mine mouth.

Section D (sample 12147) was measured at the face of right entry 4 off the second main entry, 6,300 feet east of the mine mouth.

A composite sample was made by combining face samples 12144 to 12147, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 12148.

Sections of coal bed in Pennsylvania No. 22 mine.

Section	A		B		C		D		E	
Laboratory No.	12126		12127		12140		12141		12142	
Roof, cannel coal and "draw slate."	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	8	1	5¼	0	11	0	10	0	4½
Mother coal and "sulphur"	a 0	½	..	..	0	½	a 0	¼	0	½
Coal	1	3½	..	..	0	6½	..	..	0	7½
Mother coal and "sulphur"	..	..	..	..	..	..	..	..	a 0	¾
Coal	..	..	..	..	..	..	..	..	0	3
Coal gray, bony	0	½	a 0	1½	0	1½	..	..	a 0	1½
Coal	0	9	1	1	1	2¼	1	7½	1	0
Coal	a 0	1	a 0	½	a 0	½	a 0	1	..	..
"Mining slate"	a 0	1	a 0	¾	a 0	1	a 0	1½	a 0	¼
Coal	a 0	1½	a 0	1	a 0	½	a 0	1	..	..
Coal	1	0	1	½	1	0	0	8½	0	10
Floor, hard fire clay.										
Thickness of bed	4	1	3	11	3	11¾	3	5¾	3	3
Thickness of coal sampled	3	9	3	7½	3	9¾	3	2	3	1½

a Not included in sample.

Section A (sample 12126) was measured at the face of the main ninth entry, 7,200 feet southwest of the mine mouth.

Section B (sample 12127) was measured on the rib of room 9, right entry 4, off the main ninth entry, 6,200 feet southwest of the mine mouth.

Section C (sample 12140) was measured at the face of room 17, off right entry 6, off the main ninth entry, 7,400 feet from the mine mouth.

Section D (sample 12141) was measured at the face of the main back heading, 7,100 feet from the mine mouth.

Section E (sample 12142) was measured at the face of the fourteenth right off the main heading, 7,200 feet from the mine mouth.

A composite sample was made by combining face samples 12126, 12127, 12140, 12141, and 12142. The results of an ultimate analysis of this sample are given under laboratary No. 12143.

Notes.—At the time of sampling the coal was undercut both by hand and with electric chain machines and shot down with FFFF black powder and "40 per cent" nitroglycerin dynamite. There was one loading track at the tipple with a capacity of 50 empty and 100 loaded cars. All of the coal was shipped as run-of-mine. The tipple was equipped with bar screens with 3-inch openings; if screened about 50 per cent of the coal would pass through the 3-inch spaces. Pickers were employed on the cars in loading. The capacity of the storage bin was 165 tons. The mines had a capacity of 1,500 tons a day, with a daily average output of 1,000 tons, 60 per cent of which was from advance workings. There was approximately 5,000 acres of unmined coal tributary to this opening.

TWIN ROCKS. BIG BEND NO. 1 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. W19668, W19669, W19672, W19674, W19676, 12208, and 12215 (p. 82).

Mine.—Big Bend No. 1, a drift mine in Black Lick Township at Twin Rocks, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Miller, B, or Lower Kittanning. Carboniferous age, Allegheny formation. Average thickness, 3 feet 8 inches. Roof, smooth, hard, gray shale 15 feet thick, overlain with sandstone. Floor, hard, smooth shale, underlain with fire clay.

The bed was measured and sampled at five points in the mine by E. W. Miller on April 29, 1911, as described below:

Sections of coal bed in Big Bend No. 1 mine.

Section	A		B		C		D		E	
Laboratory No.	W19668		W19669		W19672		W19674		W19676	
Roof, sandstone and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	3	7½	1	11	1	9	0	3½	4	3
"Sulphur"	..	..	0	¼	..	..	..	..	..	..
"Coal"	..	..	0	5	..	..	..	..	..	..
"Sulphur" and shale	..	..	0	¼	[a]0	¾	[a]0	½	..	..
Coal	..	..	1	2¼	1	9½	3	3½	..	..
Floor, shale and fire clay.										
Thickness of bed	3	7½	3	6¾	3	7¼	3	7½	4	3
Thickness of coal sampled	3	7½	3	6¼	3	6½	3	7	4	3

[a] Not included in sample.

Section A (sample W19668) was measured at the face of room 11 off the drainage heading, 2,600 feet northeast of the main opening.

Section B (sample W19669) was measured at the face of the water-level heading 2,000 feet northeast of the main opening.

Section C (sample W19672) was measured on the pillar opposite room 16, Claghorn section, 3,000 feet east of the main opening.

Section D (sample W19674) was measured on the pillar to left of main heading, 2,500 feet east of the main opening.

Section E (sample W19676) was measured on the pillar to the left of the main heading, opposite right headings 6 and 7, 2,000 feet east of the main opening.

A composite sample was made by combining face samples W19668 and W19669. The results of an ultimate analysis of this sample are shown under laboratory No. 12215.

A composite sample was made by combining face samples W19672, W19674, and W19676. The results of an ultimate analysis of this sample are shown under laboratory No. 12208.

Notes.—At the time of sampling, the coal was undercut by hand and broken down with black powder. There was no screen, the entire output being shipped as run of-mine. Pickers were employed on the cars in loading. There was one loading track with a capacity of 10 cars. The mine had a daily capacity of 450 tons, with an average output of 375 tons a day, 70 per cent of which was from advance workings.

TWIN ROCKS. BIG BEND NO. 6 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. W19670, W19671, W19673, W19675, W19677, 12707, and 12227 (p. 83).

Mine.—Big Bend No. 6, a drift mine in Black Lick Township, 1 mile north of Twin Rocks, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Miller, B, or Lower Kittanning. Carboniferous age, Allegheny formation. Average thickness, 5 feet 9 inches. The roof, a hard, smooth gray shale, 15 feet thick, is overlain by a sandstone cap rock 25 feet thick. The floor is a hard, smooth shale; below the shale is a fire clay.

The bed was measured and sampled at five points in the mine by E. W. Miller on April 29, 1911, as described below:

Sections of coal bed in Big Bend No. 6 mine.

Section	A		B		C		D		E	
Laboratory No	W19670		W19671		W19673		W19675		W19677	
Roof, shale and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	1½	0	3¼	2	1¼	..	..	0	¾
"Sulphur" streak	[a]0	¼	[a]0	⅛	[a]0	1½	..	..	..	..
Shale	..	..	..	..	..	..	..	..	[a]0	½
Coal	3	7¾	3	4½	1	7	3	9½	3	7½
Floor, shale and fire clay.										
Thickness of bed	3	9½	3	7⅞	3	9¾	3	9½	3	8¾
Thickness of coal sampled	3	9¼	3	7¾	3	8¼	3	9½	3	8¼

[a] Not included in sample.

Section A (sample W19670) was measured at the face of room 1, left side of right heading 1 off No. 9 drift, 900 feet south of No. 9 opening. There were sulphur streaks in the coal represented by this section.

Section B (sample W19671) was measured on the pillar between rooms 23 and 24 off the main heading, 1,500 feet northeast of the main opening. There were irregular sulphur streaks in the coal represented by this section.

Section C (sample W19673) was measured at the face of heading 13, 325 feet from the outcrop and 3,500 feet southeast of the main opening. There were sulphur streaks throughout the coal represented by this section.

Section D (sample W19675) was measured on the pillar between rooms 27 and 28 off the main heading, 2,000 feet northeast of the main opening. There were small irregular sulphur streaks in the coal represented by this section.

Section E (sample W19677) was measured at the face of room 10 on the left side of No. 9 drift, 700 feet southwest of No. 9 opening. Sulphur balls and streaks were frequent in the coal represented by this section.

A composite sample was made by combining face samples W19671 and W19675. The results of an ultimate analysis of this sample are shown under laboratory No. 12207.

Notes.—At the time of sampling the coal was undercut by hand and shot down with black powder. There was one loading track at the tipple with a capacity of 16 cars. There were no screens, the entire output being loaded as run-of-mine. Pickers were employed on the cars in loading. The mine had a daily capacity of 450 tons and an average daily output of 375 tons, of which about 70 per cent was from advance workings.

TWIN ROCKS. COMMERCIAL No. 3 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12377, W20477, W20478, W20479, W20480, and W20481 (p. 83).

Mine.—Commercial No. 3, a slope mine at Twin Rocks, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Miller or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet 9 inches. The roof, a black shale 5 to 8 feet thick, is overlain by sandstone. The floor is shale and bottom coal, underlain by fire clay.

The bed was measured and sampled at five points in the mine by E. W. Miller on June 10, 1911, as described below:

Sections of coal bed in Commercial No. 3 mine.

Section	A		B		C		D		E	
Laboratory No	W20477		W20478		W20479		W20480		W20481	
Roof, sandstone and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	5¾	0	1	0	5½	0	1	0	7¾
"Sulphur" streak	..	..	0	⅛	0	⅛	0	¼	0	⅛
"Sulphur" ball	ᵃ1	2	..	..	..	..	..	..	..	..
Coal	3	4½	1	9½	2	0	2	1½	3	¼
"Sulphur"	..	..	ᵃ0	⅝	ᵃ0	⅛	0	⅛	..	..
Coal	..	..	1	6¾	0	5½	0	4½	..	..
"Sulphur"	..	..	..	..	ᵃ0	⅛	0	⅜	..	..
Coal	..	..	..	..	0	5½	..	..	..	..
"Sulphur"	..	..	..	..	0	⅛	..	..	..	..
Coal	..	..	..	..	0	1¼	1	4	..	..
Floor, shale, bottom coal, and fireclay.										
Thickness of bed	5	¼	3	6	3	6¼	3	11¾	3	8⅛
Thickness of coal sampled	3	10¼	3	5⅜	3	6	3	11¾	3	8⅛

ᵃ Not included in sample.

Section A (sample W20477) was measured at the face of the main heading, 6,200 feet northwest of the mine mouth.

Section B (sample W20478) was measured on the pillar of fourth right heading, 5,000 feet from the main entry and 4,500 feet northeast of the mine mouth.

Section C (sample W20479) was measured at the face of seventh right heading, 450 feet from the main entry and 5,600 feet northwest of the mine mouth.

Section D (sample W20480) was measured at the face of seventh left entry, 700 feet from the main entry and 5,300 feet northwest of the mine mouth.

Section E (sample W20481) was measured at the face of fifth right heading, 5,000 feet from the main entry and 4,200 feet north of the mine mouth.

A composite sample was made by combining face samples W20477, W20479, W20480, and W20481). The results of an ultimate analysis of this sample are given under laboratory No. 12377.

Notes.—At the time of sampling the coal was undercut by hand and with machines and shot down with FFF black powder. There was one loading track at the tipple which had a capacity of 28 empty and 28 loaded cars. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. The mine was equipped to produce about 1,000 tons a day and was operating at full capacity. There was approximately 850 acres of unmined coal tributary to this opening.

TWIN ROCKS. COMMERCIAL No. 4 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12364, W17061, W20501, W20503, W20505, W20506, W20508 (p. 83).

Mine.—Commercial No. 4, a slope mine near Twin Rocks, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet 9 inches. Roof, a black shale 5 to 8 feet thick, above which is a sandstone cap rock. Floor, shale, underlain by fire clay.

The bed was measured and sampled at five points in the mine by P. M. Riefkin on June 12, 1911 and at one point by G. S. Pope on January 26, 1911, as described below:

Sections of coal bed in Commercial No. 4 mine.

Section	A		B		C		D		E		F	
Laboratory No.	W20501		W20503		W20505		W20506		W20508		W17061	
Roof, sandstone and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	4	2	1	2	0	2	0	1	11½	0	6
"Sulphur"	0	¾	0	¼	0	½	0	⅛	[a]0	½	0	¼
Coal	..	..	..	..	0	3¼	..	..	..	..	..	..
"Sulphur"	..	..	..	..	0	½	..	..	..	..	..	..
Coal	..	..	..	..	0	2¼	..	..	..	..	..	..
Mother coal	..	..	..	..	0	¼	..	..	..	..	..	..
Coal	1	3¼	1	3½	1	2	1	10½	1	7½	3	1¼
Floor, shale and fire clay.												
Thickness of bed	3	8	3	4¾	3	8¾	3	10⅝	3	7½	3	7½
Thickness of coal sampled	3	8	3	4¾	3	8¾	3	10⅝	3	7	3	7½

[a] Not included in sample.

Section A (sample W20501) was measured 100 feet from the face of the main heading, 5,200 feet north of the mine mouth.

Section B (sample W20503) was measured on the sixth pillar of east entry 9 off right entry 2 off the main entry, 3,800 feet northeast of the mine mouth.

Section C (sample W20505) was measured at the face of east entry 6 off right entry 3 off the main entry, 4,000 feet northeast of the mine mouth.

Section D (sample W20506) was measured on the rib near the face of east entry 7 off right entry 3 off the main entry, 4,800 feet northeast of the mine mouth.

Section E (sample W20508) was measured at the face of right entry 3 off the main entry, 4,900 feet northeast of the mine mouth.

Section F (sample W17061) was measured at the face of the third main entry off No. 4 slope.

A composite sample was made by combining face samples W20501, W20505, W20506, and W20508. The results of an ultimate analysis of this sample are shown under laboratory No. 12364.

Notes.—At the time of sampling the coal was undercut both by hand and with machines and broken down with FFF black powder. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track with a capacity of 26 empty and 26 loaded cars. The capacity of the mine was 600 tons a day with an average daily output of 450 tons. There were approximately 850 acres of unmined coal.

TWIN ROCKS. COMMERCIAL NO. 5 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12375, W17062, W20499, W20500, W20502, W20504, and W20507 (p. 84).

Mine.—Commercial No. 5, a slope mine 1 mile from Twin Rocks, on the Cresson division of the Pennsylvania Railroad.

Coal bed.—Miller or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet 10 inches. Roof, shale 3 to 15 feet thick, overlain by a sandstone cap rock; floor, shale underlain with fire clay.

The bed was measured and sampled at five points in the mine by E. W. Miller on June 12, 1911, and at one point in the mine by P. M. Riefkin on January 26, 1911, as described below:

Sections of coal bed in Commercial No. 5 mine.

Section	A		B		C		D		E		F	
Laboratory No.	W17062		W20499		W20500		W20502		W20504		W20507	
Roof, sandstone and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	10	0	11½	0	4½	1	11	0	1½	1	2
"Sulphur"	[a] 0	½	0	¼	[a] 0	¼	[a] 0	½	[a] 0	½	..	..
Coal	..	..	1	¼	0	5½	..	..	0	2½	..	..
"Sulphur"	..	..	0	¼	0	¼	..	..	[a] 0	½	0	¼
Coal	..	..	0	5¼	1	5	..	..	1	1¼	1	2½
"Sulphur"	..	..	0	¼	[a] 0	¾	..	..	0	¼	..	..
Coal	..	..	0	1½	1	2	..	..	0	7¼	..	..
"Sulphur"	..	..	0	¼	[a] 0	1	..	..	[a] 0	½	0	¼
Coal	2	9½	1	0	..	..	1	9	1	5½	1	3¼
Floor, shale and fire clay.												
Thickness of bed	3	8	3	7½	3	7¼	3	8½	3	7¾	3	8¼
Thickness of coal sampled	3	7½	3	7½	3	5¼	3	8	3	6¼	3	8¼

[a] Not included in sample.

Section A (sample W17062) was measured at the face of room 1 off east heading 10 off the main entry.

Section B (sample W20499) was measured at the face of the main heading; 4,700 feet northeast of the mine mouth.

Section C (sample W20500) was measured at the face of room 7 off right heading 1; 500 feet from the second parallel entry and 3,400 feet northeast of the mine mouth.

Section D (sample W20502) was measured at the face of east heading 12; 325 feet from the main entry and 4,700 feet northeast of the mine mouth.

Section E (sample W20504) was measured at the face of east heading 11; 800 feet from the main entry and 4,000 feet northeast of the mine mouth.

Section F (sample W20507) was measured at the face of room 10 off east heading 10; 750 feet from the main heading and 3,900 feet northeast of the mine mouth.

A composite sample was made by combining face samples W20499, W20500, W20502, W20504, and W20507. The results of an ultimate analyses of this sample are shown under laboratory No. 12375.

Notes.—At the time of sampling the coal was undercut by hand and shot down with FFF black powder. There were no screens, the entire output being shipped as run-of-mine coal. Pickers were employed on the cars in loading. There was one loading track at the tipple with a capacity of 28 empty and 28 loaded cars. The mine was equipped to produce 500 tons a day and had an average daily output of 300 tons. An increase to 800 tons was contemplated.

CENTER COUNTY.

OSCEOLA MILLS. ELECTRIC MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12067, 12068, 12069, 12070, and 12190 (p. 84).

Mine.—Electric, a drift mine, one-half mile south of Osceola Mills on the Pennsylvania Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 5 feet. Roof, smooth shale; above the shale is a cap rock about 15 feet thick. Floor, smooth hard fire clay.

The bed was measured and sampled at four points in the mine by P. M. Riefkin on April 18, 1911, as described on the following page.

Sections of coal bed in Electric mine.

Section	A		B		C		D	
Laboratory No	12067		12068		12069		12070	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Top coal	[a]0	1½	[a]0	1	[a]0	1	[a]0	1
Coal	0	9¼	0	9	0	8¼	0	6
Sulphur, soft	..	..	0	¼	0	¼	0	½
Coal	..	..	0	4½	0	4	0	4¾
Coal, hard	[a]0	3	[a]0	2	[a]0	2½	[a]0	4
Coal, bony	[a]0	4	[a]0	3	[a]0	4	[a]0	3¼
Coal	1	11¼	1	9	1	8	1	7½
"Sulphur" streaks, soft	..	..	0	¼	..	..	..	..
Coal	..	..	0	2¾	..	..	..	..
"Sulphur"	0	½	[a]0	¾	[a]0	1	[a]0	¼
Coal	..	..	0	5	..	..	..	..
"Sulphur"	..	..	[a]0	½	..	..	..	..
Coal	1	0	0	7	1	2¾	1	3
Bottom coal	[a]0	4½	[a]0	5½	[a]0	3	[a]0	4
Floor, hard fire clay.								
Thickness of bed	4	10	5	2½	4	10¾	4	10¼
Thickness of coal sampled	3	9	4	1¾	3	11¼	3	9¾

[a] Not included in sample.

Section A (sample 12067) was measured at the face of right entry 1, off the new drift entry.

Section B (sample 12068) was measured on the pillar at room 9, off left entry 5, off the main entry.

Section C (sample 12069) was measured on the pillar at room 7, off right entry 4, off the main heading.

Section D (sample 12070) was measured on the pillar at room 5, off left entry 5, off the main heading.

A composite sample was made by combining face samples 12068, 12069, and 12070. The results of an ultimate analysis of this sample are shown under laboratory No. 12190.

Notes.—At the time of sampling the coal was undercut and then broken down with FFFF black powder. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track with a capacity of 20 empty and 15 loaded cars. The mine had a capacity of 400 tons a day; the average output was about 300 tons, 80 per cent of which was from pillars and 20 per cent from advance workings. There were approximately 200 acres of pillars to be taken out through this opening.

OSCEOLA MILLS. MOSHANNON NO. 10 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12057, 12058, 12059, 12060, 12061, and 12062 (p. 85).

Mine.—Moshannon No. 10, a drift mine in the Clearfield district one-fourth of a mile north of Osceola Mills on the Pennsylvania Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Thickness, 3½ to 6 feet, average, 5 feet. Roof, smooth shale; floor, hard fire clay.

The bed was measured and sampled at five points in the mine by P. M. Riefkin on April 18, 1911, as described below:

Sections of coal bed in Moshannon No. 10 mine.

Section	A		B		C		D		E	
Laboratory No	12057		12058		12059		12060		12061	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Top coal	0	1	[a]0	½	..	..	..	..	[a]0	2
Coal	0	6	0	6½	0	8¾	0	7	0	3¾
Coal, hard (gray)	[a]0	4	[a]0	4	[a]0	2½	[a]0	2½	[a]0	3¾
Coal, bony	[a]0	6	[a]0	3½	[a]0	4½	[a]0	3	[a]0	4½
Coal	1	5	2	½	1	9	1	8	1	6½
"Sulphur"	[a]0	¼	[a]0	½	0	¼	[a]0	¼	..	..
Coal	0	11	0	10½	0	5	1	0	..	..
"Sulphur"	[a]0	1	..	..	[a]0	½	..	..	[a]0	½
Coal	0	6½	..	..	0	4¾	..	..	0	4¼
Bottom coal	0	2	0	3	0	4½	..	..	0	4¼
Floor, fire clay.										
Thickness of bed	4	6¾	4	5	4	3¾	3	8¾	3	5½
Thickness of coal sampled	3	7½	3	8½	3	8¼	3	3	2	6¾

[a] Not included in sample.

Section A (sample 12057) was measured on the rib of room 18, 40 feet from left entry 2, off right entry 4.

Section B (sample 12058) was measured at the face of room 21, off the left main heading.

Section C (sample 12059) was measured at the face of room 16, off the right main heading, near the first crosscut to the right.

Section D (sample 12060) was measured at the face of right heading 1.

Section E (sample 12061) was measured at the face of room 16, off right heading 2.

A composite sample was made by combining face samples 12057 to 12061. The results of an ultimate analysis of this sample are given under laboratory No. 12062.

Notes.—At the time of sampling the coal was undercut by electric cutting machines and shot down with FFFF black powder. About 90 per cent of the output was shipped as run-of-mine coal, the machine cuttings, which amounted to about 10 per cent of the output, being loaded separately. Pickers were employed on the cars in loading. There was one loading track with a capacity of 25 empty and 15 loaded railroad cars. The mine was equipped to produce 700 tons a day and had an average daily output of 500 tons, about 75 per cent of which was from advance workings. An increase of about 50 per cent in the output was contemplated. There was approximately 600 acres of unmined coal tributary to this opening.

OSCEOLA MILLS. WESTON MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12063, 12064, and 12065 (p. 85).

Mine.—Weston, a drift mine in the Clearfield district, one-half mile south of Osceola Mills, on the Pennsylvania Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 5 feet. Roof, smooth black shale; floor, hard smooth fire clay. Neither roof nor floor material got mixed with the coal in mining.

The bed was measured and sampled at three points in the mine by P. M. Riefkin on April 18, 1911, as described below:

Sections of coal bed in Weston mine.

Section	A		B		C	
Laboratory No.	12063		12064		12065	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Top coal			[a]0	1	[a]0	1
Coal	0	9½	1	2	1	4
"Sulphur"	[a]0	¼	..	..	..	..
Coal	0	4½	..	..	..	..
Coal, bony (gray)	[a]0	2	[a]0	1	[a]0	1
Coal, bony	[a]0	4	[a]0	4	[a]0	4
Coal, gray	[a]0	1½			..	..
Coal	2	5½	2	11	3	3
"Sulphur"	..	..	[a]0	½	..	..
Coal	..	..	0	3	..	..
Bottom coal	[a]0	2½	[a]0	3	[a]0	2
Floor, hard fire clay.						
Thickness of bed	4	5¾	5	1½	5	3
Thickness of coal sampled	3	7½	4	4	4	7

[a] Not included in sample.

Section A (sample 12063) was measured at the face of left entry 4, off the main entry.

Section B (sample 12064) was measured on the rib of the main heading at left entry 7.

Section C (sample 12065) was measured on a pillar at room 8, off right entry 1, off the main heading.

A composite sample was made by combining face samples 12063 and 12064. The results of an ultimate analysis of this sample are shown under laboratory No. 12066.

Notes.—At the time of sampling the coal was undercut by hand and broken down with FFFF black powder. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track with a capacity of 20 empty and 15 loaded cars. The mine had a capacity of 400 tons and a daily average output of 200 tons. Approximately 400 acres of unmined coal was to be developed from a new opening on this property.

CLEARFIELD COUNTY.

BOARDMAN. POTTS RUN NO. 3 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12093, 12094, 12095, 12096, 12097, and 12098 (pp. 85, 86).

Mine.—Potts Run No. 3, a drift mine in the Clearfield district, 1 mile west of Boardman, on the New York Central Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet 10 inches. Roof, shale; floor, fire clay.

The bed was measured and sampled at five points in the mine by P. M. Riefkin on April 24, 1911, as described below:

Sections of coal bed in Potts Run No. 3 mine.

Section	A		B		C		D		E	
Laboratory No	12093		12094		12095		12096		12097	
Roof, draw shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in*	*Ft.*	*in.*
Coal	..	..	..	..	..	..	[a] 0	1½	0	3½
Sulphur	..	..	..	..	..	..	..	..	Streak.	
Coal	2	3	2	2¼	2	2	2	3	1	11
Bony coal to shale	[a] 0	1	[a] 0	½	[a] 0	½	[a] 0	1	[a] 0	1½
Coal, gray	0	2½	..	..	..	..	..	..	..	..
Coal	1	1	1	5	1	4	1	6	1	3¼
Floor, fire clay.										
Thickness of bed	3	7½	3	7¾	3	6½	3	11½	3	7¼
Thickness of coal sampled	3	6½	3	7¼	3	6	3	9	3	5¾

[a] Not included in sample.

Section A (sample 12093) was measured at the face of right entry 3, off the main entry, 1,400 feet southwest of the mine mouth.

Section B (sample 12094) was measured 10 feet from the face on the rib of right entry 4, off the main entry, 1,700 feet southwest of the mine mouth.

Section C (sample 12095) was measured at the face of left entry 3, off the main entry, 1,680 feet northeast of the mine mouth.

Section D (sample 12096) was measured at the face of the main heading, 2,000 feet north of the mine mouth.

Section E (sample 12097) was measured at the face of room 12, off right entry 2, off the main entry.

A composite sample was made by combining face samples 12093 to 12097, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 12098.

Notes.—At the time of sampling the coal was undercut by hand and broken down with FFFF black powder. Dynamite was used for brushing the roof and floor. There was one loading track with a capacity of 18 empty and 15 loaded cars. Pickers were employed on the car in loading. The mine had a capacity of 550 tons a day.

BRISBIN. LENORE NO. 1 AND NO. 2 MINES.

Sample.—Bituminous coal; Windber field; analyses Nos. 12044, 12045, 12046, and 12047 (p. 86).

Mines.—Lenore No. 1 and Lenore No. 2, drift mines in the Clearfield district, 1 mile southwest of Brisbin, on the Pennsylvania Railroad.

Coal bed.—Upper Freeport or E. Carboniferous age, Allegheny formation. Average thickness of coal mined, 3 feet. Roof, smooth sandstone; above the sandstone is a second cap rock. Floor, smooth, hard, dark fire clay.

The coal bed was measured and sampled at one point in No. 1 mine and at two points in No. 2 mine by P. M. Riefkin on April 17, 1911, as described below:

Sections of coal bed in Lenore No. 1 and No. 2 mines.

Section	A		B		C	
Laboratory No	12044		12045		12046	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	5½	..	..	..	..
Mother coal	Streak.		..	..	..	..
Coal	1	11	2	3¼	2	2¾
Coal, hard	0	3	0	3¾	0	4
Floor, hard, dark fire clay.						
Thickness of bed	2	7½	2	7	2	6¾
Thickness of coal sampled	2	7½	2	7	2	6¾

Section A (sample 12044) was measured at the face of left heading 3, off No. 2 drift.

Section B (sample 12045) was measured at the face of left heading 2, off No. 2 drift.

Section C (sample 12046) was measured at the face of left heading 1, off the main heading off No. 1 drift.

Notes.—At the time of sampling the coal at these mines was undercut by hand and shot down with FFFF black powder. Both mines load coal over the same tipple. There was one loading track with a capacity of 10 empty and 10 loaded cars. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. The mines were equipped to produce 200 tons a day, with an average daily output of 150 tons, all from advance workings. There was approximately 300 acres of unmined coal tributary to these entries. The two mines are equipped as a single mine.

MADERA. BUCHER MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12376, W20435, W20436, W20437, W20438, and W20439 (p. 86).

Mine.—Bucher, a drift mine in the Clearfield district, 2 miles north of Madera, on the Pennsylvania Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet 6 inches. The roof, a dark shale, is overlain by cannel coal 2 feet thick, above which is a sandstone cap rock. The floor is a hard shale 1 to 2 feet thick, underlain by a hard, rough clay about 10 feet thick.

The bed was measured and sampled at five points in the mine by H. I. Smith on June 9, 1911, as described below:

Sections of coal bed in Bucher mine.

Section	A		B		C		D		E	
Laboratory No	W20435		W20436		W20437		W20438		W20439	
Roof, sandstone, cannel coal, and dark shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	..	..	..	..	..	..	1	½	0	8½
Bony coal	[a] 0	2	[a] 0	½	[a] 0	1	..	..	..	..
Mother coal	..	..	..	..	..	..	..	..	0	½
Coal	0	7	0	11	0	11	..	..	0	3½
Cannel coal	0	1	0	1	0	1	0	1	0	1
Coal	0	11½	0	10½	0	11	0	8	0	10
"Mining slate"	[a] 0	¼	[a] 0	¼	Streak.		[a] 0	¼	[a] 0	1½
Coal	0	2	..	..	0	4½	[a] 0	2½	0	2½
"Sulphur"	..	..	..	..	0	¼	[a] 0	½	0	¼
Coal	1	1½	1	4	0	10¼	1	6	0	11
Floor, shale and fire clay.										
Thickness of bed	3	1¼	3	3	3	3	3	6¾	3	2¾
Thickness of coal sampled	2	11	3	2½	3	2	3	3½	3	1¼

[a] Not included in sample.

Section A (sample W20435) was measured at the face of ninth left entry, off first cross heading, 3,400 feet northeast of the mouth of the main entry.

Section B (sample W20436) was measured at the face of eighth right heading, 3,800 feet northeast of the mouth of the main heading.

Section C (sample W20437) was measured at the face of tenth right heading, 4,500 feet northeast of the mouth of the main entry.

Section D (sample W20438) was measured at the face of seventh left heading, 3,000 feet northeast of the mouth of the main entry.

Section E (sample W20439) was measured at the face of seventh right heading, 3,200 feet northeast of the mouth of the main entry.

A composite sample was made by combining face samples W20435 to W20439, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 12376.

Notes.—At the time of sampling the coal was undercut with machines and broken down with black powder. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track at the tipple with a capacity of about 40 cars. The mine had a capacity of 400 tons a day and an average daily output of 250 tons, of which the larger part was from advance workings. There was approximately 600 acres of unmined coal tributary to this opening.

MADERA. MORGAN RUN MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12368, W20402, W20403, W20404, W20405, and W20406 (p. 87).

Mine.—Morgan Run, a drift mine in the Clearfield district, 1 mile east of Madera on the Pennsylvania Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet 6 inches. The roof, a hard, dark, smooth shale, is overlain by 2 feet of cannel coal; above the cannel is sandstone. Floor, shale, underlain by clay.

The bed was measured and sampled at five points in the mine by H. I. Smith on June 8, 1911, as described below:

Sections of coal bed in Morgan Run mine.

Section	A		B		C		D		E	
Laboratory No.	W20402		W20403		W20404		W20405		W20406	
Roof, sandstone, cannel coal, and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	½	0	8½	0	2½	0	3	0	9
"Sulphur"	Streak.		Streak.		Streak.		Streak.		Streak.	
Coal	0	3½	0	2	0	9	0	3½	0	4
Mother coal	0	1	..	..	..	..	..	..	..	..
Cannel coal	..	..	0	2½	..	..	..	..	0	1
"Sulphur"	..	..	..	..	Streak.		[a] 0	½	..	..
Coal	0	7	0	3	..	..	0	6	0	11
Cannel coal	..	..	..	..	0	1¼	0	1	0	1
"Binder slate"	[a] 0	¾	[a] 0	½	Streak.		Streak.		[a] 0	1
Coal	0	3½	0	5½	1	3	0	10	1	3½
"Binder slate"	Streak.		[a] 0	1¼	Streak.		[a] 0	1	..	..
Coal	0	2	0	4½	0	4	0	10	..	..
"Sulphur"	Streak.		Streak.		..	..	Streak.		..	..
Coal	0	3	..	..	..	..	0	5½	..	..
"Sulphur" ball	[a] 0	½	..	..	[a] 0	½	..	..	..	..
Coal	0	7¾	1	0	0	10½	..	..	..	..
Bony coal	..	..	..	..	0	1	0	¾	..	..
Floor, shale.										
Thickness of bed	3	5½	3	3¾	3	7¾	3	5¼	3	6½
Thickness of coal sampled	3	4¼	3	2	3	7¼	3	3¾	3	5½

[a] Not included in sample.

Section A (sample W20402) was measured at the face of south entry 4, off right entry 5.

Section B (sample W20403) was measured at the face of east entry 3, off right entry 5.

Section C (sample W20404) was measured at the face of south entry 3, off right entry 5.

Section D (sample W20405) was measured at the face of south entry 1, off right entry 5.

Section E (sample W20406) was measured at the face of the main heading.

A composite sample was made by combining face samples W20402 to W20406, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 12368.

Notes.—At the time of sampling the coal was undercut by hand and broken down with black powder. There was one loading track at the tipple with a capacity of 16 empty and 16 loaded cars. Pickers were employed on the cars in loading. The mine was equipped to produce 400 tons a day; the average daily output was 380 tons. The unmined area was approximately 500 acres.

MORRISDALE. MORRISDALE No. 1, No. 2, AND No. 3 MINES.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12072 to 12082, inclusive, and 12090 and 12092 (pp. 87, 88).

Mines.—Morrisdale No. 1, Morrisdale No. 2, and Morrisdale No. 3 shaft mines, each about 130 feet deep, in the Clearfield district, 1 mile south of Morrisdale, on the New York Central & Hudson River Railroad and the Pennsylvania Railroad.

Coal beds.—Lower Kittanning or B, and the Middle Kittanning or C. Carboniferous age, Allegheny formation. The coal, as mined, has an average thickness of 4 feet 6 inches in the C bed, and about 3 feet 6 inches in the B bed. The roof of both beds is a sandy shale; the floor is a hard fire clay with smooth surface. The B bed was measured and sampled at three points in No. 1 mine and at three in No. 3 mine and the C bed at two points in No. 1 mine and at two in No. 2 mine by P. M. Riefkin on April 20 and 21, 1911, as described below:

Sections of B bed in Morrisdale mines.

Mine	Shaft No. 1.			Shaft No. 3.		
Section	A	B	C	D	E	F
Laboratory No.	12074	12075	12081	12076	12077	12078
Roof, shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Top coal	[a] 0 ¼	0 ⅛	.. ..	[a] 0 ½	[a] 0 ½	[a] 0 1
Coal	0 4½	0 5	.. ..	0 6	0 8	0 6½
"Sulphur"	Streak.	[a] 0 ½	.. ..	.. ..	.. ..	.. ..
Coal	0 6½	0 4½	0 10¼	.. ..	.. ..	.. ..
Bone	[a] 0 ½	[a] 0 3	[a] 0 3¾	[a] 0 ½	[a] 0 6½	[a] 0 7
Bony coal	[a] 0 4	[a] 0 2½	[a] 0 3¼	[a] 0 6	[a] 0 1½	.. ..
Coal	1 1	1 6	2 0	2 2	2 3½	2 8
"Sulphur"	[a] 0 ¼	.. ..	Streak.	[a] 0 ½	.. ..	.. ..
Coal	0 9½	.. ..	.. ..	0 10	.. ..	.. ..
"Sulphur"	Streak.	0 ½	0 ½	.. ..	.. ..	.. ..
Coal	0 10	1 5¼	0 8	.. ..	.. ..	.. ..
Floor, hard clay.						
Thickness of bed	4 ½	4 3⅜	4 1¾	4 1½	3 8	3 10½
Thickness of coal sampled	3 7½	3 9⅜	3 6¾	3 6	2 11½	3 2½

[a] Not included in sample.

Section A (sample 12074) was measured at the face of the main heading, 5,240 feet from the bottom of shaft No. 1.

Section B (sample 12075) was measured at the face of room 13, off eighth right heading off the main heading, 5,040 feet from the bottom of shaft No. 1.

Section C (sample 12081) was measured at the face of room 45, off fourth right entry, off the main heading, 5,200 feet from the bottom of shaft No. 1.

Section D (sample 12076) was measured at the face of room 5, off first left heading, off new slope tunnel, 1,400 feet from shaft No. 3.

Section E (sample 12077) was measured at the face of shaft No. 2 heading, main dip, 5,500 feet from shaft No. 3.

Section F (sample 12078) was measured at the face of the Morrisdale heading, 6,300 feet from shaft No. 3.

A composite sample was made of face samples 12074, 12075, and 12081. The results of an ultimate analysis of this sample are shown under laboratory No. 12090.

A composite sample was also made of face samples 12076, 12077, and 12078. The results of an ultimate analysis of this sample are shown under laboratory No. 12092.

Sections of C bed in Morrisdale mines.

Mine	Shaft No. 2.		Shaft No. 1.	
Section	A	B	C	D
Laboratory No.	12072	12073	12079	12080
Roof, shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Top, bony coal	0 1½	0 2	[a]0 3¾	Streak.
Coal	.. ..	.. ..	0 7½	0 6
Mother coal	.. ..	.. ..	.. ..	0 ⅛
Coal	.. ..	1 7½	1 7	1 4
"Sulphur"	.. ..	.. ..	0 ¼	.. ..
Coal, gray	.. ..	0 2½	.. ..	0 1
Coal	2 11½	2 0	2 5½	1 3½
Coal, gray	0 1	.. ..	.. ..	0 1
Coal	0 4¼	.. ..	.. ..	0 10
Shale	.. ..	.. ..	.. ..	[a]0 ¼
Mother coal	0 ¼	.. ..	.. ..	.. ..
Coal	0 8½	.. ..	.. ..	0 2
Bony coal	[a]0 1½	.. ..	.. ..	0 ⅛
Coal	0 6	.. ..	.. ..	0 2
Bottom coal	.. ..	[a]0 1	.. ..	.. ..
Floor, fire clay.				
Thickness of bed	4 10½	4 1	5 0	4 6
Thickness of coal sampled	4 9	4 0	4 8¼	4 5¾

[a] Not included in sample.

Section A (sample 12072) was measured at the face of the sixth heading, 4,500 feet from the bottom of shaft No. 2.

Section B (sample 12073) was measured at the face of room 9, off the second left heading, off Lucky strike, 2,200 feet from the bottom of shaft No. 2.

Section C (sample 12079) was measured at the face of the sump heading, 3,100 feet from shaft No. 1.

Section D (sample 12080) was measured at the face of the McKinley heading off Truck main heading, 5,850 feet from the bottom of shaft No. 1.

A composite sample was made by combining face samples 12072 and 12073. The results of an ultimate analysis of this sample are given under laboratory No. 12091.

A composite sample was also made by combining face samples 12079 and 12080. The results of an ultimate analysis of this sample are shown under laboratory No. 12082.

Notes.—At the time of sampling the coal was undercut with air punchers and shot down with black powder. There were four loading tracks at the tipple with a capacity of 120 empty and 160 loaded railroad cars. The entire production was shipped as run-of-mine; pickers were employed on the cars in loading. The mine was equipped to produce about 2,700 tons a day and had a daily output of 2,240 tons. About 75 per cent of the coal was derived from advance workings. It was planned to increase the production about 600 tons at an early date. There was approximately 4,000 acres of unmined coal tributary to these shafts. All of the mines loaded coal over the same tipple.

MUNSON. COLORADO NO. 5 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12109, 12110, 12111, 12112, 12113, and 12114 (pp. 88, 89).

Mine.—Colorado No. 5, a drift mine in the Clearfield district, at Munson, on the New York Central & Hudson River Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 4 feet. Roof, shale; floor, fire clay. Particles of the roof or floor did not become mixed with the coal in mining.

The bed was measured and sampled at five points in the mine by H. I. Smith on April 22, 1911, as described below:

Sections of coal bed in Colorado No. 5 mine.

Section	A		B		C		D		E	
Laboratory No.	12109		12110		12111		12112		12113	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Top coal	[a]0	9	[a]0	9½	[a]0	9½	[a]0	7½	[a]0	9½
Coal, bony	[a]0	4	[a]0	3	[a]0	4	[a]0	6	[a]0	3
Coal, gray	1	0	1	2½	0	6¼	0	5½	0	8
Coal	..	..	..	..	0	1¾	0	1	0	3
Coal, gray	..	..	..	..	0	3½	0	½	0	1
Coal	..	..	..	..	..	..	0	1½	..	..
Coal, gray	..	..	..	..	..	..	0	3	..	..
Coal	2	0	1	10½	2	0	2	1	1	8
"Sulphur"	Streak.		Streak.		0	¼	Streak.		0	½
Coal	..	..	0	1	0	3	0	1½	1	¾
Floor, fire clay.										
Thickness of bed	4	1	4	2½	4	4¼	4	3½	4	10¼
Thickness of coal sampled	3	0	3	2	3	2¾	3	2	3	9¾

[a] Not included in sample.

Section A (sample 12109) was measured at the face of sixth left entry, off the line heading.

Section B (sample 12110) was measured at the face of eighth left entry, off the main heading, 3,000 feet from the drift mouth.

Section C (sample 12111) was measured at the face of the main heading, 3,600 feet from the drift mouth.

Section D (sample 12112) was measured at the face of the line heading.

Section E (sample 12113) was measured at the face of sixth left heading, off the main heading.

A composite sample was made by combining face samples 12109 to 12113, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 12114.

Notes.—At the time of sampling the coal was undercut by chain machines in the lower part of the bed and broken down with black powder. At the tipple were two loading tracks with a capacity of 20 empty and 15 loaded cars. The cars were shifted twice daily by the railroad company. The coal was hand picked wherever necessary. The mine had a capacity of 600 tons, with a daily output of 350 tons, derived principally from advance workings.

OSCEOLA MILLS. FAIRMOUNT NO. 2 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. W17010 and W17012 (p. 89).

Mine.—Fairmount No. 2, a drift mine in the Clearfield district, 3 miles west of Osceola Mills on the Pennsylvania Railroad.

Coal bed.—Lower Freeport or D. Carboniferous age, Allegheny formation. Thickness, 5 to 6 feet. Roof, bone; floor, shale underlain by fire clay.

The bed was measured and sampled at two points in the mine by P. M. Riefkin on January 23, 1911, as described below:

Sections of coal bed in Fairmount No. 2 mine.

Section	A		B	
Laboratory No.	W17010		W17012	
Roof, bone.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	11½	1	4½
Mother coal	0	¼	..	..
"Sulphur"	..	..	0	¼
Coal	1	5½	1	6
"Sulphur"	0	¼	Trace.	
Coal	0	7½	2	4½
Floor, shale.				
Thickness of bed	5	1	5	3¼
Thickness of coal sampled	5	1	5	3¼

Section A (sample W17010) was measured at the face of room 5, off second left heading.

Section B (sample W17012) was measured at the face of room 14, off eighth right heading.

A composite sample was made by combining face samples W17010 and W17012. The results of an ultimate analysis of this sample are shown under laboratory No. 17111.

Notes.—At the time of sampling the coal was undercut by hand and shot down with black powder. There were no screens at the tipple, the entire output being shipped as run-of-mine. The average daily output was about 300 tons.

PHILIPSBURG. GUION MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12367, W20369, W20370, W20371, W20372, and W20373 (p. 89).

Mine.—Guion, a drift mine in the Clearfield district, 2 miles west of Philipsburg (Center County) on the Pennsylvania Railroad.

Coal bed.—Miller, B, or Lower Kittanning. Carboniferous age, Allegheny formation. Average thickness, 3 feet 4 inches. Roof, black shale 1 foot thick, overlain by roof coal 1 foot thick; above the roof coal is a sandstone cap rock. Floor, a hard sulphur-bearing rock 8 inches thick, underlain by soft smooth fire clay.

The bed was measured and sampled at five points in the mine by H. I. Smith on June 7, 1911, as described below:

Sections of coal bed in Guion mine.

Section	A		B		C		D		E	
Laboratory No	W20369		W20370		W20371		W20372		W20373	
Roof, sandstone, roof coal and black shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	0	9	..	..	[a] 0	2½	..	..	[a] 0	2½
Coal with flakes of sulphur on "slips"	..	..	..	..	[a] 0	4½	0	3½	..	..
Coal	..	..	1	2	..	..	1	0	0	5
Coal, hard	0	8	0	2	..	..	0	5	..	..
Shale	..	..	..	..	[a] 0	½	..	..	[a] 0	1
Coal	1	6	1	8	0	3½	1	5	0	5
"Sulphur"	..	..	..	..	[a] 0	½	..	..	..	..
Cannel coal	0	2½	0	2	..	..	0	4	..	..
Coal, hard	..	..	..	..	0	10½	..	..	0	8
Coal, soft, fibrous	..	..	..	..	0	11	..	..	0	9
"Sulphur"	..	..	..	..	Streak.		..	..	0	¼
Coal	..	..	..	..	0	7	..	..	0	11½
Floor, sulphur-bearing rock and fire clay.										
Thickness of bed	3	1½	3	2	3	4	3	5½	3	6¼
Thickness of coal sampled	3	1½	3	2	2	8	3	5½	3	2¾

[a] Not included in sample.

Section A (sample W20369) was measured at the face of first right heading, off right entry 2, 4,000 feet from the mine mouth.

Section B (sample W20370) was measured at the face of second right heading, 4,000 feet from the mouth of the main entry.

Section C (sample W20371) was measured at the face of the back heading, off right entry 1, 2,500 feet from the mouth of the main entry.

Section D (sample W20372) was measured 50 feet from the face of first left heading, off right entry 2.

Section E (sample W20373) was measured at the face of first left heading, off right entry 1, about 2,200 feet from the mouth of the main entry.

A composite sample was made by combining face samples W20369 to W20373. The results of an ultimate analysis of this sample are shown under laboratory No. 12367.

Notes.—At the time of sampling the coal was undercut with machines and shot down with black powder. There was one loading track, with a capacity of about 25 cars. There were no screens, the entire output being shipped as run-of-mine. The mine had a capacity of 300 tons a day, and an average daily output of about 250 tons. An increase in the production to 900 tons a day was contemplated. There was approximately 300 acres of unmined coal tributary to this opening.

SMOKE RUN. VIOLA MINE.

Sample.—Semibituminous coal; Windber field; analyses No. 11748, 11749, 11750, 11751, W18355, W18356, W18357, W18358, and W18359 (pp. 89, 90).

Mine.—Viola, a slope mine in Gulic Township, Clearfield district, 3½ miles southeast of Smoke Run, on the Little Muddy Run branch of the Pennsylvania Railroad.

Coal bed.—Lower Kittanning, Miller, or B. Carboniferous age, Allegheny formation. Average thickness, about 5 feet 4 inches. The bed is practically level. Roof, "draw slate" 18 inches thick, overlain by sandstone; floor, hard smooth fire clay. Cover, 100 to 125 feet.

The bed was measured and sampled at three points in the mine by L. M. Jones on February 23, 1911, as described below:

Sections of coal bed in Viola mine.

Section	A		B		C	
Laboratory No	11748		11749		11750	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, soft and coarse	0	5	..	..	0	10
Coal, soft and fine	1	4	1	4	0	7
Coal, bony	..	..	0	½	..	..
Coal	..	..	0	4	..	..
Coal, bony	[a]0	10	[a]0	10	[a]1	0
Coal, soft and fine	2	8	0	10	2	3¼
Bone (local)	..	..	..	..	0	½
"Sulphur" band	0	¼	[a]0	¼	..	..
Coal, soft	0	4¾	0	11¼	0	7½
"Sulphur" band	..	..	[a]0	½	..	..
Coal, soft and dirty	..	..	0	7	..	..
Floor, fire clay.						
Thickness of bed	5	8	5	½	5	4¼
Thickness of coal sampled	4	10	4	½	4	4¼

[a] Not included in sample.

Section A (sample 11748) was measured near the face of ninth left heading beyond room 18, 1,300 feet from the main haulage way.

Section B (sample 11749) was measured at the face of the second slant to the right near room 3.

Section C (sample 11750) was measured at the face of the main entry, 4,750 feet southwest of the mine mouth.

A composite sample was made by combining face samples 11748, 11749, and 11750. The results of an ultimate analysis of this sample are shown under laboratory No. 11751.

The bed was also measured and sampled at five points in the mine by P. M. Riefkin on March 7, 1911, as described below:

Sections of coal bed in Viola mine.

Section	A		B		C		D		E	
Laboratory No	W18355		W18356		W18357		W18358		W18359	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	7	1	7½	1	7¾	0	9½	1	3¾
"Sulphur"	..	..	..	..	..	..	0	¼	0	¼
Coal	..	..	..	..	..	..	0	10½	0	6½
Soft bone	[a]0	3	0	4½	[a]0	3	[a]0	2½	[a]0	4
Coal, bony	[a]0	8	0	5	[a]0	9	0	7½	[a]0	6
Coal	..	..	0	10½	..	..	..	..	..	..
"Blue binder"	..	..	[a]0	½	..	..	..	..	..	..
Coal	2	2½	1	3½	2	5¾	2	2¾	1	10
"Sulphur"	0	¼	[a]0	½	0	¾	[a]0	¼	[a]0	1
Coal	0	7½	0	4	0	5½	0	4	0	4½
Floor, shale.										
Thickness of bed	5	4¼	5	0	5	7¾	5	1¼	5	0
Thickness of coal sampled	4	5¼	4	11	4	7¾	4	10½	4	1

[a] Not included in sample.

Section A (sample W18355) was measured at the face of tenth left entry, 100 feet from room 12.

Section B (sample W18356) was measured near the face of the third slant entry, 100 feet above room 1.

Section C (sample W18357) was measured near the face of left heading 9.

Section D (sample W18358) was measured at the face of the main entry, 4,800 feet southwest of the mine mouth.

Section E (sample W18359) was measured at the face of the second slant entry, 100 feet above the third slant entry.

Notes.—The double-entry room-and-pillar system of mining is employed. At the time of sampling, the coal was undercut by hand and shot down with FFF black powder. There was one loading track at the tipple with a capacity of 20 empty and 17 loaded cars. There were no screens, the entire production being shipped as run-of-mine. Pickers were employed on the cars in loading. The average daily output was about 550 tons. There was 800 acres of unmined coal.

HUNTINGDON COUNTY.

ROBERTSDALE. ROBERTSDALE MINE.

Sample.—Semibituminous coal; Broad Top field; analyses Nos. 12115, 12116, 12117, 12118, 12119, and 12120 (p. 90).

Mine.—Robertsdale, a slope mine in the Broad Top district, at Robertsdale, on the East Broad Top railroad, with connections to the Pennsylvania Railroad at Mount Union.

Coal bed.—Fulton, Lower Kittanning, or B. Carboniferous age, Allegheny formation. Average thickness, 5 feet. Roof, dark, smooth shale; floor, hard, smooth fire clay.

The bed was measured and sampled at five points in the mine by H. I. Smith on April 25, 1911, as described below:

Sections of coal bed in Robertsdale mine.

Section	A		B		C		D		E	
Laboratory No	12115		12116		12117		12118		12119	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Bone and shale	[a] 0	1½	[a] 0	8	[a] 0	5	[a] 0	1	..	..
Coal	1	4	0	2	0	6	1	9	0	8
Coal, bony	[a] 0	1	[a] 0	¼	[a] 0	1	..	..	..	..
Coal	0	2¼	0	2	0	2	..	..	..	..
Coal, bony	[a] 0	2¼	[a] 0	¼	0	1	..	..	0	¼
Coal	0	3	0	8	1	1	..	..	0	3¾
Coal, gray	0	½	0	¼	0	¾	0	¼	..	..
Coal	0	11	0	11	1	0	0	8¾	1	½
"Middle rock"	[a] 1	1	[a] 0	8	[a] 0	6½	[a] 0	5	[a] 0	5
Coal	..	..	..	..	0	1	..	..	..	..
Bone and coal	[a] 0	2	[a] 0	2	[a] 0	1½	[a] 0	2	[a] 0	1
Coal	1	2¼	1	2½	1	9½	1	4	1	0
"Sulphur" and bone	[a] 0	1¾	0	¼	[a] 0	2	[a] 0	1	0	⅛
Coal	0	6¾	0	7	..	..	0	5	0	4
Bone and "sulphur"	..	..	[a] 0	½	..	..	[a] 0	3	[a] 0	1
Floor, hard fire clay.										
Thickness of bed	6	3¼	5	4	6	1¼	5	3	3	11⅝
Thickness of coal sampled	4	5¾	3	9	4	9¼	4	3	3	4⅝

[a] Not included in sample.

Section A (sample 12115) was measured at the face of north heading 6.

Section B (sample 12116) was measured at the face of the main west heading.

Section C (sample 12117) was measured at the face of south heading 6.

Section D (sample 12118) was measured at the face of south heading 8.

Section E (sample 12119) was measured at the face of south heading 5.

A composite sample was made by combining face samples 12115 to 12119, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 12120.

Notes.—At the time of sampling the coal was undercut by hand and shot down with FFFF and FFF black powder. At the tipple was one loading track with a capacity of 50 empty and 50 loaded cars. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. The mine had a capacity of about 900 tons a day with an average daily output of 825 tons, two-thirds of which was from advance workings. There was approximately 500 acres of unmined coal. The slope is connected under ground with the Woodvale shaft.

WOODVALE. WOODVALE MINE.

Sample.—Semibituminous coal; Broad Top field; analyses Nos. 12121, 12122, 12123, 12124, and 12125 (pp. 90, 91).

Mine.—Woodvale, a shaft mine in the Broad Top district at Woodvale, on the East Broad Top railroad, with connections to the Pennsylvania Railroad at Mount Union.

Coal bed.—Fulton, Lower Kittanning, or B. Carboniferous age, Allegheny formation. Average thickness, 5 feet. Roof, shale, overlain by sandstone; floor, fire clay.

The bed was measured and sampled at four points in the mine by H. I. Smith on April 24, 1911, as described below:

Sections of coal bed in Woodvale mine.

Section	A		B		C		D	
Laboratory No.	12121		12122		12123		12124	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	7½	1	7½	1	8¼	0	7
Mother coal and "sulphur"	0	½	..	..	0	½	0	¼
Coal	0	2¼	..	..	0	2	0	11¼
Bony coal	[a] 0	½	0	¼	0	⅜	0	¼
Coal	0	5	0	10	0	9	0	9½
Bony coal and "sulphur"	..	..	..	..	..	..	[a] 0	1
Coal	..	..	..	..	..	..	0	1½
Bony coal	0	½	..	..	[a] 0	1	[a] 0	1
Coal	0	10½	..	..	[a] 0	1½	0	10½
"Middle rock"	[a] 0	7½	[a] 0	7½	[a] 0	5¾	[a] 0	5
Bony coal, gray	..	..	[a] 0	2	0	1½	[a] 0	2
"Sulphur"	..	..	[a] 0	½	..	..	..	..
Coal	..	..	..	..	..	..	0	4
Soft parting	..	..	..	..	..	..	0	⅛
Coal	1	7½	0	9	0	10	0	10¼
Bony coal and "sulphur"	..	..	[a] 0	1½	[a] 0	1½	0	¼
Coal	..	..	0	4½	0	4	0	7
Floor, fire clay.								
Thickness of bed	4	5¾	4	6¾	4	9⅜	5	10⅞
Thickness of coal sampled	3	9¾	3	7¼	3	11⅝	5	1⅞

[a] Not included in sample.

Section A (sample 12121) was measured at the face of first south heading.

Section B (sample 12122) was measured at the face of the third south or upcreek heading.

Section C (sample 12123) was measured at the face of the second west tunnel.

Section D (sample 12124) was measured at the face of No. 7 or Bolinger heading.

A composite sample was made by combining face samples 12121 to 12124, inclusive, The results of an ultimate analysis of this sample are shown under laboratory No. 12125.

Notes.—At the time of sampling the coal was undercut by hand and broken down with FFFF and FFF black powder. There were two loading tracks at the tipple with a capacity of 50 empty and 50 loaded cars. The tipple was equipped with bar screens. The entire output was shipped as run-of-mine coal. Pickers were employed on the cars in loading. The daily output was 600 tons, which was from both advance workings and from pillars. There was approximately 800 acres of unmined coal. This mine is connected underground with the Robertsdale slope mine.

INDIANA COUNTY.

CLYMER. EMPIRE M (McKEAN) MINE.

Sample.—Bituminous coal; Punxsutawney field; analyses Nos. 12219, W19678, W19680, W19682, W19685, and W19687 (p. 91).

Mine.—Empire M (McKean,) a drift mine 1½ miles north of Clymer, on the New York Central Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet. Roof hard, smooth shale; floor, fire clay.

The bed was measured and sampled at five points in the mine by P. M. Riefkin on April 29, 1911, as described below:

Sections of coal bed in Empire M (McKean) mine.

Section	A		B		C		D		E	
Laboratory No	W19678		W19680		W19682		W19685		W19687	
	Ft.	*In.*	*Ft.*	*In.*	*Ft.*	*In.*	*Ft.*	*In.*	*Ft.*	*In.*
Roof, shale and bone.										
Coal	0	11	0	11	0	8½	1	0	1	½
Mother coal	..	..	..	..	0	⅛	..	..	..	..
Coal	..	..	..	..	0	4¼	..	..	..	..
Cannel coal	0	1	0	1½	0	¾	0	¾	0	½
Coal	0	8	1	0	0	8½	0	9	..	..
"Sulphur," soft	..	..	..	..	0	¼	Streak.		..	..
Coal	..	..	..	..	0	4½	0	5½	..	..
"Sulphur," hard	0	¼	0	¼	[a] 0	⅜	..	..	..	..
Mother coal	..	..	..	..	..	..	0	½	..	..
Coal	1	6½	0	7½	0	7½	1	½	1	6½
"Sulphur"	..	..	Streak.		Streak.		..	..	0	⅜
Mother coal	..	..	..	..	..	..	0	1	..	..
Coal	..	..	0	6	0	4¾	..	..	0	7½
Floor, soft fire clay, smooth.										
Thickness of bed	3	2¾	3	2¼	3	3½	3	5¼	3	3⅜
Thickness of coal sampled	3	2¾	3	2¼	3	3⅛	3	5¼	3	3⅜

[a] Not included in sample.

Section A (sample W19678) was measured at the face of room 56, off right entry 2, off the main entry, 4,000 feet east of the mine mouth.

Section B (sample W19680) was measured at the corner of a crosscut at room 40, off right entry 2, off the main entry, 3,000 feet southeast of the mine mouth.

Section C (sample W19682) was measured at the corner of the pillar of room 44, off right entry 1, off the main entry, 2,500 feet west of the mine mouth.

Section D (sample W19685) was measured near the face of the first parallel off the south heading, off right entry 1, off the main entry, 2,800 feet southwest of the mine mouth.

Section E (sample W19687) was measured at the face of the third parallel off the south heading, off right entry 1, off the main entry, 3,000 feet northwest of the mine mouth.

A composite sample was made by combining face samples W19678, W19680, W19685, and W19687. The results of an ultimate analysis of this sample are shown under laboratory No. 12223.

Notes.—At the time of sampling the coal was undercut by hand and shot down with FFFF black powder. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track with a capacity of 25 loaded and 50 empty cars. The mine was equipped to produce 700 tons a day; the average daily output was 360 tons, a large part of which was from advance workings. There was about 1,200 acres of unmined coal tributary to this opening.

CLYMER. EMPIRE R MINE.

Sample.—Bituminous coal, Punxsutawney field; analyses Nos. 12219, W19679, W19681, W19683, W19684, and W19686 (p. 91).

Mine.—Empire R, a drift mine, 1 mile north of Clymer on the New York Central & Hudson River Railroad.

Coal bed.—Lower Kittanning or B. Carboniferous age, Allegheny formation. Average thickness, 3 feet; cannel coal is persistent throughout the bed. Roof, smooth shale; floor, hard, smooth fire clay.

The bed was measured and sampled at five points in the mine by P. M. Riefkin on April 29, 1911, as described below:

Sections of coal bed in Empire R mine.

Section	A		B		C		D		E	
Laboratory No.	W19679		W19681		W19683		W19684		W19686	
Roof, shale and bony coal.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	8¼	..	..	0	5	0	4	..	..
"Sulphur" and mother coal	0	⅜	..	..	Streak.		0	⅝	..	..
Coal	0	5	1	½	0	7¼	0	7	0	8¾
Cannel coal, gray	0	¾	0	½	0	1	0	¾	0	1¼
Coal	0	7	0	7½	0	11¼	0	6½	..	..
"Sulphur" and mother coal	[a] 0	⅝	..	..	Streak.		Streak.		..	..
Coal	0	2	..	..	0	4¼	0	10¼	..	..
"Sulphur"	..	..	..	..	0	½	0	¼	..	..
Mother coal	0	¼	Streak.		..	..	..	..	..	..
Coal	1	2¼	1	5½	1	5½	0	9¼	2	4½
Floor, hard fire clay, smooth.										
Thickness of bed	3	2½	3	2	3	10¾	3	2⅝	3	2½
Thickness of coal sampled	3	1⅞	3	2	3	10¾	3	2⅝	3	2½

[a] Not included in sample.

Section A (sample W19679) was measured at a cross-cut at the corner of room 47, off left entry 1 off the main entry, 4,700 feet southeast of the mine mouth.

Section B (sample W19681) was measured on the pillar at room 28, off left entry 2 off the main entry, 2,200 feet east of the mine mouth.

Section C (sample W19683) was measured at the face of room 3, off right entry 1 off the main entry, 1,100 feet south of the mine mouth.

Section D (sample W19684) was measured at the face of room 11, off left entry 5 off the main entry, 1,900 feet southeast of the mine mouth.

Section E (sample W19686) was measured near the face of room 43, off left entry 4 off the main entry, 3,100 feet from the mine mouth.

A composite sample was made by combining face samples W19679, W19683, W19684, and W19686. The results of an ultimate analysis of this sample are given under laboratory No. 12219.

Notes.—At the time of sampling the coal was undercut by hand and shot down with FFFF black powder. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track with a capacity of 24 empty and 24 loaded cars. The mine was equipped to produce 500 tons a day and had a daily average production of 275 tons. About 75 per cent of the coal was derived from pillars. There was approximately 235 acres of unmined coal tributary to this entry.

GLEN CAMPBELL. INDIANA NO. 6 MINE.

Sample.—Bituminous coal; Punxsutawney field; analyses Nos. 12102, 12103, 12104, 12105, 12106, and 12107 (p. 92).

Mine.—Indiana No. 6, a drift mine at Glen Campbell, in Montgomery Township, 2½ miles north of Huvershurst Junction on the New York Central & Hudson River Railroad. The mine is connected with the main line of the New York Central by the Huvershurst & Southwestern Railroad.

Coal bed.—E or Upper Freeport. Average thickness, about 4 feet 10 inches. Roof, shale; floor, hard, smooth fire clay. Cover at points of sampling, 100 to 225 feet. The bed was sampled at five points in the mine by P. M. Riefkin on April 25, 1911, as described on the following page.

Sections of coal bed in Indiana No. 6 mine.

Section	A		B		C		D		E	
Laboratory No	12102		12103		12104		12105		12106	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	a0	3	a0	9	a0	6	a0	7½	a0	5¾
Coal	1	4	1	8	1	10	1	5	0	7
"Sulphur"	..	..	..	..	..	..	a0	¼	..	..
Mother coal	0	¼	0	¼	0	⅛	..	..	0	½
Coal	..	..	..	..	0	3	..	..	1	4
Mother coal	..	..	..	..	0	¼	..	..	0	⅛
Coal, hard	2	0	1	9	..	..	..	..	..	..
Coal	..	..	..	..	1	6	2	1	1	6½
Coal, bony	a0	¾	a0	1½	a0	1½	a0	1	a0	1
Coal	0	4	0	5	0	5	0	5	0	6¾
Coal, bony	a0	3¾	a0	1½	a0	2½	a0	1	a0	3
Floor, clay.										
Thickness of bed	4	3¾	4	10¼	4	11⅜	4	8¾	4	10⅝
Thickness of coal sampled	3	8¼	3	10¼	4	1⅜	3	11	4	⅞

a Not included in sample.

Section A (sample 12102) was measured at the face of left heading 7, off the main entry, 3,000 feet northwest of the mine mouth.

Section B (sample 12103) was measured at the face of room 10, off left entry 1, off the main heading, 1,000 feet northeast of the mine mouth.

Section C (sample 12104) was measured on the rib near the face of room 14 in right heading 7, off the main entry, about 2,000 feet northwest of the mine mouth.

Section D (sample 12105) was measured at the face of left heading 6, off the main entry, about 2,200 feet southwest of the mine mouth.

Section E (sample 12106) was measured near the face of room 1, off right heading 8, off the main entry, about 2,300 feet northwest of the mine mouth.

Notes.—The mine is worked on the room-and-pillar system. At the time of sampling, FFF black powder was used both for breaking the coal and brushing the roof or floor. The coal was undercut by hand. The output was 350 tons a day, with a capacity of 550 tons. About 40 per cent of the coal mined was from pillars and about 60 per cent from advance workings. There was about 100 acres of coal tributary to this opening.

HOMER CITY. TEARING RUN MINE.

Sample.—Bituminous coal; Punxsutawney field; analyses Nos. 12373, W20428, W20429, W20430, W20431, and W20432 (pp. 92, 93).

Mine.—Tearing Run, a drift mine, 1 mile northwest of Homer City, on the Pennsylvania Railroad.

Coal bed.—E, or Upper Freeport. Carboniferous age, Allegheny formation. Roof, shale about 10 to 12 inches thick, overlain by sandstone; floor, hard, smooth shale. Average thickness, 5 feet 6 inches. There is a persistent binder of bony coal about 15 inches below the shale roof.

The bed was measured and sampled at five points in the mine by P. M. Riefkin on June 9, 1911, as described below:

Sections of coal bed in Tearing Run mine.

Section	A		B		C		D		E	
Laboratory No	W20428		W20429		W20430		W20431		W20432	
Roof, sandstone and gray shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	3	1	3	1	5½	1	4½	1	6
"Sulphur"	a0	¾	a0	½	..	..	..	..	..	..
Coal	..	..	0	3	..	..	..	..	..	..
Bony coal	..	..	a0	10	a0	11	a0	9¼	a0	8¾
Coal	1	11½	1	6¾	1	7¾	1	1½	1	3½
"Sulphur"	..	..	0	¼	Streak.		0	¼	0	⅛
Coal	..	..	..	..	..	..	..	..	0	6
"Sulphur"	..	..	..	..	..	..	..	..	0	¼
Mother coal	0	½	..	..	..	..	..	..	..	..
Coal	0	4½	2	0	2	0	2	8	1	11½
Floor, shale.										
Thickness of bed	3	8¼	5	11½	6	¼	5	11½	6	⅛
Thickness of coal sampled	3	7½	5	1	5	1¼	5	2¼	5	3⅜

a Not included in sample.

Section A (sample W20428) was measured on the rib of room 10, off left entry 1, off the main heading.

Section B (sample W20429) was measured at the face of right entry 2, off the main entry.

Section C (sample W20430) was measured on the rib near the face of left entry 2, off the main entry.

Section D (sample W20431) was measured on stump No. 4, right entry 1, off the main entry.

Section E (sample W20432) was measured at the "shoo fly" (cut-through), 100 feet beyond left entry 3, off the main entry.

A composite sample was made by combining face samples W20428, W20429, W20430, and W20432. The results of an ultimate analysis of this sample are shown under laboratory No. 12373.

Note.—At the time of sampling the coal was undercut with machines and by hand, and shot down with FFF and FFFF black powder. At the tipple there was one loading track, with a capacity of 17 empty and 17 loaded cars. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. The mine had a capacity of 500 tons a day and an average daily output of about 350 tons, of which the larger part was from advance workings. An increase in the output to about 500 tons was contemplated. There was approximately 400 acres of unmined coal tributary to this entry.

JEFFERSON COUNTY.

SYKESVILLE. SYKESVILLE MINE.

Sample.—Bituminous (coking) coal; Punxsutawney field; analyses Nos. 12455 and 12458, and 12457 and 12471 (p. 93).

Mine.—Sykesville, a shaft mine 1 mile south of Sykesville, on the Bloomsburg & Sullivan Railroad.

Coal bed.—Lower Freeport or D. Carboniferous age, Allegheny formation. Average thickness, 5 feet 6 inches; dip, 2° southeast. The main roof is a strong sandstone which in the south workings is underlain by shale. In the north workings there is no shale and in places the sandstone cuts the coal entirely out. The immediate floor, a hard clay, is separated from a sandstone of poor quality by 4 inches of brown limestone. Cover at points of sampling, about 300 feet.

The bed was measured and sampled at two points in the mine, and the roof coal above the bed was measured and sampled at two points by H. I. Smith and J. T. Ryan, on July 24, 1911, as described below:

Sections of coal bed in Sykesville mine.

Section	A		B	
Laboratory No	12455		12458	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Bony coal	0	8½	0	8
Coal	0	8½	0	7½
Cannel-like bone	0	5	0	3
Coal	..	..	1	11
Bony coal	0	1	0	½
Coal	3	1	1	3½
"Mining slate"	[a]0	1½	[a]0	1½
Coal	0	9	0	9½
Floor, clay.				
Thickness of bed	5	10½	5	8½
Thickness of coal sampled	5	9	5	7

[a] Not included in sample.

Section A (sample 12455) was cut from the pillar at the last cut-through between eighth left main and the back heading, off main south heading.

Section B (sample 12458) was measured at the face of the third right butt heading off the eighth left, off the main south heading.

Sections of roof coal in Sykesville mine.

Section	A	B
Laboratory No.	12457	12471
Shale.	*Ft. in.*	*Ft. in.*
Coal	0 1	.. ..
Bony coal	0 1	.. ..
Coal	0 5¾	.. ..
Shale	[a]0 1	.. ..
Coal	0 1½	.. ..
"Sulphur"	0 ¼	.. ..
Coal	0 4	.. ..
Shale	[a]1 9	.. ..
Coal	0 10	0 8
Shale	[a]0 3	[a]0 2½
Coal	0 7	.. ..
Bone	0 2	0 11
Coal	0 10½	1 ½
Shale	[a]0 4½	[a]0 5
Lower Freeport bed.		
Thickness of bed	6 ½	.. ..
Thickness of coal sampled	3 7	2 7½

[a] Not included in sample.

Section A (sample 12457) was measured in a cavity in the roof between the third right butt and back heading, off the eighth main off the main south heading.

Section B (sample 12471) was measured in a cavity in the roof on the third right butt heading off the eighth left main, about 200 feet from the face of the main heading.

Notes.—The coal is mined by the room-and-pillar system. At the time of sampling the system in vogue was to drive the entries to the boundary line, turning the room necks as the entry was driven, and then drive the rooms and pull the ribs on the retreating system. The coal was undercut with compressed-air punching machines and broken down with a permissible explosive. The coal was picked on the cars. None of the coal was shipped as run-of-mine. Two-thirds of the coal passed over a 2½ by 4 inch screen. All the screenings were crushed and coked, about 600 tons being coked daily. There were two loading tracks with a capacity of 15 loaded cars and 15 empty cars. The daily output was 1,200 tons, all from advance workings. About 800 acres of this tract remained to be mined.

LACKAWANNA COUNTY.

DUNMORE. SLOAN MINE.

Sample.—Anthracite coal; Northern field; analysis No. 11440 (p. 93).

Mine.—Sloan, Dunmore, about 3 miles east of Scranton.

Coal bed.—Dunmore No. 2. Carboniferous age, Allegheny formation. The bed was sampled at one point by N. H. Darton on December 5, 1910. The sample represents 7 feet of coal. The sample was taken in connection with investigations of the occurrence of gases in coal beds.

LUZERNE COUNTY.

PITTSTON. COLLIERY NO. 14.

Sample.—Anthracite coal; Northern field; analysis No. 11441 (p. 93).

Mine.—Colliery No. 14, a shaft mine 2 miles southwest of Pittston, on the Erie Railroad.

Coal bed.—Pittston. Carboniferous age, Allegheny formation. Thickness, about 13 feet. A sample was taken by N. H. Darton on November 21, 1910, in the last chamber at the end of the Brannagan slope. The sample represents the full thickness of the bed, and does not contain any impurities, being all marketable coal.

Notes.—The mine is opened by slope, drift, and shaft. At the time of sampling the coal was mined by hand. The mine employed about 1,700 men.

WILKES-BARRE. DORRANCE MINE.

Sample.—Anthracite coal; Northern field; analysis No. 11782 (p. 93).

Mine.—Dorrance; a shaft mine in Wilkes-Barre Township about 1 mile west of Wilkes-Barre, with rail connections to the Lehigh Valley Railroad.

Coal bed.—Red Ash. Carboniferous age, Allegheny formation. Thickness, about 12 feet, all of which is marketable coal. Roof, soft sandstone and sandy shale; floor, fire clay. The sample was taken by N. H. Darton at the end of the first west gangway, 600 feet northwest of the shaft, and represents the entire section across the face.

Notes.—The mine is opened by a shaft 1,127 feet deep and the coal is mined by the pillar-and-chamber system. At the time of sampling the mine had an average production of 1,800 tons a day.

SOMERSET COUNTY.

ACOSTA. BELMONT No. 1 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12209, W19536, W19537, W19538, W19539, and W19543 (pp. 93, 94).

Mine.—Belmont No. 1, a drift mine in the Somerset district, one-quarter mile north of Acosta on the Boswell branch of the Baltimore & Ohio Railroad.

Coal bed.—C′, or Upper Kittanning. Carboniferous age, Allegheny formation. Average thickness, 3 feet 4 inches. Roof gray-black shale about 18 feet thick; floor, shale with an irregular surface.

The bed was measured and sampled at five points in the mine by G. S. Pope and E. W. Miller, April 26, 1911, as described below:

Sections of coal bed in Belmont No. 1 mine.

Section	A		B		C		D		E	
Laboratory Nos	W19536		W19537		W19538		W19539		W19543	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	9½	0	9¼	0	5½	0	9¼	0	9
Bone and shale	[a]0	1½	[a]0	1	..	..	0	¼	[a]0	1½
Coal	..	..	..	..	..	..	0	11½	..	..
"Sulphur" soft ball	..	..	..	..	0	¼	0	½	..	..
Coal	..	..	..	..	0	1½	0	1¼	..	..
"Sulphur" soft ball	..	..	..	..	..	..	0	½	..	..
Mother coal	..	..	..	..	0	¼	..	..	..	..
Coal	..	..	..	..	0	2½	..	..	..	..
Coal (breast harder than rest of section)	..	..	..	..	0	3½	..	..	..	..
Coal	2	7	2	7½	2	7½	1	3	2	4
Floor, shale.										
Thickness of bed	3	6	3	5¾	3	9½	3	3¼	3	2½
Thickness of coal sampled	3	4½	3	4¾	3	9½	3	3¼	3	1

[a] Not included in sample.

Section A (sample W19536) was measured at the face of room 4 off the main heading, 1,300 feet from the mine mouth.

Section B (sample W19537) was measured at the face of the third left entry, 60 feet from the main entry.

Section C (sample W19538) was measured at the face of room 3 off the first right entry.

Section D (sample W19539) was measured at the face of the second left entry, 250 feet from the main entry.

Section E (sample W19543) was measured at the face of the main heading, 1,550 feet from the mine mouth.

Notes.—At the time of sampling the coal was overcut by hand in the top of the bed, and FFFF and FFF black powder was used in breaking it down. There was one loading track with a capacity of 16 cars. There were no screens, the entire output being shipped as run-of-mine. Pickers were employed on the cars in loading. The coal is soft and friable. The mine was comparatively new, having started shipping coal in October, 1910. It had a capacity of 250 to 300 tons a day with an average daily output of 150 tons, the larger part of which was from advance workings. There was approximately 275 acres of unmined coal tributary to this opening.

BERLIN. CORONET NO. 3 MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 12214, W19532, W19533, W19534, W19535, and W19542 (p. 94).

Mine.—Coronet No. 3, a drift mine in the Somerset district, 1½ miles east of Berlin on the Berlin branch of the Baltimore & Ohio Railroad.

Coal bed.—"Smokeless" or Upper Freeport. Carboniferous age, Allegheny formation. Average thickness, 3 feet 6 inches. Roof, bony coal overlain by sandstone; floor, hard smooth sandstone.

The bed was measured and sampled at five points in the mine by G. S. Pope, April 25, 1911, as described below:

Sections of coal bed in Coronet No. 3 mine.

Section	A		B		C		D		E	
Laboratory No	W19532		W19533		W19534		W19535		W19542	
Roof, bony coal and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	10	1	4½	1	6	1	0	2	6
Shale	0	1	..	..	..	..	..	..	[a]0	2¾
Mother coal	..	..	0	¼	0	½	..	..	..	..
Coal	..	..	..	..	..	..	..	..	0	2¼
"Sulphur"	..	..	..	..	..	..	0	¼	0	¼
Coal	..	..	1	2	1	0	1	7	0	3
Shale	..	..	[a]0	¾	[a]0	1½	[a]0	2	..	..
"Sulphur"	..	..	..	..	..	..	..	..	0	¼
Coal	..	..	0	¾	..	..	0	3½	0	4¾
Bony coal	..	..	0	¼	..	..	0	⅛	..	..
Coal	0	10½	0	9	0	10½	0	6	..	..
Shale	..	..	..	..	..	..	..	..	[a]0	¾
Floor, sandstone.										
Thickness of bed	3	9½	3	5½	3	6½	3	6⅞	3	8
Thickness of coal sampled	3	9½	3	4¾	3	5	3	4⅞	3	4½

[a] Not included in sample.

Section A (sample W19532) was measured at the face of the eighth left entry, 15 feet beyond room 15 and 2,950 feet from the mouth of the main entry.

Section B (sample W19533) was measured at the face of the sixth right entry, 125 feet from the main heading and 2,500 feet from the mouth of the main entry.

Section C (sample W19534) was measured at the face of room 13, seventh left entry, 2,450 feet from the mouth of the main entry.

Section D (sample W19535) was measured on the pillar between rooms 2 and 3, off the main air course between third and fourth right entries, 1,400 feet from the drift mouth.

Section E (sample W19542) was measured at the face of seventh right entry, 50 feet beyond room 8, 2,800 feet from the mouth of the main entry.

Notes.—At the time of sampling the coal was undercut by hand and shot down with FFF and FFFF black powder. At the tipple there was one loading track, with a capacity of 12 cars. There were no screens, the entire output being shipped as run-of-mine coal. Pickers were employed on the cars in loading.

The coal near the roof gradually becomes harder and merges into bone, without any distinct line of separation between the beds.

The mine was equipped to produce about 300 tons a day, and had an average daily output of 150 tons. The coal was derived from both advance and pillar workings. The mine had been in operation about six years.

Confluence. Linmer Mine.

Sample.—Bituminous coal; Somerset field; analysis No. 13631 (p. 94).

Mine.—Linmer; a drift mine about 4½ miles southeast of Confluence, on Whites Creek, near Beachly post office, on a branch of the Baltimore & Ohio Railroad.

Coal bed.—C′ or Upper Kittanning. Carboniferous (Pennsylvanian) age, Allegheny formation. Average thickness, about 4 feet 3 inches. Roof, sandstone; floor, hard clay.

The bed was measured and sampled by C. T. Lupton on March 29, 1912. The sample represented a 3-foot 6½-inch section of coal.

The sample was taken at the end of the back subentry, off main entry 1, at a point about 2,500 feet southeast of mine mouth.

Elk Lick. Engle Mine.

Sample.—Semibituminous coal; Windber field; analysis No. 13983 (p. 95).

Mine.—Engle; about 2½ miles east of Elk Lick on Pine Run.

Coal bed.—D (?) Carboniferous (Pennsylvanian) age; Allegheny formation. Average thickness, about 3 feet. Roof and floor, shale. The strata at this locality dip about 13° W. and strike N. 38° E.

The bed was measured and sampled by C. T. Lupton on April 2, 1912, as described below:

Section of coal bed in Engle mine.

Laboratory No	13983	
Roof, shale.	*Ft.*	*in.*
Coal	2	4
Bone and coal alternating	[a]0	3
Coal	0	6½
Floor, shale.		
Thickness of bed	3	1½
Thickness of coal sampled	2	10½

[a] Not included in sample.

The sample was collected in a room in this "country bank," about 600 feet east of the mine mouth.

Elk Lick. Grassy Run No. 1 Mine.

Sample.—Semibituminous coal; Somerset field; analysis No. 13664 (p. 95).

Mine.—Grassy Run No. 1, about 2½ miles northwest of Elk Lick, on a branch of the Baltimore & Ohio Railroad.

Coal bed.—Four-foot Carboniferous (Pennsylvanian) age; Monongahela formation; about 35 feet stratigraphically above the Pittsburgh bed. Average thickness, about 4 feet 6 inches. Roof, shale 1 foot 3 inches, overlain by "wild coal," floor, sandy bone.

The bed was measured and sampled at one point in the mine by C. T. Lupton on April 1, 1912, as described below:

Section of coal bed in Grassy Run No. 1 mine.

Laboratory No	13664	
Roof, shale.	*Ft.*	*in.*
Coal	2	0
Bone	[a]0	1
Coal	2	6
Floor, sandy bone.		
Thickness of bed	4	7
Thickness of coal sampled	4	6

[a] Not included in sample.

The sample was taken in a room about 1,200 feet southwest from the mine mouth.

HOLSOPPLE. LENORE MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 15078, 15079, 15080, 15081, and 15082 (p. 95).

Mine.—Lenore, a shaft mine, one-eighth of a mile north of Holsopple station, on the Baltimore & Ohio Railroad.

Coal bed.—B or Miller. Carboniferous age, Allegheny formation. Thickness, 3 feet to 4 feet, fairly uniform. The bed has no regular shale or sulphur partings. The roof is a sandstone which does not fall or get mixed with the coal. Floor, hard smooth underclay and black "rash." Cover at points of sampling, 150 to 250 feet.

The bed was measured and sampled at four points in the mine by G. H. Deike and Edwin Higgins on November 6, 1912, as described below:

Sections of coal bed in Lenore mine.

Section	A		B		C		D	
Laboratory No.	15078		15079		15080		15081	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, hard, bright	0	7	0	9	0	9	0	8
Coal, dull gray	0	⅛	0	¼	0	1½	0	¼
Coal, soft, bright	1	1	..	...	0	11½	0	8
"Sulphur"	[a]0	1	..	...	Streak.		0	$\frac{1}{16}$
Coal, soft, bright	1	7	2	9¾	1	2¼	1	7¾
Mother coal	..	...	..	...	0	¼	..	...
Coal, soft, bright	..	...	..	...	0	3	..	...
Floor, underclay.								
Thickness of coal bed	3	4⅛	3	7	3	3½	3	$\frac{1}{16}$
Thickness of bed sampled	3	3⅛	3	7	3	3½	3	$\frac{1}{16}$

[a] Not included in sample.

Section A (sample 15078) was measured on rib in right entry 1, 200 feet from main entry.

Section B (sample 15079) was measured at the face of south entry 3.

Section C (sample 15080) was measured 30 feet from the face of the main entry.

Section D (sample 15081) was measured 30 feet from the face of south entry 7.

A composite sample was made by combining samples 15079, 15080, and 15081. The results of an ultimate analysis of this sample are given under laboratory No. 15082.

Notes.—The mine is opened by a shaft 87 feet deep and the coal is mined by the room-and-pillar method. At the time of sampling the coal was undercut by compressed-air cutting machines and by hand and blasted down with FFF black blasting powder. A permissible explosive was used for brushing the floor and roof. The entire output was shipped as run-of-mine. Pickers were employed on the car. There were two loading tracks with a capacity of 20 empty cars and 30 loaded cars. The average daily output was 200 tons with a maximum day's run of 400 tons. The capacity was estimated to be 600 tons a day.

HOLSOPPLE. ONEIDA MINE.

Sample.—Semibituminous coal; Windber field; analyses Nos. 15073, 15074, 15075, 15076, and 15077 (p. 95).

Mine.—Oneida, a slope mine one-eighth of a mile north of Holsopple, on the Baltimore & Ohio Railroad.

Coal bed.—Known as the Upper Kittanning or C′, Carboniferous age, Allegheny formation. Thickness, 3 feet 6 inches to 4 feet; average, 3 feet 8 inches. Roof, heavy sandstone; floor, hard limestone. Particles from the roof or floor did not become mixed to any extent with the coal in loading.

The bed was measured and sampled at four points in the mine by G. H. Deike and Edwin Higgins on November 6 and 7, 1912, as described below:

Sections of coal bed in Oneida mine.

Section	A		B		C		D	
Laboratory No.	15073		15074		15075		15076	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, soft, bright, friable	0	9	0	11	0	11½	1	0
Coal, hard, gray, bony	0	4	0	⅜	0	½	0	1
Coal, bright, friable	..	..	..	...	0	2	..	..
Coal, hard, gray	..	..	..	...	0	6½	..	..
Coal, hard, bright	2	3	1	7	1	9½	2	11
"Sulphur" streak	..	..	0	½	0	½	..	..
Coal, hard, bright	..	..	0	11	..	..	..	..
Coal	..	..	..	..	0	1	..	..
Floor, limestone.								
Thickness of bed	3	4	3	5⅞	3	7½	4	0
Thickness of coal sampled	3	4	3	5⅞	3	7½	4	0

Section A (sample 15073) was measured on a pillar in room 6 off left entry 5

Section B (sample 15074) was measured at the face of room 3 off left entry 6.

Section C (sample 15075) was measured 15 feet from face of left entry 5.

Section D (sample 15076) was measured at the face of room 23 off left entry 3.

A composite sample was made by mixing face samples 15074, 15075, and 15076. The results of an ultimate analysis of this sample are given under laboratory No. 15077.

Notes.—At the time of sampling the coal was undercut by hand and with compressed-air punching machines and shot down with FFF black powder. Permissible explosives were used for blasting the roof or floor. The tipple was used both by the Oneida and Lenore mines but the coal was not mixed, as each mine had a separate dump. All coal was shipped as run-of-mine. Pickers were employed on the cars. There were two loading tracks at the tipple with a capacity of 20 empty and 30 loaded railroad cars. The average daily output was 200 tons, the maximum day's run being 300 tons.

JEROME. JEROME No. 2 MINE

Sample.—Semibituminous coal; Windber field; analyses Nos. 12210, W19638, W19639, and W19642 (p. 95).

Mine.—Jerome No. 2, a shaft mine in the Somerset district, at Jerome, on the Somerset & Cambria branch of the Baltimore & Ohio Railroad.

Coal bed.—C′, or Upper Kittanning, Carboniferous age, Allegheny formation. The coal has an average thickness of 6 feet (including 6 inches of bone). The bone and shale are persistent at the top of the bed and irregular in the body of the bed. The roof is a bony coal averaging 6 inches in thickness overlain by a limestone capping. Floor, hard smooth sandstone.

The bed was measured and sampled at one point in the mine on May 22, 1911, and at 3 points, on April 27, 1911, by G. S. Pope, as described below:

Sections of coal bed in Jerome No. 2 mine.

Section	A		B		C	
Laboratory No.	W19638		W19639		W19642	
Roof, bony coal and limestone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Bone and shale	[a] 0	5½	[a] 0	6½	[a] 0	4
Coal	1	1½	1	3	1	0
Bone, shale, and coal	[a] 0	9	[a] 0	1	[a] 0	11
Coal	4	2	4	2	3	9½
Floor, sandstone, hard.						
Thickness of bed	6	6	6	½	6	½
Thickness of coal sampled	5	3½	5	5	4	9½

[a] Not included in sample.

Section A (sample W19638) was measured at the face of south entry 7 off east entry 1½.

Section B (sample W19639) was measured at the face of south entry 4 off east entry 2.

Section C (sample W19642) was measured at the face of east entry 2.

Sample 12210 was measured at the face of east entry 1.

A composite sample was made by combining face samples W19638, W19639, and W19642. The results of an ultimate analysis of this sample are shown under laboratory No. 12224.

Notes.—In April, 1911, the coal was undercut with machines and by hand and was broken down with a permissible explosive. The mine was well equipped. The tipple had 4 loading tracks with a capacity of 80 cars. The coal was screened with bar screens 10 feet long, with 3-inch openings; 75 per cent of the coal passed through. The coal was hand picked both on the conveyor and on the car in loading. The mine had a capacity of 1,700 tons a day with an average daily output of 1,020 tons, which was largely from advance workings. It was planned to increase the production to about 1,500 tons. The operating company owned or controlled about 1,000 railroad cars.

PINEHILL. CONSOLIDATION NO. 112 MINE.

Sample.—Semibituminous coal; Somerset field; analyses Nos. 13981, 13982, and 14037 (p. 96).

Mine.—Consolidation No. 112, at Pinehill, on the Berlin branch of the Baltimore & Ohio Railroad.

Coal bed.—"Little Pittsburgh." Carboniferous (Pennsylvanian) age; in the upper part of the Conemaugh formation, about 147 feet stratigraphically below the Pittsburgh bed. Average thickness, about 3 feet 4 inches. Strata are flat-lying, with a maximum dip of about 5 per cent. Roof, shale grading into sandstone above; floor, carbonaceous shale with some clay.

The bed was sampled and measured by C. T. Lupton at two points on April 3, 1912, as described below:

Sections of coal bed in Consolidation No. 112 mine.

Laboratory Nos.	13981		13982	
	Ft.	*in.*	*Ft.*	*in.*
Roof, sandy shale, grading into sandstone above.				
Coal (usually not mined)	..	..	[a]0	7
Shale	..	..	[a]0	8
Coal	2	6	2	0
Bone, variable	0	¼	[a]0	½
Coal, with occasional small lenses of pyrite	1	¾	1	1
Floor, shale and clay.				
Thickness of bed	3	7	4	4½
Thickness of coal sampled	3	7	3	1

[a] Not included in sample.

Sample 13981 was taken in room 44. right subentry 8 off the west main entry, about 3,600 feet west of the mine mouth.

Sample 13982 was taken in room 8, right subentry 1 off left entry 12, about 5,000 feet southwest of the mine mouth.

A composite sample was made by combining face samples 13981 and 13982. The results of an ultimate analysis of this sample are shown under laboratory No. 14037.

PINEHILL. CONSOLIDATION NO. 113 MINE.

Sample.—Semibituminous coal; Somerset field; analysis No. 13984 (p. 96).

Mine.—Consolidation No. 113; at Pinehill, on the Berlin branch of the Baltimore & Ohio Railroad.

Coal bed.—"Stoner." Carboniferous age (Pennsylvanian) age; Conemaugh formation; about 290 feet stratigraphically below the Pittsburgh bed. Average thickness, about 4 feet. Roof, shale grading upward into sandstone.

The sample was collected by C. T. Lupton on April 3, 1912, from a mine car brought directly from room 2, right entry 10, about 3,700 feet southwest from the mine mouth. Information regarding the thickness of the bed as described below was furnished by C. W. Hoffa, superintendent of the mine.

Section of coal bed in Consolidation No. 113 mine, at Pinehill.

	Ft.	in.
Roof, shale, grading into sandstone above.		
Coal	3+	0
Shale (variable)	0	3
Coal	0	11
Floor (not stated).		
Thickness of bed	4	2

RALPHTON. RALPHTON NO. 3 MINE.

Sample.—Semibituminous coal; Somerset field; analyses Nos. W19627, W19628, W19629, W19630, and W19631 (p. 96).

Mine.—Ralphton No. 3, a drift mine one-half mile east of Ralphton on the Boswell branch of the Cambria and Somerset division of the Baltimore & Ohio Railroad.

Coal bed.—C′ or Upper Kittanning. Carboniferous age, Allegheny formation. The coal has an average thickness of 3 feet 5 inches. The roof is a strong gray shale between which and the main coal bed is a seam of roof coal 3 to 4 inches thick. The floor is a smooth, hard limestone.

The bed was measured and sampled at five points in the mine by G. S. Pope on April 29, 1911, as described below:

Sections of coal bed in Ralphton No. 3 mine.

Section	A		B		C		D		E	
Laboratory No	W19627		W19628		W19629		W19630		W19531	
Roof, shale.	Ft.	in.	Ft.	in.	Ft.	in.	Ft.	in.	Ft.	in.
Shale, bone, and coal	[a]0	4½	[a]0	3	[a]0	4	[a]0	3	[a]1	4
Coal	..	..	0	1	..	..	..	..	..	..
Shale	[a]0	4	0	¼	..	..	..	..	..	..
Coal	..	..	0	9	0	9½	..	..	0	7½
Bone	..	..	0	½	0	¼	..	..	0	¼
Coal	3	3	2	7	2	6½	3	6½	2	8½
Floor, limestone.										
Thickness of bed	3	11½	3	8¾	3	8¼	3	9½	4	8¼
Thickness of coal sampled	3	3	3	5¾	3	4¼	3	6½	3	4¼

[a] Not included in sample.

Section A (sample W19627) was measured at the third crosscut off south entry 2, about 3,050 feet from the mine mouth.

Section B (sample W19628) was measured at the face of room 11 off the straight main entry, about 3,150 feet from the mine mouth.

Section C (sample W19629) was measured at the face of room 4 off right entry 6, about 2,800 feet from the mine mouth.

Section D (sample W19630) was measured at the face of room 29 off right heading 4, about 2,900 feet from the mine mouth.

Section E (sample W19631) was measured 20 feet inside the third crosscut off south entry 3, about 3,700 feet from the mine mouth.

Notes.—At the time of sampling the coal was undercut by hand and shot down with FFF and FFFF black powder. The tipple had one loading track with a capacity of 22 cars. There were no screens, the entire output being loaded as run-of-mine. Pickers were employed on the cars in loading. The mine had a capacity of 1,000 tons and an average daily output of 850 tons, most of which was from advance workings.

RALPHTON. RALPHTON NO. 4 MINE.

Sample.—Semibituminous coal; Somerset field; analyses Nos. 12202, 12218, W19632, W19633, W19635, W19636, and W19637 (p. 97).

Mine.—Ralphton No. 4, a drift mine, 1½ miles north of Ralphton, on the Baltimore & Ohio Railroad.

Coal bed.—C′ or Upper Kittanning. Carboniferous age, Allegheny formation. The coal has an average thickness of 3 feet 10 inches and is relatively free from partings. Roof, firm, smooth shale; floor, smooth, hard limestone.

The bed was measured and sampled at five points in the mine by G. S. Pope on April 28, 1911, as described below:

Sections of coal bed in Ralphton No. 4 mine.

Section	A		B		C		D		E	
Laboratory No	W19632		W19633		W19635		W19636		W19637	
Roof, smooth shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	3	3	10¾	3	7½	3	7	3	11
Hard gray coal inclined to bone	0	1	..	..	..	..	..	..	..	..
Coal	2	7½	..	..	..	..	..	..	..	..
Floor, hard smooth limestone.										
Thickness of bed	3	11½	3	10¾	3	7½	3	7	3	11
Thickness of coal sampled	3	11½	3	10¾	3	7½	3	7	3	11

Section A (sample W19632) was measured on a pillar between the south main heading and the air course opposite right entry 1.

Section B (sample W19633) was measured at the face of the south main entry.

Section C (sample W19635) was measured at the face of room 7 off the south main entry.

Section D (sample W19636) was measured on pillar 1 at the second "shoo-fly" (crossover) off main entry 1.

Section E (sample W19637) was measured at the face of left entry 4 off the south main entry.

Notes.—At the time of sampling the coal was undercut in the lower part of the bed by hand and was shot down with black powder. The tipple had one loading track with a capacity of 15 railroad cars. There were no screens, the output being loaded as run-of-mine. The mine had a capacity of 450 tons and an average daily output of 400 tons.

WASHINGTON COUNTY.

AVELLA. PENOBSCOTT MINE.

Sample.—Bituminous coal; Pittsburgh field; analyses Nos. 15673 and 15674 (p. 97).

Mine.—Penobscott, a drift mine, 2 miles west of Avella, on Wabash-Pittsburgh Terminal Railway, near State line.

Coal bed.—Pittsburgh. Carboniferous age, Monongahela formation. Thickness averages about 4½ feet where measured. In this vicinity this coal bed has been locally eroded and replaced by sandstone. The roof left in mining is coal, the top clay above the main bed of coal being weak.

The bed was measured and sampled at two points in the mine by D. D. Condit on January 8, 1913, as described below:

Sections of bed in Penobscott mine.

Section	A		B	
Laboratory No.	15673		15674	
Roof, coal 3 to 6 inches.	*Ft.*	*in.*	*Ft.*	*in.*
Clay	[a]1	1	[a]1	0
Coal	2	2½	2	3
Clay	[a]0	¼	[a]0	¼
Coal	[a]0	3½	[a]0	3½
Clay	[a]0	¾	[a]0	¼
Coal	1	8½	1	7
Floor, clay and shale.				
Thickness of bed	5	4½	5	2
Thickness of coal sampled	3	11	3	10

[a] Not included in samples.

Section A (sample 15673) was measured in room 15, eighth left entry off main entry, 2,500 feet northwest of mine mouth. Cover at point of sampling, 80 feet.

Section B (sample 15674) was measured in a room off seventh right entry, 3,100 feet northeast of mine mouth. Cover, 70 feet.

BAIRD. SCHOENBERGER MINE.

Sample.—Bituminous coal; Pittsburgh field; analysis Nos. 11164, 11165, 11166, 11167, 11168, 11169 and 11170 (pp. 97, 98).

Mine.—Schoenberger, a drift mine at Baird, a station on the Monongahela division of the Pennsylvania Railroad.

Coal bed.—Pittsburgh, Carboniferous age, Monongahela formation. Average thickness, 5 feet 10 inches. The roof is a "draw slate" 8 inches thick, above which is a shale; 15 feet above the shale is the main cap rock. The floor is a shale, below which is a limestone. Dip, about 1 per cent southwest.

The bed was sampled and measured at six points in the mine by A. A. Straub on November 3, 1910, as described below:

Sections of coal bed in Schoenberger mine.

Section	A		B		C		D		E		F	
Laboratory No	11164		11165		11166		11167		11168		11169	
Roof, slate.	*Ft.*	*In.*	*Ft.*	*In.*	*Ft.*	*In.*	*Ft.*	*In.*	*Ft.*	*In.*	*Ft.*	*In.*
Coal, bright	3	4⅜	2	7½	2	1	2	2¼	1	10¼	3	2¼
"Sulphur" band	..	..	[a]0	⅜	0	¾	[a]0	⅜	0	⅜	..	..
Coal, bright	..	..	0	9½	0	11¾	0	2½	0	7¼	..	..
Shale band	[a]0	½	..	..	[a]0	¼	..	..	..	..	..	..
"Sulphur" band	..	..	0	½	..	..	0	½	0	½	..	..
Coal, bright	0	2½	0	2¾	0	2⅝	0	1	..	..	..	..
"Sulphur" band	..	..	..	..	..	..	0	⅜	..	..	..	..
Coal, bright	..	..	..	..	..	..	0	8¼	0	9¾	..	..
Shale band	[a]0	¾	[a]0	¾	[a]0	⅜	[a]0	¾	[a]0	¾	[a]0	¾
Coal, bright	..	..	0	1½	..	..	0	3	0	2¼	0	1½
"Sulphur" band	..	..	0	⅜	..	..	..	..	..	..	..	..
Shale band	..	..	..	..	..	..	[a]0	⅜	[a]0	¼	[a]0	½
Coal, bright	..	..	1	2	..	..	0	6¼	1	6¼	..	..
"Sulphur" band	..	..	0	⅜	..	..	0	⅜	0	⅜	..	..
Coal, bright	1	10⅛	0	9½	1	10⅛	1	5⅜	0	5½	1	9
Floor, shale and limestone.												
Thickness of bed	5	6¼	5	9⅝	5	2⅞	5	7½	5	6¾	5	2
Thickness of coal sampled	5	5	5	8¾	5	2¼	5	6⅜	5	5¾	5	¾

[a] Not included in sample.

Section A (sample 11164) was measured at the face of room 10, entry 32 off face entry 12, about 4,000 feet southeast of the mine mouth.

Section B (sample 11165) was measured at the face of room 4, entry 32 off face entry 11, about 4,000 feet southeast of the mine mouth.

Section C (sample 11166) was measured at the face of the right main entry, about 7,000 feet south of the mine mouth.

Section D (sample 11167) was measured at the face of room 11, entry 32 off the right main entry, about 6,000 feet south of the mine mouth.

Section E (sample 11168) was measured at the face of face entry 11, about 4,800 feet southeast of the mine mouth.

Section F (sample 11169) was measured on a pillar, in right entry 29 off the main entry, about 5,500 feet south of the mine mouth.

Notes.—The coal is mined by the double entry, room-and-pillar system. At the time of sampling electric chain-cutting machines were used and the coal was broken down with FF black powder. The mine had a capacity of about 3,000 tons a day, 75 per cent of which was from advance workings. There were 4 loading tracks with a capacity of 65 cars at the tipple. Pickers were employed on the cars when coal was being loaded. About 10 per cent of the coal was shipped as run-of-mine and remainder was screened to various commercial sizes. There was about 250 acres of unmined coal tributary to this drift mouth.

FINLEYVILLE. CINCINNATI MINE.

Sample.—Bituminous coal; Pittsburgh field; analyses Nos. 17082, 17083, 17084, and 17085 (p. 98).

Mine.—The Cincinnati mine is opened by a drift about two miles northeast from Courtney and a manway at Mingo 2 miles southeast of Finleyville.

Coal bed.—Pittsburgh, Carboniferous age, Monongahela formation. Thickness, 4 feet 8 inches to 6 feet 8 inches, fairly uniform. In the center of the bed are two characteristic clay partings about three inches apart. The roof is a "draw slate," over which there is a roof coal and shale. The floor is an underclay, beneath which is a limestone bedrock.

The bed was measured and sampled at three points in the mine by H. I. Smith, H. D. Mason, jr., and W. A. Raudenbush on May 7, 1913, as described below:

Sections of coal bed in Cincinnati mine.

Section	A		B		C	
Laboratory No	17082		17083		17084	
Roof, "draw slate."	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	4½	1	6½	1	7½
Bony coal	0	½	0	½	0	1
Coal	1	9½	0	10	1	7
Mother coal and "sulphur"	0	⅛	0	½	..	..
Coal	0	3½	1	½	..	..
Shale binder	Streak.		[a]0	¼	0	¼
Coal	0	3½	[a]0	3½	0	2½
Shale binder	0	¼	[a]0	¼	0	¾
"Sulphur" ball on one side	..	..	..	..	0	3
Coal	0	7½	0	2	0	3¾
Mother coal	0	½	0	¼	0	¾
Coal	1	2	1	9	1	9
Bottom coal	..	..	[a]0	4	0	4½
Floor, underclay.						
Thickness of bed	5	8⅜	6	1½	6	3½
Thickness of coal sampled	5	8⅜	5	5¼	6	3½

[a] Not included in sample.

Section A (sample 17082) was measured 60 feet outby the last cut-through on butt entry 6 off face entry 14, about 6,200 feet northwest of the mouth of Mingo slope entry.

Section B (sample 17083) was measured 60 feet inby room 2 on butt entry 10 off face entry 14, about 6,800 feet northwest of the mouth of Mingo slope entry.

Section C (sample 17084) was measured inby the first crosscut on butt entry 12 off face entry 14, about 7,400 feet northwest of the mouth of Mingo slope entry.

A composite sample was made by mixing face samples 17082, 17083, and 17084. The results of an ultimate analysis of this sample are given under laboratory No. 17085.

Notes.—The coal is mined by the room-and-pillar system. At the time of sampling it was undercut with electric cutting machines in the bottom of the bed and was broken down with permissible explosives. The cuttings were loaded out with the coal. The "draw slate" is treacherous and must be broken down and loaded out after each cut of coal. The entire output of the mine went to a tipple on the Monongahela River. Although this mine is one of the first on the Monongahela River it still contained considerable coal.

MONONGAHELA CITY. CATSBURG MINE.

Sample.—Bituminous coal; Pittsburgh field; analyses Nos. 11157, 11158, 11159, 11160, 11161, 11162, 11163, 11172, 11541, and 12281 (pp. 98, 99.)

Mine.—Catsburg, a drift mine one-fourth of a mile south of Monongahela City with railroad connections to the Monongahela River division of the Pennsylvania lines.

Coal bed.—Pittsburgh. Average thickness, 5 feet 6 inches. Dip, 1 per cent southeast. The roof is a "draw slate," about 8 inches thick, above which is a fair quality of roof coal 6 to 12 inches thick. Floor, soft fire clay. The bed was measured and sampled at six points in the mine by J. J. Rutledge and A. A. Straub on November 1, 1910, as described below:

Sections of coal bed in Catsburg mine.

Section	A		B		C		D		E		F	
Laboratory No.	11157		11158		11159		11160		11161		11162	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bright and hard	2	3	2	11⅛	2	9¼	3	1	2	8¾	3	2
"Sulphur" band	0	½	..	..	..	..	..	..	..	..	..	..
Coal	0	8½	..	..	..	..	..	..	..	..	..	..
Shale, dark	ᵃ0	½	ᵃ0	⅞	ᵃ0	¼	ᵃ0	½	ᵃ0	⅝	ᵃ0	½
Coal, bright	0	2½	0	2½	0	2½	0	2¾	0	2½	0	2¾
Shale, dark	ᵃ0	½	ᵃ0	1⅝	ᵃ0	¾	ᵃ0	½	ᵃ0	⅝	ᵃ0	½
Coal, bright	2	2½	1	1⅝	1	1½	2	2	2	5½	0	9½
"Sulphur" band	..	..	..	..	0	½	..	..	..	..	0	¼
Coal, bright	..	..	..	..	0	11½	..	..	..	..	1	5¼
Floor, fire clay.												
Thickness of bed	5	6	4	5¾	5	2¼	5	6¾	5	6	5	8¾
Thickness of coal sampled	5	5	4	3¼	5	1¼	5	5¾	5	4¾	5	7¾

ᵃ Not included in sample.

Section A (sample 11157) was measured at the face of sixty-sixth butt entry, 2½ miles northeast of the mine mouth.

Section B (sample 11158) was measured on the rib of room 17, seventy-fifth entry, 2½ miles northeast of the mine mouth.

Section C (sample 11159) was measured at the face of room 23, seventy-ninth butt entry, 2¾ miles northeast of the mine mouth.

Section D (sample 11160) was measured on the chain pillar at room 46, seventy-fourth butt entry, 2½ miles northwest of the mine mouth.

Section E (sample 11161) was measured at the face of the north second main entry, 2½ miles north of the mine mouth.

Section F (sample 11162) was measured at the face of room 101 on sixty-eighth butt entry, 2¾ miles northwest of the mine mouth.

Three cars of slack coal from this mine were shipped to the Pittsburgh experiment station for testing purposes. The car samples are designated as follows: Pittsburgh No. 113, laboratory No. 11172; Pittsburgh No. 131, laboratory No. 11541; Pittsburgh No. 181, laboratory No. 12281.

Notes.—The coal is mined by the double-entry, room-and-pillar system. At the time of sampling electric chain-cutting machines were used. The coal was shot down with FFF black powder, dynamite being used for brushing the roof. The mine had a daily production of about 875 tons, which was derived both from advance workings and pillars. Only a little of the coal was shipped as run-of-mine. There were 3 loading tracks, with a capacity of about 45 cars, at the tipple. Pickers were employed both in the mine and on the railroad cars when coal was loaded. About 33 per cent of the coal produced was screenings. There was about 1,400 acres of unmined coal tributary to this opening.

WESTMORELAND COUNTY.

GREENSBURG. No. 4 MINE.

Sample.—Bituminous coal; Pittsburgh field; Greensburg district; analysis Nos. 12265 and 12853 (p. 99.)

Mine.—No. 4; 4 miles north of Greensburg.

Coal bed.—Pittsburgh. Carboniferous age, Monongahela formation. The coal from this mine was sampled by E. R. Linkenhoker on June 7 and November 13, 1911, respectively, while the coal was being unloaded at the Bureau of Mines testing plant, Pittsburgh, Pa.

Sample 12265 represented a carload of lump coal (Pittsburgh No. 178).

Sample 12853 represented a carload of lump coal (Pittsburgh No. 254).

LUCESCO. LUCESCO MINE.

Sample.—Bituminous coal; Pittsburgh field; analyses Nos. 12393, 12394, 12395, 12396, and 12397 (p. 99).

Mine.—Lucesco, a drift mine in the Allegheny River district, near Lucesco, a post office at Kiskiminetas Junction, on the Conemaugh and Allegheny divisions of the Pennsylvania Railroad.

Coal bed.—Known as the Upper Freeport or E. Carboniferous age, Allegheny formation. The thickness is rather variable, ranging from 2 to 4 feet. The roof near the opening is a sandy shale of variable thickness, averaging 1½ feet. At the points where the samples were taken the roof is sandstone. The floor is an underclay, 1½ to 3 feet thick, underlain by limestone. The floor is taken up in the inner workings to get the necessary height and the material is packed in gob walls along the entries. The cover increases rapidly from the outcrop and rises to 300 feet at places.

The bed was measured and sampled at four points in the mine by L. M. Jones, on June 29, 1911, as described below:

Sections of the coal bed in the Lucesco mine.

Section	A		B		C		D	
Laboratory No.	12393		12394		12395		12396	
Roof, hard sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal with fine "sulphur" streaks	0	3½	1	4	..	..	..	..
Mother coal	..	..	0	¼	..	..	..	..
Coal, hard	1	10½	1	1¾	2	10	2	8½
Mother coal or shale	0	½	0	¼	..	..	[a]0	½
Coal	0	1	0	9¾	..	..	0	2
Shale and coal	[a]0	1	[a]0	1½	[a]0	1	0	½
Coal	0	6	0	4½	0	6	0	3
Shale and "sulphur"	[a]0	½	..	..	[a]0	2	[a]0	1½
Coal	0	7	..	..	0	4	0	4
Floor, fire clay.								
Thickness of bed	3	6	3	10	3	11	3	8
Thickness of coal sampled	3	4½	3	8½	3	8	3	6

[a] Not included in sample.

Section A (sample 12393) was cut from the face of the ninth left entry, 4,000 feet south of the mine mouth.

Section B (sample 12394) was cut from the face of the fifth left entry, 165 feet beyond room 46, 3,600 feet from the mine mouth.

Section C (sample 12395) was cut from the face of room 48, first right entry, 3,200 feet southeast of the mine mouth.

Section D (sample 12396) was cut from the face of room 20, third right entry, 2,300 feet southeast of the mine mouth.

A composite sample was made by mixing face samples 12393, 12394, 12395, and 12396. The results of an ultimate analysis of this sample are shown under laboratory No. 12397.

Notes.—At the time of sampling, the coal at this mine was undercut by hand and blasted with FFF black powder. From the mine opening 40 feet above the tipple floor a gravity incline led to the tipple, which extended over the loading track and the main line of the Pennsylvania Railroad, and was equipped to load the engines on the latter track. The daily output averaged 300 tons a day.

MADISON. KEYSTONE MINE.

Sample.—Bituminous coal, Pittsburgh field. Analyses Nos. 16270 and 16271 (p. 99).

Mine.—Keystone, a shaft mine at Madison.

Coal bed.—Pittsburgh. Carboniferous age, Monongahela formation. Thickness about 6½ feet.

Sample 16270 was taken from a car of slack and sample 16271 from a car of run-of-mine coal, by E. R. Linkenhoker while the cars were being unloaded at the Pittsburgh testing station.

SOUTH DAKOTA.

HARDING COUNTY.

BUFFALO. HILTON MINE.

Sample.—Lignite. Northwestern South Dakota field; analysis No. 13221 (p. 100.)

Mine.—Hilton, a drift mine near center of sec. 6, T. 20 N., R. 5 E., 10 miles north of Buffalo.

Lignite bed.—No name. Cretaceous or Tertiary age; Lance formation.

The bed was measured and sampled by E. M. Parks on October 4, 1911, as described below:

Section of lignite bed in the Hilton mine.

Laboratory No.	13221	
	Ft.	*in.*
Roof, lignite.	*a* 1	11
Clay, gray	*a* 0	6
Lignite	2	10
Clay	*a* 0	1¼
Lignite, brown, woody	4	4
Clay (with water) reported	*a* 2	0
Lignite, reported	*a* 2	0
Floor, clay.		
Thickness of bed	13	8¼
Thickness of lignite sampled	7	2

a Not included in sample.

Note.—The mine was seldom worked. The sample was collected at the face of the drift entry, 120 feet from the outcrop at the mine mouth. The sample was fairly dry.

BUFFALO. OPEN PIT.

Sample.—Lignite. Northwestern South Dakota field; analysis No. 13220 (p. 100).

Mine.—Mendenhall; 18 miles east of Buffalo, in sec. 1, T. 17 N., R. 7 E.

Lignite bed.—Upper Cretaceous or Tertiary age; Lance formation.

The bed was measured and sampled by E. M. Parks on August 18, 1911, as described below:

Section of lignite bed in open prospect.

Laboratory No.	13220	
	Ft.	*in.*
Dirt, bituminous with coaly streaks *a*	0	5½
Lignite, tough	0	5
Lignite, bony	0	2
Lignite, tough	0	9
Bone	0	1
Lignite	0	7
Clay	0	1
Lignite, tough, with two thin sandy partings	4	⅝
Clay.*a*		
Thickness of bed	6	7
Thickness of lignite sampled	6	1⅝

a Not included in sample.

The sample was collected from the face of prospect after about two feet of lignite had been removed, and was fairly dry.

Note.—This prospect was seldom worked.

BUFFALO. OPEN PIT.

Sample.—Lignite. Northwestern South Dakota field; analysis No. 13222 (p. 100).

Mine.—Mendenhall; 18 miles east of Buffalo in sec. 1, T. 17 N., R. 7 E.

Lignite bed.—Upper Cretaceous or Tertiary age, Lance formation.

The bed was measured and sampled by E. M. Parks on August 17, 1911, as described below:

Section of lignite bed in open prospect.

Laboratory No.	13222	
	Ft.	*in.*
Clay, brown	[a]2	5
Lignite	[a]0	4
Clay, laminated	[a]1	1
Lignite, dirty	[a]0	6
Dirt, bituminous	[a]0	7
Lignite with bone interbedded	1	7
Shale with lignite interbedded	[a]0	3
Lignite, dirty	0	3½
Shale	0	½
Lignite	4	4
Thickness of bed	11	5
Thickness of lignite sampled	6	3

[a] Not included in sample.

The sample was collected from the face of the stripping after about 2 feet of lignite had been removed.

Note.—This prospect was seldom worked.

RALPH. NEWCOMB MINE.

Sample.—Lignite; Perkins and Harding counties field; analysis No. 15062 (p. 100).

Mine.—Newcomb, an open-pit mine, in the SW. ¼ sec. 10, T. 20 N., R. 9 E., 5 miles southeast of Ralph and about 28 miles south of Reeder, N. Dak., on the Chicago, Milwaukee & St. Paul Railroad.

Lignite bed.—Designated the Widow Clark bed. Tertiary (?) age, Ludlow lignitic member of the Lance formation. It is the most important bed of lignite in that part of Harding County. Its average thickness in T. 20 N., R. 9 E., is about 4 feet, but it varies from 2 feet 8 inches in sec. 9 to 10 feet 10 inches in sec. 10. It is usually of good quality with very small amounts of impurities. It is tough and slightly woody, when freshly mined, but slacks readily upon exposure to air. It is used entirely for domestic purposes.

The bed was measured and sampled October 16, 1912, by J. B. Reeside, jr., as shown below:

Section of lignite bed in the Newcomb mine.

Laboratory No.	15062	
Roof, shale.	*Ft.*	*in.*
Lignite, dirty	[a]0	6
Parting, shale	[a]0	1
Lignite, dirty	[a]0	1
Parting, shale	[a]0	1
Lignite, fine quality	10	1
Floor, not stated.		
Thickness of bed	10	10
Thickness of lignite sampled	10	1

[a] Not included in sample.

Notes.—The lignite is mined by stripping, and the sample was collected from the face of the bed, which was somewhat weathered.

PERKINS COUNTY.

LODGEPOLE. NELSON MINE.

Sample.—Lignite; northwestern South Dakota field; analysis 14354 (p. 100).

Mine.—Nelson; 3 miles southwest of Lodgepole, and about 18 miles south of the Chicago, Milwaukee & St. Paul Railway, in sec. 29, T. 21 N., R. 12 E.

Lignite bed.—Tertiary age, Fort Union formation. Thickness of part sampled, 7 feet. Neither the top nor the bottom of the bed was exposed.

The sample was collected in the entry 140 feet northwest of the entry mouth, by E. R. Lloyd and E. G. Woodruff, on July 3, 1912.

STROOL. JONES MINE.

Sample.—Lignite; northwestern South Dakota field; analysis No. 12453 (p. 100).

Mine.—Jones; a drift mine 6 miles northwest of Strool, in sec. 35, T. 19 N., R. 10 E.

Lignite bed.—The lignite bed is of Tertiary(?) age, Lance formation, and showed the following section:

Section of lignite bed in Jones mine, 6 miles northwest of Strool.

Laboratory No.	12453	
Roof, sandy shale.	*Ft.*	*in.*
Lignite	[a]1	2
Shale, sandy	[a]0	½
Lignite	4	2
Floor, clay.		
Thickness of bed	5	4½
Thickness of lignite sampled	4	2

[a] Excluded from sample.

Note.—The lignite is brown and woody. Section and sample were taken at end of main entry by D. E. Winchester, July 22, 1911.

STROOL. KNUDSEN MINE.

Sample.—Lignite; northwestern South Dakota field; analysis No. 12454 (p. 100).

Mine.—Knudsen; a drift mine 5 miles southwest of Strool, in sec. 2, T. 17 N., R. 10 E.

Lignite bed.—The lignite bed is of Tertiary(?) age, Lance formation, and has the following section:

Section of lignite bed in the Knudsen mine.

Laboratory No.	12454	
Roof, shale.	*Ft.*	*in.*
Lignite	2	7
Shale, brown	[a]0	4
Lignite, bony	[a]0	7
Floor, clay.		
Thickness of bed	3	6
Thickness of lignite sampled	2	7

[a] Not included in sample.

The lignite is brown and woody. Section and sample were taken at end of second entry by D. E. Winchester on July 21, 1911.

STROOL. PHILLIPS MINE.

Sample.—Lignite; northwestern South Dakota field; analysis No. 12488 (p. 100).

Mine.—Phillips; a drift mine 5 miles south of Strool, in sec. 17, T. 17 N., R. 11 E.

Lignite bed.—The lignite is of Tertiary(?) age, Lance formation, and had a thickness of 2 feet 5 inches at the point of sampling. Both roof and floor are shale. The lignite is dark brown or black, has well-developed cleavage, and otherwise closely resembles the subbituminous coal of eastern Wyoming.

The bed was measured and sampled by D. E. Winchester on July 26, 1911. The sample was taken 150 feet from the mine mouth.

TENNESSEE.

ANDERSON COUNTY.

BRICEVILLE. CROSS MOUNTAIN No. 1 MINE.

Sample.—Bituminous coal; Cumberland Plateau field; analyses Nos. 13237, 13238, 13264, 13265, and 13290 (p. 101).

Mine.—Cross Mountain No. 1, a drift mine one-half mile west of Briceville on a branch of the Southern Railroad.

Coal bed.—Known as the Coal Creek, but not correlated with other beds in eastern Tennessee. The coal is of Carboniferous (Pottsville) age, and the average thickness is 3 feet 10 inches, the range being from 3 feet to 4 feet 6 inches. It is rather hard coal and has a well-defined cleat. The roof is a hard sandy gray shale, called "slate" by the miners. Over the first 3,000 feet of entry this roof is poor, but after that it is very strong. The floor is an underclay which is usually rather hard but softens and swells when wet and when pillars are being pulled.

The coal bed was measured and sampled at three points in the mine during January, 1912, by J. J. Rutledge and E. B. Sutton, as described below:

Sections of coal bed in Cross Mountain No. 1 mine.

Section	A	B	C
Laboratory No.	13237	13238	13290
Roof, gray shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	0 7	.. ..	.. ..
Cannel coal	[a] 0 1	.. ..	.. ..
Coal	3 0	3 10	4 0
Floor, underclay.			
Thickness of bed	3 8	3 10	4 0
Thickness of coal sampled	3 7	3 10	4 0

[a] Not included in sample.

Section A (sample 13237) was cut from the right rib of left entry 26, outby last break-through to the right.

Section B (sample 13238) was cut from a chain pillar on right entry 27, 18 feet outby the twelfth and last break-through.

Section C (sample 13290) was cut from the right rib of right entry 27, about 30 feet from main entry.

Sample 13264 was taken in room 3, off left entry 26.

Sample 13265 was taken in room 2, off left entry 26.

Notes.—In 1912 the Cross Mountain No. 1 mine was opened by a drift and worked by the room-and-pillar method. The coal was undercut by electric chain machines in the lower part of the bed. Very little explosive was used, as the coal frequently dropped while the head of the undercutting machine was being removed. When powder was necessary FFF black blasting powder was used exclusively, 40 per cent strength dynamite being used for brushing the roof. The coal was marketable either as steam or domestic coal, the latter quality being said to give the mine a decided advantage in the markets. The screens used were varied to suit the market.

BLEDSOE COUNTY.[a]

HERBERT. PROSPECT.

Sample.—Bituminous coal, Cumberland Plateau field; analysis No. 10915 (p. 101).

Mine.—Prospect on State land near Herbert post office, about 4 miles north of Saratoga Springs and about 14 miles northwest of Pikeville.

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[a] For a description of the coals here listed see Bull. 9, Tennessee Geol. Survey.

Coal bed.—The coal is provisionally correlated with the Sewanee coal. Carboniferous system, Pennsylvanian series, Pottsville group, Walden sandstone formation. Roof, shale.

The bed was measured and sampled by H. G. Hart, under the direction of W. G. Phalen, on September 10, 1910. The sample represented a 6-foot 4-inch cut of coal. In another place the coal was 6 feet 4 inches thick, with a shale roof and clay floor. The base of the coal was not reached, but the thickness reported is 7 feet 4 inches.

The sample was taken near the mouth of the drift.

The relations of this coal bed are indicated by Phalen[a]. These descriptions include a consideration of the geologic position of the coal, together with notes on its physical and chemical character.

HERBERT. PROSPECT.

Sample.—Bituminous coal, Cumberland Plateau field; analysis No. 10916 (p. 101).

Mine.—Prospect on State land near Herbert post office, about 4 miles directly north of Saratoga Springs and 14 miles northwest of Pikeville, the county seat of Bledsoe County.

Coal bed.—The coal is provisionally correlated with the Sewanee coal, Carboniferous system, Pennsylvanian series, Pottsville group, Walden sandstone formation. Roof, shale; floor, clay.

The bed was measured and sampled on September 16, 1910, by H. G. Hart, under the direction of W. C. Phalen. The sample represented a 3-foot cut of coal. It was collected at the face of a new opening which had been driven about 10 feet.

Notes.—The prospect was not being worked at the time of collection.

LITTON. A. L. HALE MINE.

Sample.—Bituminous coal; Cumberland Plateau field; analysis No. 12584 (p. 101).

Mine.—A. L. Hale, a local mine 2 miles northwest of Litton, 15 miles northeast of Nashville, Chattanooga & St. Louis Railroad at Pikeville.

Coal bed.—Angel, Carboniferous age, Pottsville formation. Dip 20° southwest. Roof, sandstone; floor, clay.

The bed was measured and sampled by W. A. Nelson on August 31, 1911. The sample represented an 8-foot cut of coal.

LITTON. PROSPECT.

Sample.—Bituminous coal; Cumberland Plateau field; analysis No. 10803 (p. 101).

Mine.—Prospect at the head of Sequatchie Valley, 4 miles north of Litton and 12 miles northeast of Pikeville, the terminus of the Sequatchie Valley branch of the Nashville, Chattanooga & St. Louis Railroad.

Coal bed.—The position of the coal is 36 feet below the base of the Sewanee sandstone, Carboniferous system, Pennsylvanian series, Pottsville group, Lookout sandstone formation.

The bed was measured and sampled August 20, 1910, under the direction of W. C. Phalen. Eight feet of coal was measured.

The sample was collected 60 feet from the drift mouth in the main and only entry.

PIKEVILLE. J. W. MCFARLAND PROSPECT.

Sample.—Bituminous coal; Cumberland Plateau field; analysis No. 10731 (p. 101).

Mine.—J. W. McFarland prospect, 4 miles west of Pikeville, on the Cumberland Plateau.

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[a] Phalen, W. C., Preliminary Report of the Coal Resources of the Pikeville Special Quadrangle of Eastern Tennessee: Tenn. Geol. Survey Bull. 9, 1911, pp. 65-67.

Coal bed.—Morgan Springs. Carboniferous system, Pennsylvanian series, Pottsville group, Walden sandstone formation. Roof, soft blue clay, 18 inches thick.

The bed was measured and sampled August 3, 1910, under the direction of W. C. Phalen, in a country bank approximately 50 feet from the drift mouth. The sample represented a 21½-inch cut of coal.

Note.—This is the highest important coal in the Cumberland Plateau in this region.

PIKEVILLE. VAUGHN BROS. PROSPECT.

Sample.—Bituminous coal; Cumberland Plateau field; analysis No. 11049 (p. 101).

Mine.—Outcrop, about 10 miles west of Pikeville, on Cane Creek, above the junction of Cane and Meadow Creeks.

Coal bed.—The coal bed is provisionally correlated with the Sewanee coal. Carboniferous system, Pennsylvanian series, Pottsville group, Walden sandstone formation. Roof, shale; floor, clay.

The bed was measured and sampled by W. C. Phalen on October 10, 1910. The sample represented a 3-foot 9-inch cut of coal.

PIKEVILLE. VAUGHN BROS. PROSPECT.

Sample.—Bituminous coal; Cumberland Plateau field; analysis No. 11050 (p. 101).

Mine.—From an outcrop on Cane Creek, above the junction of Meadow and Cane Creeks. The outcrop is located about 10 miles west of Pikeville, the county seat of Bledsoe County.

Coal bed.—The coal bed is provisionally correlated with the Sewanee coal. Carboniferous system, Pennsylvanian series, Pottsville group, Walden sandstone formation.

The bed was measured and sampled by W. C. Phalen on October 10, 1910. The sample represented a 3-foot 2-inch cut of coal.

CUMBERLAND COUNTY.[a]

LITTON. J. H. HALE & SON PROSPECT.

Sample.—Bituminous coal; Cumberland Plateau field; analysis No. 10800 (p. 101).

Mine.—J. H. Hale & Son prospect. Head of Sequatchie Valley, 4 miles from Litton post office and 12 miles northeast of Pikeville, the terminus of the Nashville, Chattanooga & St. Louis Railroad.

Coal bed.—Sewanee (?) coal. Occurring in the shale and sandstone interval above the Sewanee sandstone. Carboniferous age, Pennsylvanian series, Pottsville group, Walden sandstone formation. Roof, shale.

The bed was measured and sampled by H. G. Hart, under the direction of W. C. Phalen, on August 20, 1910.

The sample was collected 30 feet from the mouth of the prospect and represented a 5-foot 6-inch cut of coal, which was underlain by 1 foot of bone.

LITTON. J. H. HALE & SON PROSPECT.

Sample.—Bituminous coal; Cumberland Plateau field; analysis No. 10802 (p. 101).

Mine.—J. H. Hale & Son prospect, head of Sequatchie Valley, 4 miles north of Litton and 12 miles northeast of Pikeville, the terminus of the Sequatchie Valley branch of the Nashville, Chattanooga & St. Louis Railroad.

Coal bed.—Morgan Springs. The highest important coal in this part of the Cumberland Plateau. Carboniferous age, Pennsylvanian series, Pottsville group, Walden sandstone formation. Roof, shale; floor, clay, underlain with sandstone.

The bed was measured and sampled August 20, 1910, under the direction of W. C. Phalen. The sample represented a 3-foot 6-inch cut of coal.

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[a] For fuller descriptions of the coals here listed, see Bull. 9, Tennessee Geol. Survey, 1911.

RHEA COUNTY.[a]

DAYTON. NORTH POLE MINE.

Sample.—Bituminous coal; Cumberland Plateau field; analyses Nos. 10689, 10690, 10691, 10692, and 10693 (p. 102).

Mine.—North Pole, at Dayton.

Coal bed.—Richland. Carboniferous system, Pennsylvanian series, Pottsville group, Walden sandstone formation. The bed lies above the Sewanee sandstone (the lower conglomerate member). In some places the coal bed rests directly on it, though in other places a few feet of shale or clay intervene. Locally a few feet of bony coal or "rash" occur between the main coal and the conglomerate. Roof, shale; floor, bone or clay.

The bed was measured and sampled at five points by W. C. Phalen on July 30, 1910.

Sample 10689 was taken 3,000 feet from the mine mouth and represented a 25½-inch cut of coal. The lower 7 or 8 inches of coal are very soft; the rest is hard.

Sample 10690 was collected at the face of a room in left heading 5, about 2,600 feet from mine mouth, and represented a 23-inch cut of coal. Roof, shale; floor, clay.

Sample 10691 represents the bony coal or "rash" underneath the good coal in the room where sample 10640 was taken.

Sample 10692 was collected at the face of left heading 8, about 3,000 feet from the mouth of the mine and represented a 22-inch cut of coal, with bony or clay streaks. Roof, massive shale; floor, clay.

Sample 10693 was collected from the face of right heading 7, between 2,700 and 2,800 feet from the mine mouth, and represented a 24-inch cut of coal. Roof, shale; floor, clay.

Notes.—The so-called rash is a bony coal, and the sample was collected to prove this point and also to determine whether a use might be discovered for it. It is discarded in mining. It occurs locally between the coal and the top of the underlying conglomerate.

The coal is coked and the coke is used by the Dayton Coal & Iron Co. in the company's furnace at Dayton, on the Cincinnati, New Orleans & Texas Pacific Railroad.

DAYTON. NEW PROSPECT MINE.

Sample.—Bituminous coal; Cumberland Plateau field; analyses Nos. 10696, 10697, 10698, and 10699 (p. 102).

Mine.—New Prospect, at Dayton.

Coal bed.—Nelson. Carboniferous system, Pennsylvanian series, Pottsville group, Lookout sandstone formation.

The bed was measured and sampled by W. C. Phalan on July 29, 1910.

Sample 10696 was taken at the face of a room in left heading 3, about 1,200 feet from the mine mouth. The bed measured 3 feet 10½ inches. There was a 2½ layer of bone 8 inches from the top of the bed. The sample represents the lower 3 feet. Roof, sandstone; floor, bone, sometimes compact, underlain with clay and sandstone.

Sample 10697 was taken at the face of a room off left heading 2 and represented 2 feet 3 inches of coal. The lower 3 inches was laminated and soft coal. Roof, hard, sandy clay; floor, bone.

Sample 10698 was taken at the end of the right heading 2 and represented a 3-foot 2-inch cut of coal. Roof, dense, compact clay with stringers of coal varying in thickness from a knife's edge to 4 inches; floor, clay.

Sample 10699 was taken in the main heading, 1,500 to 1,600 feet from the mine mouth, and represented a 2-foot 5-inch to 2-foot 7-inch cut of coal, overlain by shale, coal, and clay. Floor, clay.

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[a] For a fuller account of the coals here listed see Bull. 9, Tennessee Geol. Survey, 1911.

Notes.—This is the principal coal bed of the Lookout formation, and one of the most important in the southeastern part of the Pikeville Special quadrangle. The coal is coked and the coke is used by the Dayton Coal & Iron Co. in its furnace at Dayton. The daily output of the mine at the time of sampling was 150 tons.

UTAH.

CARBON COUNTY.

HIAWATHA. HIAWATHA NO. 1 AND NO. 2 MINES.

Sample.—Bituminous coal; Book Cliffs field; analyses Nos. 10904, 10905, 10907, and 10908 (p. 103).

Mine.—Hiawatha No. 1 and No. 2; drift mines at Hiawatha; railroad connections via Utah Southern Railway to Rio Grande & Western Railway at Price, 23 miles distant.

Coal bed.—Hiawatha. Cretaceous age, Mesaverde formation. Thickness, 15 feet. Dip, 1° southeast in No. 1 mine and 1° southwest in No. 2 mine. Roof, "draw slate" 18 inches thick, with sandstone cap above; floor, smooth, hard sandstone. The cover at points where samples were taken is 150 to 300 feet.

The bed was measured and sampled at four points in the mine by H. M. Wolflin, on September 9, 1910, as described below:

Sections of coal bed in Hiawatha No. 1 and No. 2 mines.

Section	A		B		C		D	
Laboratory No.	10904		10905		10907		10908	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Top coal	..	..	..	..	[a]2	6	..	..
Coal, bony	..	..	..	..	[a]0	5½	[a]0	7½
Coal, clean	3	11	5	2	4	3	4	3
Coal, bony	0	1	0	1	0	½	[a]0	4
Coal, clean	..	..	..	..	..	..	1	1
Coal, bony	..	..	..	..	..	..	0	½
Coal, clean	7	2	6	2	6	0	3	4½
Coal, bony	..	..	..	..	..	..	0	½
Coal, clean	..	..	..	..	..	..	1	10
Floor, sandstone.								
Thickness of bed	11	2	11	5	13	3	11	7
Thickness of coal sampled	11	2	11	5	10	3½	10	7½

[a] Not included in sample.

Section A (sample 10904) was measured at the face of room 7 off left entry 1, at a depth of 300 feet below the surface at No. 1 mine.

Section B (sample 10905) was measured at the face of left aircourse 2 at No. 1 mine.

Section C (sample 10907) was measured at the face of room 2 off dip entry 2 at No. 2 mine.

Section D (sample 10908) was measured at the face of room 1 off rise entry 2 at a depth of 150 feet below the surface at No. 2 mine.

Notes.—At the time of sampling the screening plant was equipped for producing ½, ⅝, 1¼, and 3-inch coal. No hand picking. There were 4 loading tracks with a capacity for 60 cars.

The mine was worked by the room-and-pillar system; in the rooms the coal was undercut and shot down with permissible explosives; some black powder was also used; in the entries the coal was shot from the solid; daily output 1,000 tons; coal acreage 3,260. A gravity incline connected the mines with the tipple 2 miles down the canyon. Mines began shipping coal in February, 1910.

PRICE. JESSE KNIGHT PROSPECT.

Sample.—Bituminous coal; Book Cliffs field; analysis No. 14678 (p. 103).

Mine.—Jesse Knight, a prospect in Right Fork of Deadman Canyon in SE. ¼ sec. 7, T. 13 S., R. 11 E., 9 miles northeast of Price, Utah, which is on the Denver & Rio Grande Railroad.

Coal bed.—No name. Cretaceous age, Mesaverde formation. This is the same bed from which sample No. 14801 was taken, and in this vicinity dips about 5° N.-NE.

The bed was measured and sampled on August 28, 1912, by F. R. Clark, as described below:

Section of coal bed in Jesse Knight prospect.

Laboratory No.	14678	
	Ft.	*in.*
Coal	1	2
Shale, sandy	[a]0	3
Coal	5	3
Thickness of bed	6	8
Thickness of coal sampled	6	5

[a] Not included in sample.

Notes.—The prospect was worked only during the winter season for local use. The coal is jet black, massive, with joint planes parallel to the strike of the beds. The sample was collected at a point 90 feet N. 70° W. from opening. The coal was slightly weathered.

SUNNYSIDE. NO. 1 MINE.

Sample.—Bituminous coking coal; Book Cliffs field; analysis No. 12632 (p. 103).

Mine.—No. 1; in NE. ¼ SE. ¼ sec. 32, T. 14 S., R. 14 E., at Sunnyside, which is reached by a branch of the Denver & Rio Grande Railroad.

Coal bed.—Lower. Cretaceous age, Mesaverde formation. Seven to 9 feet thick in this locality. Roof, "soapstone," said to be very weak and part of the coal is left for roof.

The bed was measured and sampled by F. R. Clark, September 14, 1911, at the face of the Fowler slope as described below:

Section of coal bed in No. 1 mine.

Laboratory No.	12632	
	Ft.	*in.*
Coal	1	6
Shale, soapstone	1	6
Coal	7	0
Thickness of bed	10	0
Thickness of coal sampled (at base of bed)	5	5

Notes.—Both beds (upper and lower veins) were being worked from this mine. The daily output at the time of sampling was about 1,600 tons. The coal was screened and loaded or sent to crushers to be crushed for the coke ovens with the coal from No. 3 mine.

SUNNYSIDE. NO. 3 MINE.

Sample.—Bituminous coking coal; Book Cliffs field; analyses Nos. 12546, 12630, and 12631 (p. 103).

Mine.—No. 3; in SW. ¼ SW. ¼ sec. 4, T. 15 S., R. 14 E., 1 mile southeast of Sunnyside, on branch of Denver & Rio Grande Railroad; connected with the railroad by the main haulage way through No. 2 mine.

Coal bed.—Cretaceous age, Mesaverde formation. Two beds are worked, the upper and lower, which, in the vicinity of the points sampled, are separated by only 6 feet of highly sandy shale forming a good roof in the one case and a good floor in the other. The lower bed is separated from a massive sandstone beneath by only a few inches of brown shale. The upper bed has a sandy shale roof containing in places a few fossil shells.

Three samples were collected from this mine (two from the upper and one from the lower bed) by F. R. Clark in 1911.

Sample 12546 was cut from the side of the outlet airway in NW. ¼ SW. ¼ sec. 9, T. 15 S., R. 14 E. The sample includes coal from the whole upper bed (6 feet 2 inches thick). The point from which sample was taken is 50 feet north-northwest from the surface opening and the coal was weathered more or less.

Sample 12630, also from the upper bed, was cut from the face of the second crosscut off right entry 1 off No. 1 rise, which is about 1¾ miles southeast from the mouth of the mine, and was fresh and unweathered. The coal is clean, black, and massive, free from partings, and was 4 feet 8 inches thick at the point sampled.

Sample 12631 from the lower bed was taken from "the dips," or entries down the dip from the main haulage way, in the mine. The bed was about 8 feet thick at this point (1¾ miles southeast from mouth of the mine), but only 6 feet 10 inches of the upper part of the bed was included in the sample.

Notes.—At the time of sampling the coal was undercut with hand picks and broken down with powder, and the output was about 900 tons daily. The greater part of the output was coked. No attempt was made to keep the coal from the two beds separate as they were so nearly alike in their tendencies to coke, as well as other physical and chemical characteristics. The coal from each bed was screened and loaded into cars or sent to the crusher to be crushed for the coke ovens.

SUNNYSIDE. PROSPECT.

Sample.—Bituminous coking coal; Books Cliffs field; analysis No. 12545 (p. 103).

Mine.—Prospect in NE. ¼ SW. ¼ sec. 5, T. 15 S., R. 14 E., three-fourths of a mile south of Sunnyside, Sunnyside district, on Sunnyside branch of the Denver & Rio Grande Railroad.

Coal bed.—Lower. Cretaceous age, Mesaverde formation. Dip, 7° N., 80° E. The bed was measured and sampled on July 30, 1911, by F. R. Clark, as shown below:

Section of coal bed in prospect three-fourths of a mile south of Sunnyside.

Laboratory No.	12545	
	Ft.	*in.*
Coal	[a] 2	8
Shale, sandy	[a] 2	2
Coal	[a] 3	0
Coal	6	0
Thickness of bed	13	10
Thickness of coal sampled	6	0

[a] Not included in sample.

The coal was more or less weathered.

The sample was taken at a point 85 feet N. 40° E. from mouth of slope.

SUNNYSIDE. PROSPECT.

Sample.—Bituminous coal; Book Cliffs field; analysis No. 12792 (p. 103).

Mine.—Prospect in the right fork of Rock Canyon in NE. ¼ SE. ¼ sec. 32, T. 13 S., R. 13 E., or 8½ miles northwest of Sunnyside.

Coal bed.—No name. Cretaceous age; Mesaverde formation. This coal appears first only about 3 miles southeast of the prospect. It is a lower coal than either of those mined at Sunnyside.

The bed was measured and sampled on October 25, 1911, by F. R. Clark. The coal was at this point 5 feet 7 inches thick, and a cut across the bed was included in the sample.

Notes.—The coal is black, massive, bright, and comparatively hard. The sample was somewhat weathered, as the cut was made only 200 feet N. 72° W. from the mine opening, and the face had been exposed to the atmosphere for a long time.

SUNNYSIDE. PROSPECT.

Sample.—Bituminous coal; Book Cliffs field; analysis No. 14801 (p. 103).

Mine.—Prospect in the SE. ¼ NW. ¼ sec. 23, T. 13 S., R. 12 E., 14 miles northeast of Wellington, Utah, and 12 miles northwest of Sunnyside, Utah.

Coal bed.—Not named. Cretaceous age, Mesaverde formation. Dip about 5° N.-NW.

The bed was measured and sampled on September 29, 1912, by F. R. Clark, as shown below:

Section of coal bed in prospect 12 miles northwest of Sunnyside.

Laboratory No.	14801	
	Ft.	*in.*
Bone	[a] 0	2
Shale, sandy	[a] 0	1½
Coal	2	5
Bone	0	1½
Coal	5	8
Thickness of bed	8	6
Thickness of coal sampled	8	2½

[a] Not included in sample.

The sample was taken at a point 100 feet S. 40° E. from opening.

Notes.—The coal is black, hard, and massive, and aside from the bone parting is free from impurities. This is the lowest bed in the series at this point that has any considerable thickness.

The coal from the prospect was mined for local use during the winter season, but there was no railroad connection.

EMERY COUNTY.

EMERY. BROWNING MINE.

Sample.—Bituminous coal; Emery field; analysis No. 12627 (p. 104).

Mine.—Browning; in the NE. ¼ SW. ¼ sec. 33, T. 22 S., R. 6 E., about 4 miles south of Emery.

Coal bed.—I. Cretaceous age, Ferron sandstone member of the Mancos formation. The bed at this place is 22 feet thick, which is the greatest thickness known in any coal bed in this field. Roof, brown shale; floor, brown shale.

The bed was measured and sampled by C. T. Lupton on September 19, 1911, as described below:

Section of coal bed in Browning mine.

Laboratory No.	12627	
Roof, brown shale.	*Ft.*	*in.*
Coal	[a] 1	0
Shale, brown	[a] 1	0
Coal with one or two 1-inch brown shale partings	[b] 20	0
Floor, brown shale.		
Thickness of bed	22	0
Thickness of coal sampled	12	0

[a] Not included in sample.

[b] Upper 8 feet not included in sample.

The sample was taken at the back end of the main entry 165 feet N. 85° E. from the pit mouth.

EMERY. CASPER MINE.

Sample.—Bituminous coal; Emery field; analysis 12652 (p. 104).

Mine.—Casper; in the SE. ¼ NW. ¼ sec. 26, T. 22 S., R. 6 E., about 4 miles southeast of Emery, in the Emery district.

Coal bed.—C. Cretaceous age, Ferron sandstone member of the Mancos formation. The bed averages about 9 feet thick in the immediate locality. Roof, sandstone; floor, soft drab clay.

The bed was measured by C. T. Lupton on September 23, 1911, as described below:

Section of coal bed in Casper mine.

Laboratory No.	12652	
Roof, sandstone.	*Ft.*	*in.*
Coal	[a] 1	3
Sandstone and shale	[a] 1	3½
Coal, very bony	[a] 0	2
Coal, good	1	6
Coal, bony	1	1½
Coal	4	6
Floor, soft drab shale.		
Thickness of bed	9	10
Thickness of coal sampled	7	3½

[a] Not included in sample.

The sample was taken at the back end of the main entry about 200 feet northwest from the pit mouth.

EMERY. SURFACE PROSPECT.

Sample.—Bituminous coal; Emery field; analysis No. 14903 (p. 104).

Surface prospect.—Cave under rock ledge, in the SW. ¼ SW. ¼ sec. 32, T. 23 S., R. 6 E., about 12 miles southwest of Emery, in the Emery district.

Coal bed.—I. Cretaceous age, Ferron sandstone member of the Mancos formation. The bed is about 12 feet thick at this locality. Roof, drab and brown clay; floor, brown sandstone.

The bed was measured and sampled by C. T. Lupton on September 27, 1912, as described below:

Section of coal bed at surface prospect about 12 miles southwest of Emery.

Laboratory No.	14903	
Roof, drab and brown clay.	*Ft.*	*in.*
Coal	5	5
Clay, brown	[a] 1	10
Coal	5	0
Floor, brown sandstone.		
Thickness of bed	12	3
Thickness of coal sampled	10	5

[a] Not included in sample.

The sample was taken from a surface prospect in a cave under a rock ledge at the place indicated above.

EMERY. WILLIAMS MINE.

Sample.—Bituminous coal; Emery field; analysis No. 12613 (p. 104).

Mine.—Williams; SE. ¼ SW. ¼ sec. 12, T. 22 S., R. 6 E.; about 2½ miles east of Emery.

Coal bed.—I. Cretaceous age, Ferron sandstone member of the Mancos formation. The bed averages about 6 feet thick in this part of the field. Roof, sandstone with a little coal in the lower part; floor, sandy shale. This coal bed is about 250 feet stratigraphically above the base of the Ferron sandstone member.

The bed was sampled and measured by C. T. Lupton on September 14, 1911, as described below:

Section of coal bed in Williams mine.

Laboratory No.	12613	
Roof, sandstone.	*Ft.*	*in.*
Coal	0	4½
Bone and coal	0	2
Coal	5	3
Floor, sandy shale.		
Thickness of bed	5	9½
Thickness of coal sampled	5	9½

The sample was taken at the back end of the main entry 200 feet S. 60° E. from the entry mouth.

SEVIER COUNTY.

EMERY. SURFACE PROSPECT.

Sample.—Bituminous coal; Emery field; analysis 15061 (p. 104).

Location.—Surface prospect; in the SW. ¼ NW. ¼ sec. 25, T. 24 S., R. 5 E., about 15 miles south of Emery.

Coal bed.—A. Cretaceous age, Ferron sandstone member of the Mancos formation. The bed averages 11 feet thick at this locality. Roof, sandstone; floor, brown shale.

The bed was measured by C. T. Lupton on November 2, 1912, as described below:

Section of coal at surface prospect, about 15 miles south of Emery.

Laboratory No.	15061	
Roof, sandstone.	*Ft.*	*in.*
Coal, bony	0	10
Coal, bright	1	7
Coal, dull	5	10
Bone	[a]0	3
Coal, bright	1	8
Coal, dull	1	1
Floor, brown shale.		
Thickness of bed	11	3
Thickness of coal sampled	11	0

[a] Not included in sample.

Sample 15061 was taken at a surface prospect at the locality stated.

FREMONT. HOGAN PROSPECT.

Sample.—Bituminous coal; Emery field; analysis No. 15090 (p. 104).

Prospect.—Hogan; about 15 miles northeast from Fremont, in the SW. ¼ NW. ¼ sec. 3, T. 26 S., R. 4 E.

Coal bed.—A. Cretaceous age, Ferron sandstone member of the Mancos formation. The bed at this place averages 5 feet thick. Roof, shale; floor, shale and bone.

The bed was measured and sampled by C. T. Lupton on October 21, 1912, as described below:

Section of coal bed in Hogan prospect.

Laboratory No.	15090	
Roof, shale.	*Ft.*	*in.*
Coal	[a]0	8
Bone	[a]0	5
Bone and bony coal	[a]1	0
Coal	3	0
Floor, shale and bone.		
Thickness of bed	5	1
Thickness of coal sampled	3	0

[a] Not included in sample.

The sample was taken at the back end of the drift, 20 feet N. 40° W. from the mouth.

Salina. Kearns & Duggins Mine.

Sample.—Bituminous coal; Wasatch Plateau field; analysis No. 15045 (p. 104).

Mine.—Kearns and Duggins; about 15 miles east of Salina, in the SW. ¼ SW. ¼ sec. 7, T. 22 S., R. 3 E., Salt Lake Meridian.

Coal bed.—"Duggins." Cretaceous age, Mesaverde formation. The bed averages 3 feet 9 inches thick at this locality. Roof, shale; floor, hard gray sandstone.

The bed was measured and sampled by C. T. Lupton on October 26, 1912, as described below:

Section of coal bed in Kearns & Duggins mine.

Laboratory No	15045	
Roof, shale.		
	Ft.	*in.*
Coal, containing resin globules	2	8
Bone	[a]0	2
Coal	0	11
Floor, gray sandstone.		
Thickness of bed	3	9
Thickness of coal sampled	3	7

[a] Not included in sample.

The sample was taken in a room off the entry, 178 feet north of the entry mouth.

SUMMIT COUNTY.

Coalville. Rees-Grass Creek Mine.

Sample.—Subbituminous coal; Coalville field; analysis No. 13216 (p. 329).

Mine.—Rees-Grass Creek, a slope mine in SW. ¼ sec. 24, T. 3 N., R. 5 E., 5 miles northeast of Coalville.

Coal bed.—Wasatch. Cretaceous age, Frontier formation. The roof of the bed in some parts of the mine is sandstone and in others shale. The floor is shale.

The bed was measured and sampled by C. H. Wegemann on October 11, 1911.

The sample was taken 400 feet below the surface and 500 feet east of the main slope, and represented an 8-foot 6½-inch cut of coal.

Notes.—A slope is run on the dip of the bed about 30° northwest and the coal cars were brought to the surface by a hoisting engine. Capacity of mine, 150 to 200 tons.

Coalville. Superior Mine.

Sample.—Subbituminous coal; Coalville field; analysis No. 13217 (p. 329).

Mine.—Superior, a slope mine on the south side of sec. 16, T. 2 N., R. 5 E., 1 mile southeast of Coalville.

Coal bed.—Wasatch. Cretaceous age, Frontier formation. Dip of bed 10° to 15° to the south.

The bed was measured and sampled by C. H. Wegemann on October 11, 1911, and represented a 7-foot 6-inch cut of coal.

The sample was taken 200 feet below the surface, 1,000 feet northwest of the foot of slope.

Note.—Capacity of mine, 150 tons.

Coalville. Wasatch Mine.

Sample.—Subbituminous coal; Weber River field; analysis No. 13218 (p. 329).

Mine.—Wasatch, a slope mine in the SW. ¼ sec. 3, T. 2 N., R. 5 E., 3 miles northeast of Coalville.

Coal bed.—Wasatch. Cretaceous age, Frontier formation. Dip about 33° northwest, decreasing to 12° farther down the slope.

The bed was measured and sampled by C. H. Wegemann on October 11, 1911.

The sample was taken on the 800-foot level, 3,000 feet northeast of main slope, and represented a 10 foot 6-inch cut of coal.

Note.—The capacity of this mine at the time of sampling was about 500 tons.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 812–813.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 285.

UINTA COUNTY.

VERNAL. BLUE BELL MINE.

Sample.—Bituminous coal; Vernal field; analysis No. 10712 (p. 105).

Mine.—Blue Bell, a mine in sec. 8, T. 3 S., R. 22 E., about 7 miles north of Vernal and 100 miles east of Heber, which is the nearest station on the Rio Grande & Western Railway.

Coal bed.—No name. Cretaceous age, Mesaverde formation. Thickness, 7 feet 8 inches. Floor, brown carbonaceous shale.

The bed was measured and sampled by C. T. Lupton on August 3, 1910, as described below:

Section of coal bed in Blue Bell mine.

Laboratory No.	10712	
Roof, not stated.	*Ft.*	*in.*
Coal, bright and solid	1	4
Coal, bony	[a] 0	2¼
Coal, bright and solid	0	4¾
Clay, brown, very hard	[a] 0	1
Clay, very soft	[a] 0	8
Coal, bright	0	9¾
Coal, dull	0	2
Coal, bright and solid	1	9
Shale, brown	[a] 0	3½
Coal, bright	2	0
Floor, shale.		
Thickness of bed	7	8¼
Thickness of coal sampled	6	5½

[a] Not included in sample.

The above sample was taken at the face of the entry about 225 feet from the mouth.

VERNAL. GREEN MINE.

Sample.—Bituminous coal; Vernal field; analysis No. 10713 (p. 105).

Mine.—Green, a mine in sec. 13, T. 4 S., R. 22 E., 8 miles east of Vernal and about 110 miles east of Heber, which is the nearest station on the Rio Grande & Western Railway.

Coal bed.—No name. Cretaceous age, Mesaverde formation. Thickness, 4 feet 8¾ inches. Roof, gray sandstone, about 2 feet thick; floor, gray sandstone about 8 feet thick.

The bed was measured and sampled by C. T. Lupton on August 2, 1910, as described below:

Section of coal bed in Green mine.

Laboratory No.	10713	
Roof, sandstone.	*Ft.*	*in.*
Coal, fair	0	5
Coal, soft, bony	[a] 0	5
Coal, hard, bony	[a] 0	4
Bone, streaked with coal	[a] 0	5½
Coal, good quality	0	9½
Coal, bony	0	½
Coal, good	0	7½
Shale and bone with thin coal streaks	[a] 0	4½
Coal, fair quality	0	7½
Coal, bony	[a] 0	3
Coal, sandy (considered as waste)	[a] 0	7
Floor, gray sandstone.		
Thickness of bed	4	11
Thickness of coal sampled	2	6

[a] Not included in sample.

The sample was taken at the face of the entry, 75 feet from the mouth. The coal at this point was slightly weathered.

VERNAL. REYNOLDS MINE.

Sample.—Bituminous coal; Vernal field; analysis No. 10812 (p. 105).

Mine.—Reynolds, a mine in sec. 20, T. 3 S., R. 19 E., about 20 miles west of Vernal and 80 miles east of Heber, which is the nearest station on the Rio Grande & Western Railway.

Coal bed.—Mancos. Cretaceous age, Mesaverde formation. Thickness, 5 feet 4 inches; dip, 20° south. Roof, shale, about 4 feet thick; floor, shale. Dip, 20° to the south.

The bed was measured and sampled by C. T. Lupton on August 18, 1910, as described below:

Section of coal bed in Reynolds mine.

Laboratory No	10812	
Roof, shale.	*Ft.*	*in.*
Clay shale with streaks of coal near bottom	[a] 0	7½
Coal, bright	[a] 0	2
Clay shale, brown and soft	[a] 0	2¼
Coal, solid	2	1¼
Clay shale, brown with coal streaks	[a] 0	2
Coal, solid	0	6
Coal, bony	[a] 0	4
Coal, solid and bright	1	2¾
Floor, brown clay.		
Thickness of bed	5	4
Thickness of coal sampled	3	10

[a] Not included in sample.

The sample was taken at the face of the main entry about 210 feet from the surface.

WASATCH COUNTY.

HANNA. PROSPECT.

Sample.—Subbituminous coal; Blacktail Mountain field; analysis No. 10998 (p. 105).

Mine.—This prospect, in sec. 26, T. 1 S., R. 9 W., on Red Creek, is about 15 miles southwest of Hanna post office and 30 miles east of Heber, which is the nearest station on the Rio Grande & Western Railway.

Coal bed.—Cretaceous age, Mesaverde formation. Thickness, about 24 feet. Roof, a brown shale, which contains a small amount of bone and coal; floor, a brown clayey shale.

The bed was measured and sampled by C. T. Lupton on September 13, 1910, as described below:

Section of coal bed in prospect 15 miles southwest of Hanna.

Laboratory No	10998	
Roof, shale.	*Ft.*	*in.*
Coal, bright and hard	18	6
Shale, brown	[a] 0	1½
Coal, bright and hard	4	5
Coal, dull and slightly dirty	1	9
Floor, clay shale.		
Thickness of bed	24	9½
Thickness of coal sampled	24	8

[a] Not included in sample.

The above sample was taken at the face of the entry about 100 feet from the mine mouth.

HANNA. WINCHESTER PROSPECT.

Sample.—Lignite; Blacktail Mountain field; analysis No. 10997 (p. 105).

Mine.—Winchester, a prospect in sec. 7, T. 1 S., R. 7 W., on Farm Creek about 3 miles east of Hanna post office and 40 miles east of Heber, which is the nearest station on the Rio Grande & Western Railway.

Coal bed.—Mancos. Cretaceous age, Mesaverde formation. Thickness, about 5 feet.

The bed was examined and sampled by C. T. Lupton on September 9, 1910, as described below:

Section of coal bed in Winchester prospect.

Laboratory No.	10997	
Roof, gray sandstone.	*Ft.*	*in.*
Coal, containing lenses of sandstone	[a] 0	1
Coal, good and bright	4	10
Floor, carbonaceous shale.		
Thickness of bed	4	11
Thickness of coal sampled	4	10

[a] Not included in sample.

The above sample was taken about 10 feet down the slope from the entry mouth.

HEBER. CUMMINGS MINE.

Sample.—Subbituminous coal; Blacktail Mountain field; analysis No. 11058 (p. 105).

Mine.—Cummings; an abandoned mine in sec. 31, T. 1 S., R. 10 E., on Current Creek, about 28 miles southeast of Heber, which is the nearest station on the Rio Grande & Western Railway.

Coal bed.—Not named. Cretaceous age, Mesaverde formation. Thickness, about 13 feet.

The sample was taken by C. T. Lupton in September, 1910, and represented a 10-foot 6-inch cut of coal. The mine was more or less caved and the sample was taken from a good surface exposure.

VIRGINIA.

DICKENSON COUNTY.

CLINTWOOD. ELBERT POWERS MINE.

Sample.—Bituminous coal; central Appalachian field; analysis No. 14767 (p. 105).

Mine.—Elbert Powers, a local mine, 3 miles southeast of Clintwood.

Coal bed.—Supposed to be Clintwood. Carboniferous age, Pottsville formation. Roof, bone; floor, coal; cover, 25 feet.

The bed was measured and sampled by W. A. Nelson, September 17, 1912, as described below:

Section of coal bed in Elbert Powers mine.

Laboratory No.	14767	
Roof, not stated.	*Ft.*	*in.*
Coal	[a] 2	7½
Bone	[a] 0	7
Coal	1	11
"Rash"	0	¾
Coal	0	3½
"Rash"	0	¾
Coal	2	4
Floor, coal.		
Thickness of bed	7	10½
Thickness of coal sampled	4	8

[a] Not included in sample.

The section was measured in the main entry 20 feet from the mine mouth.

CLINTWOOD. YEATES MINE.

Sample.—Bituminous coal; central Appalachian field; analysis No. 14766 (p. 105).

Mine.—Yeates, a local drift mine near Clintwood, 5 miles west of the Clinchfield, Carolina & Ohio Railroad.

Coal bed.—Clintwood. Carboniferous age, Pottsville formation. Thickness, uniform. The bed lies flat. Roof, shale; floor, shale.

The bed was measured and sampled by W. A. Nelson on September 21, 1912, as described below:

Section of coal bed in Yeates mine.

Laboratory No	14766
Roof, shale.	*Ft.* *in.*
Bone	0 1
Coal	0 9
Bone	0 2
Coal	0 8½
Bone	0 1¼
Coal	0 3½
Bone	0 1½
Coal	0 3
Bone	0 1
Coal	1 8
Floor, shale.	
Thickness of bed	4 2¾
Thickness of coal sampled	4 2¾

The bed was measured at the end of right entry 1, off the main entry, 100 feet from the main entry.

HENRICO COUNTY.

GAYTON. CARBON HILL MINE.

Sample.—Semibituminous coal; Richmond basin field; analyses Nos. 15050 and 15051; also natural coke analysis No. 15052 (p. 106).

Mine.—Carbon Hill, a slope mine 1 mile from Gayton and connected by private railroad line to the Chesapeake & Ohio Railroad at Lorain.

Coal beds.—B and C. Triassic age, Newark series. The bed has an approximate dip of 35° to the west. Occasional faults are encountered. The B seam contains numerous bands of shale. The floor of the B bed is an underclay; the roof of the B bed and the roof and floor of the C bed are shale. The C bed has been coked in places by the heat of igneous intrusions.

The B bed was measured and sampled at one point and the C bed at two points in the mine by H. I. Smith on November 3, 1912, as described below:

Sections of C bed in the Carbon Hill mine.

Section	A	B
Laboratory No	15050	15052
Roof, shale.	*Ft.* *in.*	*Ft.* *in.*
Natural coke	[a]0 2	2 3
Coal	3 4	[a]1 0
Floor, carbonaceous shale.		
Thickness of bed	3 6	3 3
Thickness of coal sampled	3 4	2 3

[a] Not included in sample.

Section of B bed in the Carbon Hill mine.

Laboratory No	15051
Roof, "draw slate."	*Ft.* *in.*
Coal, hard	1 0
Shale	[a]0 1
Coal streaked with shale	0 6
Shale	0 ½
Coal, very friable	1 0
Shale	0 ½
Coal	0 11
Shale	[a]0 1
Coal	1 3
Floor, clay.	
Thickness of bed	4 11
Thickness of coal sampled	4 9

[a] Not included in sample.

Section A (sample 15050) was taken in the main gangway, opposite room 45 (C bed).

Section B (sample 15052) was taken in the main gangway, opposite room 32 (C bed).

Sample 15051 was measured at the face of right gangway 1 off slope 3, 20 feet ahead of room 21 (B bed).

Notes.—The Carbon Hill mine is opened by a slope which strikes the coal a short distance from the surface and follows the dip of the coal. At the time of sampling the coal was undercut by hand and shot down with a permissible explosive. The coal mined was washed on account of the "slate" partings in B bed being very brittle. It is estimated that one-fourth of the coal mined was lost in washing by breaking to a very fine slack. It was claimed that the mine had been worked at intervals for 100 years, and that it had a probable life of 100 years more. The future tonnage was to be derived from both pillar and advanced workings. Pillars had not, as yet, been pulled. The daily output of the mine was 150 tons, with a maximum day's run of 345 tons. This amount was to be increased to 600 tons. Pickers were employed on both the cars and on belt. The coal loaded was fine, containing practically no lump. There were two loading tracks with a capacity of 15 loaded cars and 15 empty cars. Only the B and C beds were being worked.

For a description of the geological relations of these coal beds, see 19th Annual Rept., 1897–98, U. S. Geol. Survey, 1899, p. 474.

RUSSELL COUNTY.

DANTE. NO. 2 AND NO. 5 MINES.

Sample.—Bituminous coal; Russell Fork field; analyses Nos. 10734, 10735, 10737, and 10738 (p. 106).

Mine.—Nos. 2 and 5, drift mines, 2 miles north of Dante on the Carolina, Clinchfield & Ohio Railway.

Coal bed.—Upper Banner. Carboniferous age, Norton formation. Average thickness, 5 feet 2¼ inches. Roof, gray shale, of good quality; floor, smooth hard shale.

The bed was measured and sampled at four points in the mine by P. M. Riefkin on August 13, 1910, as described below:

Sections of coal bed in No. 2 and No. 5 mines.

Section	A		B		C		D	
Laboratory No.	10734		10735		10737		10738	
	Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Roof, gray shale.								
Coal, hard	..	..	..	..	..	..	0	8
Coal	0	8½	0	6¾	0	4½	1	2¼
Mother coal	..	..	..	..	..	..	0	¾
Coal, hard	0	3¼	0	5	0	3¼	..	..
Coal	0	5¾	0	7	1	1¾	0	4½
Sandstone	[a] 0	¾	[a] 0	1½	[a] 0	1½	[a] 0	1¾
Coal	1	4	1	4	0	10	0	4½
Shale	[a] 0	¾	[a] 0	2¾	[a] 0	½	[a] 0	½
Coal	0	5	0	6	0	5¼	0	6¾
Coal, hard	0	5	0	3¾	0	3½	0	6¼
Coal	1	5½	1	1	0	7	1	6
Mother coal	..	..	..	..	0	¾	..	..
Coal	..	..	..	..	1	½	..	..
Floor, hard smooth shale.								
Thickness of bed	5	2½	5	1¾	5	2½	5	5¼
Thickness of coal sampled	5	1	4	9½	5	½	5	3

[a] Not included in sample.

Section A (sample 10734) was measured at the face of the butt heading off the fourth left drift off the McClure entry, No. 2 mine.

Section B (sample 10735) was measured in room 16, off right heading 3, off the first drift off the McClure entry, No. 2 mine.

Section C (sample 10737) was measured at the face of room 14, right heading 5, off drift 9, No. 5 mine.

Section D (sample 10738) was measured from the butt on left heading 9, off drift 9, No. 5 mine.

Notes.—These mines are opened by drift entries and deliver the coal over the same tipple. At the time of sampling the coal was undercut by machines and shot down with black powder. A new tipple, under construction, was to be equipped to produce various commercial sizes. There was one loading track with a capacity of 40 cars. The combined daily production of the two mines was 1,600 to 1,700 tons, obtained from both advance and pillar workings. It was stated that the production of the mines would probably be increased to about 2,400 tons.

WISE COUNTY.

FLAT GAP. REUBEN BOLLING MINE.

Sample.—Bituminous coal; central Appalachian field; analysis No. 15174 (p. 107).

Mine.—Reuben Bolling. Local mine 2 miles east of Flat Gap and 10 miles from nearest railroad.

Coal bed.—Lower Bolling. Carboniferous age, Pottsville formation. Thickness, uniform, flat; roof, shale; floor, clay; cover, 25 feet.

The bed was measured and sampled by Charles Butts, November 1, 1912, as described below:

Section of coal bed in the Reuben Bolling mine.

Laboratory No.	15174	
	Ft.	*in.*
Roof, shale.		
Coal	[a]0	4
Clay	[a]0	10
Coal	1	5
Clay	0	1
Coal	1	5
Floor, clay.		
Thickness of bed	4	1
Thickness of coal sampled	2	11

[a] Not included in sample.

The bed was measured in the entry 15 feet from mine mouth.

GLAMORGAN. GLAMORGAN NO. 3 MINE.

Sample.—Bituminous coal; central Appalachian field; analysis Nos. 15100 and 15101 (p. 107).

Mine.—Glamorgan No. 3, a drift mine at Glamorgan, on the Wise Terminal Railroad.

Coal bed.—Glamorgan. Carboniferous age, Pottsville formation. Thickness, variable, nearly flat; roof, shale; floor, shale.

The bed was measured and sampled by Charles Butts, November 5, 1912, as described below:

Sections of coal bed in Glamorgan No. 3 mine.

Section	A		B	
Laboratory No.	15100		15101	
	Ft.	*in.*	*Ft.*	*in.*
Roof, shale.				
Coal	1	9	3	8
Bone[a]	[a]0	1	[a]0	1
Coal	1	10	0	7
Bone[a]	[a]0	1¼	..	..
Coal	0	8	..	..
Floor, not stated.				
Thickness of bed	4	5¼	4	4
Thickness of coal sampled	4	3	4	3

[a] Not included in sample.

Section A (sample 15100) was measured at the face of the second dip entry, 6,000 feet from the mine mouth.

Section B (sample 15101) was measured at the face of the Glady entry, 7,000 feet from the mine mouth.

Note.—The coal from this mine is of good coking quality.

PARDEE. PARDEE MINE.

Sample.—Bituminous coal; central Appalachian field; analysis No. 15099 (p. 107.)

Mine.—Pardee; a drift mine at Pardee, on the Roaring Fork Railroad.

Coal bed.—Pardee (Limestone, Parsons). Carboniferous age, Pottsville formation. Thickness, uniform, flat; roof, shale; floor, shale.

The bed was measured and sampled by Charles Butts, November 6, 1912, and represented a 9-foot 7-inch cut of coal.

The section was measured at face of main entry, 2,000 feet from drift mouth.

Note.—The coal was hard.

WASHINGTON.[a]

KING COUNTY.

GRAND RIDGE. GRAND RIDGE MINE.

Sample.—Subbituminous coal; analyses Nos. 11736, 11737, 11738 (p. 107).

Mine.—Grand Ridge, a slope mine one-fourth mile east of Grand Ridge and 3 miles east of Issaquah, in the SE. ¼ NE. ¼, sec. 26, T. 24 N., R. 6 E., on the North Bend branch of the Northern Pacific Railway.

Coal beds.—No. 3, No. 4, and No. 7. Tertiary (Eocene) age, Puget formation. Dip, about 32° NW.

The beds were measured and sampled on February 9, 1911, by G. W. Evans, as described below:

Sections of coal beds in the Grand Ridge mine.

Section	A		B		C	
Laboratory No.	11736		11737		11738	
Roof: Section A, sandstone; section B, shale; section C, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	7	..	..	1	6
Shale, carbonaceous	[a]0	3	[a]0	4	..	..
Clay, white, plastic	[a]0	6	..	..	[a]0	½
Coal	1	11	3	1	1	5
Clay, white, plastic	[a]0	5	..	..	[a]0	½
Bone and coal	[a]1	2	..	..	..	..
Coal	..	..	..	..	1	7
Shale, laminated	[a]0	4	..	..	..	..
Clay, plastic	..	..	..	..	[a]0	5
Coal, somewhat bony	..	..	..	..	0	8
Clay, plastic	..	..	..	..	[a]0	5
Coal, impure	..	..	..	..	[a]0	6
Shale, carbonaceous	..	..	..	..	[a]1	1
Floor: Section A, sandstone; sections B and C, shale.						
Thickness of bed	6	2	3	5	7	7½
Thickness of coal sampled	3	6	3	1	5	2

[a] Not included in sample.

Section A (sample 11736) was collected from the No. 3 bed, on the left side of chute 4. The sample was dry when taken.

Section B (sample 11737) was collected from the No. 4 bed, on the north side where the rock tunnel cuts the bed. The sample was damp when taken.

Section C (sample 11738) was collected from bed No. 7. The sample was wet when taken.

Note.—The output from each of beds Nos. 3, 4, and 7 was 250 tons a day.

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[a] For a detailed discussion of Washington coals see U. S. Geol. Survey Bull. 474.

KITTITAS COUNTY.

ELLENSBURG. BUCHANNON PROSPECT.

Sample.—Bituminous coal; Manastash field; analysis No. 13267 (p. 107.)

Locality.—Buchannon prospect, in sec. 16, T. 18 N., R. 15 E., about 25 miles west of Ellensburg, which is on the Northern Pacific and the Chicago, Milwaukee & Puget Sound Railways.

Coal beds.—No. 10d. Tertiary (Eocene) age, Manastash formation. Thickness, 3 feet 9½ inches.

The bed was measured and sampled by E. J. Saunders on August 2, 1911, as described below:

Section of coal bed in prospect, 25 miles west of Ellensburg.

Laboratory No.	13267	
	Ft.	*in.*
Bone and coal	0	3
Shale, crushed	[a]0	8
Coal and bone	0	5
Coal (lenses)	0	3½
Coal, soft, with shale	0	5
Coal	1	5
Coal, soft, with shale	0	4
Thickness of bed	3	9½
Thickness of coal sampled	3	1½

[a] Not included in sample.

The sample was collected from the face of the entry, 35 feet northeast of its mouth. The sample was wet when taken.

ELLENSBURG. PROSPECT.

Sample.—Bituminous coal; Manastash field; analysis No. 13266 (p. 107).

Locality.—Prospect in sec. 22, T. 18 N., R. 15 E., about 25 miles west of Ellensburg, which is on the Northern Pacific and the Chicago, Milwaukee & Puget Sound Railways.

Coal bed.—No. 7. Tertiary (Eocene) age, Manastash formation. Roof and floor, shale. Thickness, 2 feet 10½ inches.

The bed was measured and sampled by E. J. Saunders on August 1, 1911, as described below:

Section of coal bed in prospect, 25 miles west of Ellensburg.

Laboratory No.	13266	
Roof, shale	*Ft.*	*in.*
Coal, bony	0	8½
Coal	0	6
Bone	[a]0	1
Coal	1	4
Coal and bone (crushed)	0	3
Floor, shale.		
Thickness of bed	2	10½
Thickness of coal sampled	2	9½

[a] Not included in sample.

The sample was collected from face of the entry, 30 feet from the mouth. It was dry when taken.

Note.—The mine was idle at the time of sampling.

ELLENSBURG. PROSPECT.

Sample.—Bituminous coal; Manastash field; analysis No. 13268 (p. 107).

Locality.—Prospect in sec. 22, T. 18 N., R. 15 E., about 25 miles west of Ellensburg, which is on the Northern Pacific and the Chicago, Milwaukee & Puget Sound Railways.

Coal beds.—No. 9c. Tertiary (Eocene) age, Manastash formation.

The bed was measured and sampled by E. J. Saunders on August 1, 1911, as described below:

Section of coal bed in prospect, 25 miles west of Ellensburg.

Laboratory No.	13268	
Roof, shale.	*Ft.*	*in.*
Shale, carbonaceous	[a] 0	2
Coal, slightly bony	0	6
Coal, soft, and shale	1	0
Coal	1	8
Shale, carbonaceous	[a] 0	2
Floor, shale.		
Thickness of bed	3	6
Thickness of coal sampled	3	2

[a] Not included in sample.

The sample was collected from the face of the entry, 40 feet from its mouth. It was wet when taken.

ROSLYN. ROSLYN NO. 1 AND NO. 2 MINES.

Sample.—Bituminous coal; analyses Nos. 11522, 11523, 11524, 11525, and 11256 (p. 107).

Mines.—Roslyn No. 1 and No. 2, slope mines at Roslyn, in secs. 8 and 9, T. 20 N., R. 15 E., on a branch of the Northern Pacific Railway, with which line it connects at Clealum.

Coal bed.—Roslyn. Eocene age, Roslyn formation. Average thickness, 4 feet 4 inches; dip of bed in No. 1 mine, 9° W., and in No. 2,18° W.; roof, gray shale, 10 feet thick in Mine No. 1, and 18 to 20 feet thick in mine No. 2; floor in each mine, smooth, hard clay, in places more or less shaly; cover, at points where samples were taken, 30 to 120 feet thick.

The bed was measured and sampled at four points in the mine by H. M. Wolflin, on January 2, 1911, as described below:

Sections of coal bed in Roslyn No. 1 and No. 2 mines.

Section	A		B		C		D	
Laboratory No.	11522		11523		11524		11526	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, soft	1	1½	1	1¼	1	1½	2	6
Coal, slaty	..	..	0	¼	..	..	..	..
Coal, hard	0	1¼	..	..	0	½	..	..
Coal, soft	0	9½	1	5	1	¼	..	..
Coal, sulphurous	0	1	[a] 0	2	0	½	..	..
Coal, soft	0	6¼	..	..	..	..	..	..
Coal, bony	0	½	..	..	..	..	0	2
Coal, soft	1	6	1	3	1	9	1	7
Floor, clay.								
Thickness of bed	4	2¼	3	11½	3	11¾	4	3
Thickness of coal sampled	4	2¼	3	9½	3	11¾	4	3

[a] Not included in sample.

Section A (sample 11522) was measured at the face of the southwest slant entry of the No. 1 mine, at a depth of 80 feet below the surface.

Section B (sample 11523) was measured at the face of east entry 1, in the No. 1 mine, 120 feet below the surface.

Section C (sample 11524) was measured at the face of west entry 2, in the No. 1 mine, 100 feet below the surface.

Section D (sample 11526) was measured at the face of the main slope of the No. 2 mine, 30 feet below the surface.

A composite sample was made by mixing samples 11522, 11523, and 11324. The results of an ultimate analysis of the sample are given under laboratory No. 11525.

Notes.—At the time of sampling none of the coal from these mines was shipped as run-of-mine. About 50 per cent of the coal passed through 2¾ to 3 inch bar screens. The coal was not hand picked. There were three loading tracks, with a capacity for 30 or 40 cars. Each mine had its own tipple and screening plant. Each mine was opened by a slope; double entry, room-and-pillar system of mining used; black powder used for breaking coal after undercutting. The output of the No. 1 mine was 400 tons a day. The No. 2 mine was equipped for handling about 500 tons a day. Mine No. 1 had an unmined area of 400 to 500 acres and No. 2 of about 120 acres.

THORP. WILSON MINE.

Sample.—Bituminous coal; Manastash field; analysis No. 13269 (p. 108).

Mine.—J. Wilson, in sec. 33, T. 19 N., R. 16 E., 10 miles west of Thorp, a station on the Northern Pacific Railway.

Coal bed.—No name. Tertiary (Eocene) age, Manastash formation. The bed, including shale partings, is about 9 feet thick. Roof, gray shale; floor, carbonaceous shale.

The bed was measured by E. J. Saunders on July 17, 1911.

Section of coal bed in Joseph Wilson mine.

	Ft.	*in.*
Roof, gray shale.		
Shale, carbonaceous, slightly graphitic	1	2
Coal	0	3
Shale, carbonaceous, graphitic	0	9
Coal, impure and crushed	1	3
Shale, graphitic	0	11
Coal	1	0
Shale, carbonaceous	[a]0	2
Coal	[a]0	8
Shale, carbonaceous	[a]0	4
Slate	[a]1	0
Coal	[a]1	8
Floor, carbonaceous shale.		
Thickness of bed	9	2

[a] Not mined.

The sample analyzed represented run-of-mine coal taken from bunkers. It was dry when collected.

LEWIS COUNTY.

MENDOTA. MENDOTA NO. 1 MINE.

Sample.—Subbituminous (cannel) coal; analyses Nos. 12305 and 14084 (p. 108).

Mine.—Mendota No. 1, a slope mine at Mendota, about 6 miles east of Centralia, on the Centralia and Eastern branch of the Northern Pacific Railway.

Coal bed.—Locally termed No. 3; Tertiary (Eocene) age, Puget formation. Dip 5 to 12 per cent. Thickness, approximately 10 feet with several partings of clay ½ to 1½ inches thick; roof, soft, moist, crumbly, massive sandstone; in most sections of the mine, between the coal and the roof there is a band of soft, moist, gray clay about 2 inches thick. The sandstone roof is about 6 to 12 feet thick and is overlain with hard gray shale. The floor consists of a fairly hard and smooth sandy clay.

The coal bed was sampled at two points in the mine by H. M. Wolflin on May 22, 1911, and May 31, 1912.

Sample 12305 represented selected coal cut from the rib on the second level. This coal had the appearance of cannel and occurs in lenses and as vertical streaks. The sample was cut from a vertical streak 1½ inches wide.

Sample 14084 was cut from rib, south level, and represented a selected sample of "cannel" coal that occurs in lenses in the coal bed. These lenses are irregular in size, shape, and occurence and consist of coal that is soft, easily lighted by a match, burns with a smoky flame, and gives off a strong odor of coal tar.

Notes.—The Mendota No. 1 mine was opened by a slope driven across the pitch at an angle of 35° to 40° to the direction of the dip. At the time of sampling the coal was shot off the solid with FFF black blasting powder. The shaker screens were 5 feet 4 inches by 10 feet with holes 3 by 6 inches and 2¼ inches in diameter. The shaker screens for the smaller sizes of coal were 5 feet 4 inches by 10 feet with ⅜-inch holes. The revolving screen for washed coal was 14 feet long—8 feet with ¼-inch openings and 6 feet with ⅞-inch openings. The sizes produced in washing were on ⅞-inch, on ¼-inch and through ¼-inch holes. Three loading tracks, with a capacity of 50 empty cars and 30 loaded cars, were available. The mine had a daily output of 300 tons with a maximum day's run of 800 tons. About 2,000 acres of coal remained to be taken out from this opening.

MORTON. PROSPECT.

Sample.—Subbituminous coal; analysis No. 12563 (p. 108).

Locality.—Prospect in the SW. ¼ sec. 12, T. 12 N., R. 4 E., 2 miles southeast of Morton, which is the southern terminus of the Tacoma Eastern Railroad.

Coal bed.—Edlund. Tertiary (Eocene) age, Manastash formation. Strike, S. 70° E.; dip 16° S., 20° W.; roof and floor, sandstone. The bed is reported to be 7 feet thick near the surface, but further in is much faulted and thins to 12 inches.

The bed was measured and sampled by A. J. Collier on August 18, 1911. The sample was taken 4 feet from the surface and represented 4 feet of coal.

Note.—At the time of sampling the prospect had been abandoned owing to faults and thinning out of the bed.

MORTON. PROSPECT.

Sample.—Subbituminous coal; analysis No. 12564 (p. 108).

Locality.—Prospect in the SW. ¼ sec. 6, T. 12 N., R. 5 E., 2 miles east of Morton, which is the southern terminus of the Tacoma Eastern Railroad.

Coal bed.—Not named. Tertiary (Eocene) age, Manastash formation. Dip at surface 22° NE., increasing with depth. Thickness reported to decrease from 6 feet to 2 feet with depth. Bed faulted at end of workings 350 feet from mouth of the tunnel.

The bed was measured and sampled by A. J. Collier on August 24, 1911, as described below:

Section of coal bed in prospect 2 miles east of Morton.

Laboratory No.	12564	
Roof, sandstone.	*Ft.*	*in.*
Coal	[a] 0	3
Bone	[a] 0	7
Coal	2	0
Floor, sandstone.		
Thickness of bed	2	10
Thickness of coal sampled	2	0

[a] Not included in sample.

The sample was collected in the drift 20 feet from the mouth.

Note.—The prospect had been abandoned owing to faults and thickness of bed.

MORTON. PROSPECT.

Sample.—Semianthracite coal; analysis No. 12565 (p. 108).

Locality.—Prospect in sec. 14, T. 13 N., R. 4 E., 4 miles north of Morton, which is the southern terminus of the Tacoma Eastern Railroad.

Coal bed.—No name. Tertiary (Eocene) age. Thickness, 12 to 15 feet.

The bed was measured and sampled by A. J. Collier on August 16, 1911, as described below:

Section of coal bed in prospect 4 miles north of Morton.

Laboratory No.	12565	
	Ft.	*in.*
Coal with thin sandstone partings	[a] 6+	0
Coal and bone	[a] 2	
Coal	6+	0
Thickness of bed	14+	0
Thickness of coal sampled	6+	0

[a] Not included in sample.

The sample was collected in the entry, about 60 feet from the mouth of the entry.

PIERCE COUNTY.

ASHFORD. PROSPECT.

Sample.—Semibituminous coal; analysis No. 12581 (p. 108).

Locality.—Prospect in the NW. ¼ sec. 15, T. 15 N., R. 6 E., at Ashford on the Tacoma Eastern Railway.

Coal bed.—Misqually Chief. Tertiary (Eocene) age, Puget formation. Roof, altered shale; thickness, 13 feet 4 inches.

The bed was measured and sampled by G. W. Evans on August 26, 1911, as described below:

Section of coal bed in prospect at Ashford.

Laboratory No.	12581	
Roof, altered shale.	*Ft.*	*in.*
Shale, carbonaceous	[a] 0	5
Coal	0	6
Sandstone	[a] 0	4
Coal	1	2
Shale	[a] 0	4
Coal	5	3
Bone	0	1
Coal and dirt	1	2
Coal	2	0
Shale	[a] 0	2
Coal	0	5
Shale	[a] 0	2
Coal, shaly	1	2
Clay, soft	[a] 0	2
Floor, not stated.		
Thickness of bed	13	4
Thickness of coal sampled	11	9

[a] Not included in sample.

The sample was collected in the entry 100 feet from its mouth. It was wet when taken.

FAIRFAX. PROSPECT.

Sample.—Bituminous coal; analysis No. 12478 (p. 108).

Locality.—Prospect in the SW. ¼ sec. 2, T. 17 N., R. 6 E., 2 miles south of Fairfax, which is on a branch line of the Northern Pacific Railway.

Coal bed.—Montezuma. Tertiary (Eocene) age, Puget formation. Roof and floor, strong carbonaceous shale.

The bed was measured and sampled by G. W. Evans on July 25, 1911, as described below:

Section of coal bed in a prospect 2 miles south of Fairfax.

Laboratory No.	12478	
Roof, carbonaceous shale.	*Ft.*	*in.*
Shale, soft brown	[a]0	10
Coal, soft impure	[a]0	10
Coal, impure, varying in hardness	[a]3	11
Coal, good	4	0
Floor, carbonaceous shale.		
Thickness of bed	9	7
Thickness of coal sampled	4	0

[a] Not included in sample.

The sample was collected from the face of the entry, 45 feet from the mouth of the entry. It was wet when taken.

FAIRFAX. PROSPECT No. 1.

Sample.—Bituminous coal; analysis No. 12496 (p. 109).

Locality.—Prospect No. 1 in the NE. ¼ NW. ¼ sec. 34, T. 18 N., R. 6 E., one-half mile south of Fairfax, which is on a branch line of the Northern Pacific Railway.

Coal bed.—No. 1. Tertiary (Eocene) age, Puget formation.

The bed was measured and sampled by G. W. Evans on August 1, 1911, as described below:

Section of coal bed in Prospect No. 1.

Laboratory No.	12496	
Roof, shale.	*Ft.*	*in.*
Coal, impure	1	6
Coal	1	4
Shale, 8 inches thick.		
Shale and clay, 7 inches thick.		
Floor, sandy shale.		
Thickness of bed	2	10
Thickness of coal sampled	2	10

The sample was collected in the entry, 35 feet from its mouth. It was wet when taken.

FAIRFAX. PROSPECT No. 2.

Sample.—Bituminous coal; analysis No. 12495 (p. 109).

Locality.—Prospect No. 2 in the NE. ¼ NW. ¼ sec. 34, T. 18 N., R. 6 E., one-half mile south of Fairfax, which is on a branch line of the Northern Pacific Railway.

Coal bed.—No. 2. Tertiary (Eocene) age, Puget formation.

The bed was measured and sampled by G. W. Evans on August 1, 1911, as described below:

Section of coal bed in Prospect No. 2.

Laboratory No.	12495	
Roof, shale.	*Ft.*	*in.*
Clay, soft dark	[a]0	4
Coal, dirty	2	0
Coal	0	11
Coal, bony	[a]1	2
Bone	1	0
Floor, sandy shale.		
Thickness of bed	5	5
Thickness of coal sampled	3	11

[a] Not included in sample.

The sample collected was in the entry. 35 feet from its mouth. It was wet when taken.

WILKESON. GALE CREEK MINE.

Sample.—Bituminous coal; analyses Nos. W31921 to W31926, W32104 to W32108, 14390, W32145 to W32148, W32158 to W32161, and W32211 to W32214 (pp. 109, 110).

Mine.—Gale Creek, a drift and slope mine at Wilkeson, about 30 miles southeast of Tacoma, on a branch of the Northern Pacific Railway.

Coal bed.—Queen. Tertiary (Eocene) age, Puget group. Average thickness, 3 feet 8 inches; dip, 8° to 85° SW., with a probable average of about 30°. The cleat is indistinct and there are many faults and rolls throughout the bed. Roof, "draw slate" 5 inches thick, above which is a hard gray shale extending to a cap rock 30 to 40 feet above; floor, hard, smooth, gray clay, shaly in places.

The bed was measured and sampled at six points in the mine by H. M. Wolflin on June 18 and 25, 1912, and six other samples were taken in duplicate from 27 carloads, as described below:

Sections of coal bed in Gale Creek mine.

Section	A		B		C		D		E		F	
Laboratory No.	W31921 W31922		W31923 W31924		W31925 W31926		W32106		W32107		W32108	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
"Draw slate"	..	..	..	..	..	..	[a]0	5	..	..	[a]0	5
Coal	..	..	0	1	..	..	..	..	..	..	..	..
Bone (local)	..	..	0	¼	..	..	[a]0	1	..	..	..	..
Coal	1	11¾	2	½	2	½	1	8½	2	2	2	1
Clay-streaked coal (?)	[a]0	4½	[a]0	4	[a]0	3½	[a]0	2	[a]0	3½	[a]0	6
Coal	1	4	1	2	1	5	1	4	1	2	1	8
Floor, shaly clay.												
Thickness of bed	3	8¼	3	7¾	3	9	3	8½	3	7½	4	8
Thickness of coal sampled	3	3¾	3	3¾	3	5½	3	½	3	4	3	9

[a] Not included in sample.

Section A (samples W31921 and W31922) was measured at the face of gangway 1, off new slope.

Section B (samples W31923 and W31924) was measured at breast 16 at the face of counter No. 1 north, off new slope.

Section C (samples W31925 and W31926) was measured on the pillar of breast 10, off north gangway 1, off new slope.

Section D (sample W32106) was measured in the air course of the new slope at No. 1 lift, below north gangway 1.

Section E (sample W32107) was a pillar sample, measured in the first crosscut, off breast 14, north gangway 1, off new slope.

Section F (sample W32108) was a face sample, measured in the crosscut on the new slope opposite south gangway 2.

A composite sample was made by mixing samples 31921 to 31926, inclusive, and 32106 to 32108, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14390.

Duplicate samples Nos. W32104 and W32105 were taken from four cars while being loaded at the mine.

Duplicate samples W32145 and W32146 were taken from four cars while being loaded at the mine.

Sample W32147 was taken from four cars while being loaded at the mine.

Duplicate samples W32158 and W32159 were taken from four cars while being loaded at the mine.

Duplicate samples W32160 and W32161 were taken from four cars while being loaded at the mine.

Duplicate samples W32211 and W32212 were taken from four cars while being loaded at the mine.

Duplicate samples W32213 and W32214 were taken from three cars while being loaded at the mine.

Coal that passed over the screens, constituting about 15 per cent of the total, was dumped into the cars, but no special note was made as to which cars were partly loaded with lump and which were loaded with the coal going through the 2½-inch screen. The screened coal passed through the washery before being loaded. The above car samples were obtained by taking about 50 pounds of coal from each car as it was being loaded. The 50-pound lots from three or four cars were then combined and the sample quartered down to laboratory size.

Notes.—The Gale Creek mine was opened by slopes and drifts, and the coal was mined by the breast-and-pillar method. All of the undercutting done was by hand; the larger part of the coal was shot off the solid with 30 per cent dynamite. In places the "draw slate" fell with the coal. The tipple was equipped with shaking screens 5 by 16 feet with 2½-inch holes. None of the coal was shipped as run-of-mine. Approximately 75 to 85 per cent of the coal passed through the screens; after having been screened it went to the washery. The oversize passed on to a picking belt, where it was carefully hand picked prior to being loaded on the car. The screenings were washed, but only one size was produced, although the plant was equipped for yielding two sizes. There was one loading track with a capacity for 12 empty and 15 loaded cars. The storage bins had a capacity for about 850 tons. There was 320 acres of coal land tributary to the various openings. The larger part of the coal was worked out down to the 400-foot level. One-half to three-fourths of the coal produced was from advance workings, the remainder being obtained from robbing the pillars. The mine had a daily production of 100 tons, with a maximum capacity of about 200 tons. Development work was under way and the capacity was to be increased to 250 to 300 tons a day.

For description and analyses of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 221, 899.

WEST VIRGINIA.

BRAXTON COUNTY.

BRAXTON. BRAXTON MINE.

Sample.—Bituminous coal; analysis No. 13556 (p. 110).

Mine.—Braxton; interior district; a mine at Braxton post office, on Coal & Coke Railway.

Coal bed.—Pittsburgh. Carboniferous age; Monongahela formation. Thickness, 4 to over 7 feet; average, nearly 7 feet. The roof is shale in most parts of the mine, but locally it is sandstone, in which case the coal is thinner than elsewhere. Bands of clay from a fraction of an inch to 3 or 4 feet thick cut across the bed in numerous places. Floor, shale.

The sample was collected about 325 feet southeast from the mouth of the mine by E. R. Lloyd, on March 18, 1912. The thickness of the bed where sampled was 5 feet 3 inches, clear coal, all of which was represented in the sample.

The output of the mine at the time of sampling was about two carloads per week.

BROOKE COUNTY.

COLLIERS. LEWIS-FINLEY NO. 1 MINE.

Sample.—Bituminous coal; analyses Nos. 15671, 15672 (p. 111).

Mine.—Lewis-Finley No. 1; Wheeling district; a drift mine, on the Pennsylvania Railroad.

Coal bed.—Pittsburgh. Carboniferous age; Monongahela formation. The bed has an average thickness of about 4½ feet and lies nearly flat. Roof, clay or bony coal.

The bed was measured and sampled at two places by D. D. Condit on January 6, 1913, as described below:

Sections of coal bed in Lewis-Finley No. 1 Mine.

		A		B
Section		A		B
Laboratory No.		15671		15672
Roof, clay or bony coal.				
Coal unmeasured.				
Clay, averaging 10 inches.	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	[a]0	7	..	..
Coal	1	10	2	3
Shale	[a]0	½	[a]0	½
Coal	[a]0	3½	[a]0	3
Shale	[a]0	¼	[a]0	1
Coal	0	11½	1	9½
Coal, bony	[a]0	¼	..	..
Coal	0	8½	..	..
Coal, high in "sulphur"	[a]0	2	[a]0	3
Floor, clay.				
Thickness of bed	4	7½	4	8
Thickness of coal sampled	3	6	4	½

[a] Not included in samples.

Section A (sample 15671) was measured in room 1, off butt entry 6, off the main entry.

Section B (sample 15672) was measured in room 77, off entry 10, side No. 2, 5,800 feet northwest of the mine mouth. Cover, 100 feet.

FAYETTE COUNTY.

CAPERTON. SEWELL MINE.

Sample.—Bituminous coal; New River field; analyses Nos. 14344, 14345, 14346, 14347, 14348, 14349, and 14350 (p. 111).

Mine.—Sewell; Kanawha-New River district; a drift mine at Caperton station, on the main line of the Chesapeake & Ohio Railway.

Coal bed.—Sewell. Carboniferous (Pottsville) age. Sewell formation. Average thickness, 3 feet 8 inches; dip, 1½ per cent northeast. The cleat is well marked, and there are no rolls or faults in this mine. The roof consists of a smooth "draw slate" 3 to 9 inches thick, above which is a sandstone cap. Floor, hard, smooth shale. Pieces of the roof occasionally mixed with the coal when loading, but none of the floor material was taken up with the coal. Cover at points of sampling, 250 to 400 feet.

The bed was measured and sampled at five points in the mine by G. H. Deike on July 11, 1912, as described below:

Sections of coal bed in Sewell Mine.

Section	A		B		C		D		E	
Laboratory No.	14344		14345		14346		14348		14349	
Roof, sandy shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	9	0	6	0	11	0	6	0	10
Coal, gray	0	2½	0	1	0	3	0	3	0	1½
Coal	0	7	..	..	0	8½	0	6½	0	8½
Coal, gray	0	1½	..	..	0	1½	0	1½	0	1
Coal	1	6	1	10	2	0	1	7½	1	4½
Mother coal	0	½	..	..	..	..	0	½	0	½
Coal	0	9½	..	..	..	..	0	7½	0	9
Coal, bony	..	..	[a]0	4	[a]0	2½	..	..	[a]0	2
Floor, clay.										
Thickness of bed	4	0	2	9	4	2½	3	8½	4	1
Thickness of coal sampled	4	0	2	5	4	0	3	8½	3	11

[a] Not included in sample.

Section A (sample 14344) was measured at the face of a room turned off at the face of the drainage entry (Sugar Camp No. 3 workings).

Section B (sample 14345) was measured at the face of the last break-through in the third right air course (Sugar Camp No. 1 workings).

Section C (sample 14346) was measured at the face of the thirteenth right entry, 5,000 feet northeast of mine mouth. (Old Hill workings.)

Section D (sample 14348) was measured on pillar at the last room off cross entry 11. (Old Hill workings.)

Section E (sample 14349) was measured on the pillar of right entry 2. (Sugar Camp No. 2 workings.)

Two composite samples were made: (1) by mixing samples 14344, 14345, and 14346 for an ultimate analysis, the results of which are given under laboratory No. 14347; (2) by mixing samples 14348 and 14349 for an ultimate analysis, the results of which are given under laboratory No. 14350.

Notes.—The Sewell mine is opened by a number of drift entries, and the coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut both by hand and by electric chain machines and broken down with FFF black powder and permissible explosives. All of the coal was shipped as run-of-mine. There was a screening plant with 12-foot bar screens spaced 1¼ inches apart. The coal was clean and needed no picking when loading. There were 2 loading tracks with a capacity for 14 empty and 20 loaded cars. About 50 per cent of the coal was obtained from advance workings. There was approximately 300 acres of unmined coal tributary to the various entries. The output of the mine was 300 tons a day, with a maximum day's run of 550 days.

DUNLOOP. DUNN LOOP NO. 2 MINE.

Sample.—Semibituminous coal; New River field; analyses Nos. 14383, 14384, 14385, 14386, 14387, 14388, and 14389 (p. 111).

Mine.—Dunn Loop No. 2; Kanawha-New River district; at Dunloop, with railroad connections to the Chesapeake & Ohio Railway. Another road is under construction, the Glen Jean & Eastern, which will connect with the Virginian Railway. This will give an additional outlet for coal shipments.

Coal bed.—Sewell, Carboniferous (Pottsville) age, Sewell formation. Average thickness, 5 feet 6 inches; dip, 1¾° northwest. Roof, "draw slate" 4 to 6 inches thick, above which is a stratum of soft gray shale. There is a sandstone cap rock 20 feet above the coal. Floor, smooth, hard fire clay, and shale. Cover at points of sampling, 75 to 200 feet.

The bed was measured and sampled at five points in the mine by J. J. Rutledge on July 10, 1912, as described below:

Sections of coal bed in Dunn Loop No. 2 mine.

Section	A		B		C		D		E	
Laboratory No	14383		14384		14385		14387		14388	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	9	2	7	2	6	2	2¼	2	9
Cannel coal	0	½	0	3	0	3	0	2¾	0	3
Coal	2	11	2	9	2	11	2	1	2	10
Shale	..	..	..	..	..	..	[a]0	1	..	..
Coal	..	..	..	..	..	..	0	7	..	..
Floor, fire clay, shale.										
Thickness of bed	5	8½	5	7	5	8	5	2	5	10
Thickness of sampled	5	8½	5	7	5	8	5	1	5	10

[a] Not included in sample.

Section A (sample 14383) was measured at the last break-through, to the right on the fifth main entry, 500 feet northeast of the mouth of the fifth main entry.

Section B (sample 14384) was measured at the face of room 20, off right entry 4, off the fourth main entry 1,600 feet southeast of the mouth of the fourth main entry.

Section C (sample 14385) was measured at the face of room 2, off the first left air course off the sixth main entry, 1,200 feet east of the mouth of the sixth main entry.

Section D (sample 14387) was measured at the face of room 5, off the air course of the third main entry, 500 feet southeast of the mouth of the third main entry.

Section E (sample 14388) was measured on pillar 17 on the sixth main entry, 1,040 feet southeast of the mouth of the sixth main entry.

Two composite samples were made: (1) by mixing samples 14383, 14384, and 14385 for an ultimate analysis, the results of which are shown under laboratory No. 14386; (2) by mixing samples 14387 and 14388 for an ultimate analysis, the results of which are shown under laboratory No. 14389.

Notes.—The Dunn Loop No. 2 mine is opened by six drift entries, and the coal from these various openings all goes to the same tipple. The coal is mined by the room-and-pillar system. At the time the mine was visited the coal was undercut by hand and shot down with FFF black blasting powder and permissible explosives. Thirty per cent dynamite was used for brushing the roof. The tipple was equipped with shaking screens with openings 5½ by 3½ inches and 2½ by 1½ inches in size. Pickers were employed for picking the coal on a traveling belt. The daily production of the mine was 1,000 tons. The coal had a good appearance on the cars, the lumps being large but slightly broken by the loading system. There were 3 loading tracks, with a capacity for 29 empty and 32 loaded cars. All of the coal was shipped as run-of-mine. About 50 per cent of the coal was obtained from new workings and the remainder from the older parts of the mine.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 915–916.

ELVERTON. ELVERTON MINE.

Sample.—Semibituminous coal; New River field; analyses Nos. 14338, 14339, 14340, 14341, 14342, and 14343 (p. 112).

Mine.—Elverton; Kanawha-New River district; a drift mine, ½ miile west of Elverton station on the main line of the Chesapeake & Ohio Railway.

Coal bed.—Carboniferous (Pottsville) age. Sewell formation. Average thickness, 4 feet; dip 1½ per cent northwest. Roof, heavy smooth shale above which there is a sandstone cap rock. The coal is practically free from regular partings or impurities. The floor consists of about 4 inches of bony coal, below which is a hard smooth shale. Cover, at points where the samples were taken, ranges from 100 to 400 feet.

The coal bed was measured and sampled at four points in the mine by G. H. Deike, July 10, 1912, as described below:

Sections of coal bed in Elverton mine.

Section	A		B		C		D	
Laboratory No	14338		14339		14341		14342	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, gray	..	..	..	..	0	1½	..	..
Coal	0	9	0	10	0	8½	0	10½
Coal, gray	0	1	0	1½	0	3	0	1½
Coal	0	7	0	7½	0	4½	0	10
Coal, gray	0	2	0	1	..	..	0	1
Mother coal	..	..	..	..	0	½	..	..
Coal	1	4	0	7	1	9¼	1	0
Mother coal	..	..	..	..	0	¼	0	½
Coal	..	..	..	..	0	3⅜	..	..
"Sulphur" band	..	..	0	⅜	0	¼	..	..
Coal	..	..	0	6	0	7½	0	11
Mother coal	0	¼	0	¼	..	..	0	2
Coal	0	5½	0	5	..	..	0	7
Floor, bone and shale.								
Thickness of bed	3	4¾	3	2⅝	4	2⅝	4	7½
Thickness of coal sampled	3	4¾	3	2⅝	4	2⅝	4	7½

Section A (sample 14338) was measured at the face of the old Bayne entry, 3,900 feet southwest of No. 2 entrance.

Section B (sample 14339) was measured at the corner of the break-through, 10 feet from the face of the main entry, 3,900 feet southwest of No. 2 entrance.

Section C (sample 14341) was measured on pillar of room 2, left entry 1 off left entry 5, 3,100 feet south of No. 2 entrance.

Section D (sample 14342) was measured on the pillar of room 3, right entry 5, 3,100 feet southwest of No. 2 entrance.

Two composite samples were made: (1) by mixing samples 14338 and 14339 for an ultimate analysis, the results of which are shown under laboratory No. 14340; (2) by mixing samples 14341 and 14342 for an ultimate analysis, the results of which are shown under laboratory No. 14343.

Notes.—The Elverton mine is opened by a drift entry and the coal mined by the room-and-pillar system. At the time of sampling the coal was undercut both by hand and by electric chain machines and broken with FFF black powder. Dynamite was used for brushing the roof. The tipple was equipped with shaking screens 5 by 12 feet with openings 5 inches in diameter and also another set of screens 5 by 12 feet with openings 1½ inches in diameter. About 50 per cent of the coal passed through the first set of screens. None of the coal was shipped as run-of-mine. Pickers were employed on the cars when loading coal. The screenings were used for making coke. The coal was very clean, the lumps large, and presented a good appearance on the cars. There were 3 loading tracks with a capacity for 30 empty and 40 loaded cars. There were also storage bins with a capacity for 400 tons of coal. There was about 700 acres of coal tributary to this mine. About one-half of the coal was derived from advance workings and the remainder from pillars. The production of the mine was 250 tons a day with a maximum day's run of 650 tons.

HAWKS NEST. MILL CREEK MINE.

Sample.—Semibituminous coal; New River field; analyses Nos. 14358, 14359, 14360, 14361, and 14362 (p. 112).

Mine.—Mill Creek; Kanawha-New River district; a drift mine about one-half mile north of Hawks Nest station on the main line of the Chesapeake & Ohio Railway.

Coal bed.—Sewell. Carboniferous (Pottsville) age, Sewell formation. Average thickness, 3 feet. The bed is practically horizontal. Roof, heavy shale and bony coal 6 to 18 inches thick, above which is a sandstone cap rock. Floor, hard smooth bony coal. Material from the roof and floor did not become mixed with the coal when mining. Cover, at the points where the samples were taken, approximately 600 feet.

The coal bed was measured and sampled at four points in the mine by H. I. Smith and G. H. Deike, July 12, 1912, as described below:

Sections of coal bed in Mill Creek mine.

Section	A		B		C		D	
Laboratory No.	14358		14359		14360		14361	
Roof, shale and bony coal.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Cannel coal	..	..	0	½	..	..	..	..
Coal	1	8	1	1	0	9	1	1½
Coal, gray	0	2½	..	..	..	..	0	2
Coal	0	11	..	..	..	..	0	8
"Sulphur"	Streak.		Streak.		..	..	0	½
Coal, gray	..	..	0	2½	0	2	..	..
Coal	0	10¼	1	8	1	6¼	0	8
Bone	0	½	..	..	..	..	..	..
Floor, bony coal.								
Thickness of bed	3	8¼	3	0	2	5¼	2	8
Thickness of coal sampled	3	8¼	3	0	2	5¼	2	8

Section A (sample 14358) was measured at the face of left entry 1 at room 49.

Section B (sample 14359) was measured at the face of left entry 2 at the neck of room 29.

Section C (sample 14360) was measured at the face of the third left air course.

Section D (sample 14361) was measured at the face of right entry 5.

A composite sample was made by mixing samples 14358, 14359, 14360, and 14361 for an ultimate analysis, the results of which are shown under laboratory No. 14362.

Notes.—The Mill Creek mine was opened by drift entries and the coal mined by the room-and-pillar system. At the time of sampling the coal was undercut by compressed-air punchers and broken down with a permissible explosive, which was also used for brushing the roof. None of the coal was shipped as run-of-mine. The screening plant consisted of bar screens 4½ by 12 feet with 3-inch spaces, through which about 50 per cent of the coal passed. The coal was clean and the lumps were large and presented a good appearance when loaded on the cars. There were two loading tracks with a capacity for 12 empty and 12 loaded cars. About two-thirds of the coal was obtained from advance work and the remainder from pillars. The daily output was 250 tons with a maximum day's run of 400 tons.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 924–925.

KAY MOOR. KAY MOOR NO. 1 AND NO. 2 MINES.

Sample.—Semibituminous coal; New River field; analyses Nos. 14294, 14295, 14296, 14297, 14298, 14299, 14401, 14402, 14403, and 14404 (p. 113).

Mine.—Kay Moor No. 1 and No. 2, Kanawha-New River district; three-fourths mile south of Kay Moor, on the south side of New River with railway connections to the Chesapeake & Ohio Railway. No. 2 mine is also on the south side of the river with railway connections to the Chesapeake & Ohio Railway at South Fayette.

Coal bed.—Carboniferous (Pottsville) age, Sewell formation. Average thickness about 3 feet 6 inches; slight dip to the west. Roof, smooth sandstone 18 inches thick; floor, rough, hard clay. Material from the floor or roof did not become mixed with the coal in mining.

The coal bed was measured and sampled at five points in No. 1 mine and at three points in No. 2 mine by H. I. Smith on July 10 and 11, 1912, as described below:

Sections of coal bed in Kay Moor No. 1 mine.

Section	A		B		C		D		E	
Laboratory No	14294		14295		14296		14297		14299	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Cannel coal	..	..	..	..	0	3	..	..	..	..
Coal	0	11	0	11½	0	8	0	11	0	9
Coal, gray	0	4½	0	7½	0	7	0	8	0	7
Coal	1	1	1	6	1	3½	0	10	1	2
"Sulphur" band	Streak.		0	⅛	Streak.		0	½	Streak.	
Coal	1	3	0	8	0	8	0	8½	0	0
Coal, bony	0	1	0	1½	0	1	0	1	0	1
Floor, clay.										
Thickness of bed	3	8½	3	10⅝	3	6½	3	3	3	4
Thickness of coal sampled	3	8½	3	10⅝	3	6½	3	3	3	4

Section A (sample 14294) was measured at the face of room 1, left entry 3, off main entry 3, at 4,700 feet south of the mine mouth.

Section B (sample 14295) was measured at the face of right entry 1½, off left entry 1, off Buffalo entry, 3,300 feet southeast of the mine mouth.

Section C (sample 14296) was measured at the face of left entry 1, off right entry 9, at 3,300 feet west of the mine mouth.

Section D (sample 14297) was measured at the face of west entry 11, 3,600 feet southwest of the mine mouth.

Section E (sample 14299) was measured on the pillar between rooms 3 and 4, off right entry 1, off main entry 7.

A composite sample was made by mixing samples 14294, 14295, 14296, and 14297. The results of an ultimate analysis of this sample are shown under laboratory No. 14298.

Sections of coal bed in Kay Moor No. 2 mine, south of Kay Moor.

Section	A	B	C
Laboratory No.	14401	14402	14403
Roof, shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	3 0	0 9	0 9½
Coal, gray	.. ..	0 3	0 2
Coal	.. ..	1 10	1 3
"Sulphur" band	.. ..	.. ..	Streak.
Coal	.. ..	.. ..	0 9
Coal, bony	.. ..	0 1½	0 1
Floor, shale.			
Thickness of bed	3 0	2 11½	3 ½
Thickness of coal sampled	3 0	2 11½	3 ½

Section A (sample 14401) was measured at the face of left entry 3, off the main entry.

Section B (sample 14402) was measured at the face of room 9, off left entry 2.

Section C (sample 14403) was measured at the face of room 9, off left entry 1, at 150 feet from the entry.

A composite sample was made by mixing samples 14401, 14402, and 14403, for an ultimate analysis, the results of which are shown under laboratory No. 14404.

Notes.—Kay Moor No. 1 and No. 2 mines are opened by slope and the coal is mined by the room-and-pillar system. At the time the mines were visited the coal was undercut by air punchers and electric chain cutters and broken down with FFFF black blasting powder. None of the coal was shipped as run-of-mine. The screening plant had bar screens spaced 4 inches apart through which about 66 per cent of the coal passed, and also smaller bar screens with spaces 1½ inches wide which produced an egg coal. There were three loading tracks with a capacity for 16 empty and 16 loaded cars. There were storage bins with a capacity for 400 tons of screened coal and 600 tons of screenings. The screenings were not washed but were sent to the coking plant. About 10,000 acres of coal land was owned by the company, of which about 500 acres had been worked. The daily output of No. 1 mine was 550 tons, with a maximum day's run of 750 tons. The output of this mine will probably be increased to about 850 tons in the near future. Daily output of No. 2 mine, about 90 tons.

SOUTH NUTTAL. BROWN MINE.

Sample.—Semibituminous coal; New River field; analyses Nos. 10618, 10619, 10620, 10621, 10622, and 10623 (p. 113).

Mine.—Brown; Kanawha-New River district; a drift mine at South Nuttal on the Chesapeake & Ohio Railway.

Coal bed.—Sewell. Carboniferous (Pottsville) age, Sewell formation. Thickness, 3 to 4 feet; dip, 3° west; small faults; roof, blue shale 6 feet thick, strong and smooth; floor, hard smooth clay; cover 120 to 350 feet.

The bed was measured and sampled at five points in the mine by J. J. Rutledge on June 22, 1910, as described below:

Sections of coal bed in Brown mine.

Section	A	B	C	D	E
Laboratory No.	10618	10619	10620	10621	10622
Roof, shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	0 8½	0 10¼	0 10	0 10	.. 10
Coal, hard	0 1½	0 ½	0 1	0 1½	.. 1
Coal	2 0	2 11	2 6	2 6½	2 6¼
Coal, bony	[a]0 8	.. ..	.. ..	.. ..	.. ..
Floor, clay.					
Thickness of bed	3 6	3 9¾	3 5	3 6	3 5¼
Thickness of coal sampled	2 10	3 9¾	3 5	3 6	3 5¼

[a] Not included in sample.

Section A (sample 10618) was measured at the face of room 2 off Hill entry.

Section B (sample 10619) was measured on rib of Lockett heading, between rooms 1 and 2, about 2,000 feet from mouth of entry.

Section C (sample 10620) was measured at the neck of room 1 off right entry 6 off Wheeler entry.

Section D (sample 10621) was measured at the neck of room 4 off left entry 2 off entry 18.

Section E (sample 10622) was measured on the left rib near the face of entry 18.

A composite sample was made by mixing samples 10618 to 10622, inclusive; the results of an ultimate analysis of this sample are given under laboratory No. 10623.

Notes.—The coal is mined by the room-and-pillar and modified panel system; at the time of sampling FFF black blasting powder was used for breaking down the coal after undercutting with electric chain cutters. The coal was screened on 1½-inch bar screens, and marketed as lump and screenings; pickers were employed on the cars when loading; there were two loading tracks with capacity for about 20 cars. Output 550 tons per day.

THAYER. EPHRAIMS CREEK (BUFFALO) MINE.

Sample.—Semibituminous coal; New River field; analysis No. 10629 (p. 114).

Mine.—Ephraims Creek (Buffalo); Kanawha-New River district; a drift mine at Thayer, on the Chesapeake & Ohio Railway.

Coal bed.—Fire Creek. Carboniferous age, Pottsville formation. Average thickness, 42 inches; dip, 3° west; roof, strong dark gray shale about 4 feet thick; floor, hard smooth gray shale; coal bed contains some cannel coal.

The bed was measured and sampled at one point in the mine by J. J. Rutledge, on June 20, 1910, as described below:

Section of coal bed in Ephraims Creek (Buffalo) mine.

Section	A	
Laboratory No.	10629	
Roof, shale.	*Ft.*	*in.*
Coal	1	1
Cannel coal	0	5
Coal	2	0
Floor, clay.		
Thickness of bed	3	6
Thickness of coal sampled	3	6

This section (sample 10629) was measured at the face of room 5 off right entry 5 off the Slater entry.

Notes.—The mine is worked by the room-and-pillar system. At the time of sampling FFF black powder was used for breaking coal after undercutting with electric chain cutters. The coal was screened over 1½-inch bar screens, and the screenings were passed over a ¾-inch screen. Pickers were employed on the car when loading; the capacity of the storage bins was 200 tons. There were two loading tracks. Practically all of the coal produced was from advanced workings. Daily output, 1,100 tons.

GREENBRIER COUNTY.

RICHWOOD. SPRUCE KNOB MINE.

Sample.—Bituminous coal; analysis No. 13547 (p. 114).

Mine.—Spruce Knob; Interior district on north fork of Cherry River, 9 miles southeast of Richwood, on a lumber road.

Coal bed.—Sewell (?). Carboniferous age, Pottsville formation.

The bed was measured and sampled in a room 1,500 feet southwest of the mine mouth, on March 15, 1912, by E. R. Lloyd. The section of the bed at the point of sampling was as follows:

Section of coal bed in Spruce Knob mine.

Laboratory No.	13547	
Roof, shale.	*Ft.*	*in.*
Coal, bony	[a]0	9
Coal, good, soft	4	1
Floor, shale.		
Thickness of bed	4 -	10
Thickness of coal sampled	4	1

[a] Not included in sample.

Notes.—The average thickness of the bed is 4 feet. At the time of sampling the average daily output of the mine was about 150 tons.

McDOWELL COUNTY.

ALGOMA. PINEY MINE.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14154, 14155, 14156, 14157, 14158, 14159, 14160, 14161, and 14162 (pp. 114, 115).

Mine.—Piney; Norfolk & Western district; a drift mine about 1 mile east of Algoma, with railroad connections to the Norfolk & Western Railway at Northfork.

Coal bed.—Pocahontas No. 3. Carboniferous (Pottsville) age, Pocahontas formation. Average thickness, 5 feet 3 inches; roof, 18 inches of "draw slate," above which is 6 inches of black shale with a sandstone cap rock. Floor, smooth hard shale. Material from the floor did not become mixed with the coal in mining, but occasionally some of the roof material fell with the coal.

The bed was measured and sampled at seven points in the mine by G. H. Deike, D. J. Price, and E. B. Sutton, June 22, 1912, as described below:

Sections of coal bed in Piney mine.

Section	A		B		C		D		E		F		G	
Laboratory No.	14154		14155		14156		14158		14159		14160		14161	
Roof, shale and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	9½	1	6½	1	7	2	6	1	7	2	2	2	6
Coal, gray	0	2½	..	..	..	..	..	..	..	..	..	..	..	..
Coal	0	7	..	..	..	..	..	..	..	..	..	..	..	..
Bone	[a]0	3	[a]0	1½	[a]0	3	[a]0	3	[a]0	3	[a]0	3½	[a]0	3
Coal	4	3	3	7	3	8	3	10	3	8	3	10½	3	10
Floor, shale.														
Thickness of bed	6	1	5	3	5	6	6	7	5	6	6	4	6	7
Thickness of coal sampled	5	10	5	1½	5	3	6	4	5	3	6	½	6	4

[a] Not included in sample.

Section A (sample 14154) was measured at the face of the main entry.

Section B (sample 14155) was measured at the face of room 14 off cross entry 10.

Section C (sample 14156) was measured at the face of room 30 off cross entry 8.

Section D (sample 14158) was measured on the pillar of room 15 off cross entry 5.

Section E (sample 14159) was measured on the rib of room 33 off cross entry 7.

Section F (sample 14160) was measured on the pillar of room 15 off cross entry 6.

Section G (sample 14161) was measured on the pillar of room 8 off cross entry 3.

A composite sample was made by mixing face samples 14154, 14155, and 14156. The results of an ultimate analysis of this sample are given under laboratory No. 14157. Another composite sample was made by mixing pillar samples 14158 to 14161, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14162.

Notes.—The Piney mine is opened by drift entries and the coal mined by the room-and-pillar system. At the time the mine was visited the coal was undercut by hand; FFF black powder was used for breaking the coal and a permissible explosive for brushing the roof. The screening plant was equipped with bar screens, 14 feet long, 3-inch spaces; also shaking screens 5 by 10 feet, with ½ and ½-inch openings. About 65 per cent of the coal passed through the first screen. Pickers were employed on the car when loading coal. About 100 tons of screenings were coked per day. Daily output of mine, 1,000 tons. There were two loading tracks, with a capacity for 55 empty and 44 loaded cars. There were storage bins with a capacity for 800 tons. The coal was obtained from both advance and pillar work, and there was an ample acreage of coal for many years mining.

For analyses and descriptions of other samples of coal from this mine, see Bull. 22, Bureau of Mines, 1913, p. 966.

BIG FOUR. CIRRUS NO. 2 MINE.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14218, 14219, 14220, 14221, 14222, and 14223 (p. 115).

Mine.—Cirrus No. 2, Norfolk & Western district; a slope mine, at Big Four post office, on the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 3 and Pocahontas No. 4. Carboniferous (Pottsville) age, Pocahontas formation. Average thickness 5 feet 5 inches; slight dip to south. The roof is a shale 3 to 4 feet thick, above which is a sandstone cap rock. The floor is a hard, rough fire clay. Material from the roof and floor did not become mixed with the coal in mining. Cover, at points of sampling, about 400 feet.

The bed was measured and sampled at five points in the mine by J. J. Rutledge and D. J. Price, on June 29, 1912, as described below:

Sections of coal beds in Cirrus No. 2 mine.

	No. 3 bed.				No. 4 bed.
Section	A	B	C	D	E
Laboratory No.	14218	14219	14220	14221	14223
Roof, shale.	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*
Coal	1 7½	2 ½	1 7½	1 9	1 8½
Coal, bony	[a]0 6½	[a]0 3	[a]0 5½	[a]0 5½	[a]0 3
Coal	.. ..	.. ..	.. ..	.. ..	1 1
Coal, bony	.. ..	.. ..	.. ..	.. ..	[a]0 2
Coal	3 5	3 2	3 7	3 0	1 5
Floor, fire clay.					
Thickness of bed	5 7	5 5½	5 8	5 2½	4 7½
Thickness of coal sampled	5 ½	5 2½	5 2½	4 9	4 2½

[a] Not included in sample.

Section A (sample 14218) was measured at the break-through near the face of the main heading, 2,850 feet south of the slope mouth.

Section B (sample 14219) was measured about 30 feet from the face of the sixth south entry, 1,500 feet southwest of the slope mouth.

Section C (sample 14220) was measured at the face of room 4, off the third south heading, 2,850 feet south of the slope mouth.

Section D (sample 14221) was measured at the face of the sixth east heading, 2,100 feet south of the slope mouth.

Section E (sample 14223) was measured at the face of the second north entry, 250 feet northwest of the slope mouth. This sample was taken in Pocahontas No. 4 bed, which is 4 feet 7½ inches thick.

A composite sample was made by mixing face samples 14218 to 14221, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14222.

Notes.—The mine is opened by a slope and the coal mined by the room-and-pillar system. At the time of sampling the coal was undercut by electric punchers and broken down with FFF black powder. There was no screening plant, all of the coal being shipped as run-of-mine. Pickers were employed on the cars in loading. There were two loading tracks with a capacity for 11 empty cars and 21 loaded cars for pit No. 3 and 9 empty cars and 7 loaded cars for pit No. 4. There was approximately 500 acres of unmined coal tributary to this slope. All of the coal mined was from advance workings. The daily output was about 400 tons.

HUGER. MIDDLE STATES MINE.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14416, 14417, 14418, 14419, 14421, 14422, and 14423 (p. 115).

Mine.—Middle States; Norfolk & Western district; a shaft mine at Huger station (Olmsted post office) 2 miles from Welch, on the main line of the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 4. Carboniferous age, Pottsville formation. Average thickness about 5 feet. Roof, sandstone; floor, shale.

The bed was measured and sampled at seven points in the mine by E. B. Sutton and H. I. Smith, June 27, 1912, as described below:

Sections of coal bed in Middle States (Helena) mine.

Section	A		B		C		D		E		F		G	
Laboratory No.	14416		14417		14418		14419		14421		14422		14423	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	4½	0	1½	0	4½	1	10½	0	4½	0	1½	0	4
Bone	[a]0	1	[a]0	1	[a]0	1	0	½	[a]0	1½	[a]0	½	[a]0	1
Coal	1	11	1	11½	1	10½	2	0	1	8	0	2	1	8
Bone	[a]0	1	[a]0	1	[a]0	1	[a]0	½	0	½	0	½	[a]0	½
Coal	1	8½	1	10	2	1	0	5	1	4	1	9	2	2
Bone	[a]0	1	[a]0	3	[a]0	2½	[a]0	1	..	..	[a]0	½	[a]0	2
Coal, gray	..	..	..	..	..	..	..	..	0	2	..	..	..	..
Coal	0	10½	1	1	1	1	1	1	0	11	1	8	1	2
Coal, bony	..	..	..	..	..	..	..	..	..	..	0	1½	..	..
Coal	..	..	..	..	..	..	..	..	..	..	0	1½	..	..
Floor; shale.														
Thickness of bed	5	1½	5	5	5	9½	5	6½	4	7½	4	1	5	7½
Thickness of coal sampled	4	10½	5	0	5	5	5	5	4	6	3	11½	5	4

[a] Not included in sample.

Section A (sample 14416) was measured at the face of room 6 off the second rock heading.

Section B (sample 14417) was measured at the face of the second right air course.

Section C (sample 14418) was measured on the rib of the last break-through on the main diagonal entry.

Section D (sample 14419) was measured at the face of room 4 off first west parallel heading.

Section E (sample 14421) was measured on the pillar between rooms 3 and 6 off the north entry.

Section F (sample 14422) was measured on the pillar between rooms 8 and 9, left entry 4 off the main west heading.

Section G (sample 14423) was measured on the pillar between rooms 20 and 21, off second parallel entry off the main diagonal heading.

Two composite samples were made: (1) By mixing samples 14416 to 14419, inclusive, for an ultimate analysis, the results of which are shown under laboratory No. 14420; (2) by mixing samples 14421 to 14423, inclusive, for an ultimate analysis, the results of which are shown under laboratory No. 14424.

Notes.—The Middle States mine is opened by shaft 182 feet deep, and the coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by electric machines and shot down with permissible explosives. Dynamite was used for brushing the roof. All of the coal was shipped as run-of-mine. Pickers were employed on the car when loading. There were 2 loading tracks with a capacity for 70 empty and 30 loaded cars. Daily output, 1,000 tons. There was sufficient unmined coal to insure many years operation.

JED. HAVACO MINE.

Sample.—Semibituminous coking coal; Pocahontas field; analysis No. 13775 (p. 116).

Mine.—Havaco (formerly Jed); Norfolk & Western district; a shaft mine at Jed, on the Tug Fork branch of the Norfolk & Western Railway.

Coal bed.—Known as Pocahontas No. 3, Carboniferous (Pottsville) age, Pocahontas formation. The thickness is uniform and ranges from 4 feet 8 inches to 5 feet 4 inches, with an average of 5 feet. Dip, about 1½° to 2½° northwest. Roof, strong bedded sandstone underlain in parts with 12 to 15 inches of treacherous shale, which did not regularly fall with the coal, but was brushed in the entries. Floor, 12 to 18 inches of hard clay with smooth surface. Cover, about 400 feet.

The coal bed was measured and sampled at one point in the mine by J. W. Paul, April 5, 1912, as described below:

Sections of coal bed in Havaco (formerly Jed) mine.

Laboratory No.	13775	
Roof, dark shale.	*Ft.*	*in.*
Coal	0	5½
Mother coal	0	¼
Coal	0	6
Mother coal	0	½
Coal	0	10
Bone	[a]0	4
Coal	2	11
Floor, hard clay.		
Thickness of bed	5	1¼
Thickness of coal sampled	4	9¼

[a] Not included in sample.

Sample 13775 was measured at the face of first butt left entry.

Notes.—At the time of sampling the coal was undercut with chain machines in the bottom part of the seam and shot down with a permissible explosive. The tipple had no storage bins. The entire output was shipped as run-of-mine. There were three loading tracks with a capacity for 75 railroad cars.

For analyses and descriptions of other samples of coal from this mine, see Bureau of Mines Bull. 22, 1914, pp. 261, 993, 994.

JENKINJONES. POCAHONTAS NOS. 6, 6B, AND 7A MINES.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14185, 14186, 14187, 14188, 14189, and 14190 (p. 116.)

Mine.—Pocahontas Nos. 6, 6B, and 7A; Norfolk & Western district; drift mines; No. 6 mine is at Jenkinjones, on the Tug Fork branch ot the Norfolk & Western Railway, and No. 6B and No. 7A are one-half mile south of Jenkinjones.

Coal bed.—Pocahontas No. 3. Carboniferous (Pottsville) age, Pocahontas formation. Average thickness, 10 feet. Roof, dark shale, which is smooth and does not mix with the coal when mining. Floor, smooth hard shale, which does not adhere to the coal. Cover, approximately 300 feet.

The bed was measured and sampled at five points in the mine by J. J. Rutledge, G. H. Deike, and D. J. Price on June 26, 1912, as described below:

Sections of coal bed in Pocahontas Nos. 6, 6B, and 7 A mines.

Section	A		B		C		D		E	
Laboratory No	14185		14186		14187		14189		14190	
Roof, dark shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	6	2	8	2	10	3	3	3	0
Gray coal to sulphur	0	½	0	½	..	..	0	½	..	..
Coal	1	6	0	5	..	..	..	..	..	..
Mother coal	0	1	..	..	..	..	..	..	..	..
Coal, bony	..	..	[a]0	4½	[a]0	4½	..	..	[a]0	3½
Coal	0	9	0	5½	0	4	0	9	0	8½
Coal, bony	[a]0	3½	[a]0	2	[a]0	2½	0	1½	..	..
Coal	0	4	0	6	0	8	0	1½	..	..
Coal, gray	0	1	..	..	..	..	0	½	..	..
Coal, bony	..	..	[a]0	2½	[a]0	6	..	..	..	..
Coal	1	10	2	3½	1	5	0	2	..	..
Coal, bony	[a]0	3	[a]0	3	[a]0	½	[a]0	6	0	5
Coal	2	10	2	10	4	2	4	4	[a]3	5
Floor, light shale.										
Thickness of bed	9	6	10	2½	10	6½	9	4	7	10
Thickness of coal sampled	8	11½	9	2½	9	5	8	10	7	1½

[a] Not included in sample.

Section A (sample 14185) was measured at the face of the main heading of No. 6 mine, 250 feet west of the mine mouth.

Section B (sample 14186) was measured at the face of the aircourse of No. 6 mine, 250 feet from the mine mouth.

Section C (sample 14187) was measured at the face of the main heading of 6-B mine, 500 feet from the mine mouth.

Section D (sample 14189) was measured at the face of the first right cross heading of No. 6-B mine, 350 feet from the mine mouth.

Section E (sample 14190) was measured at the face of the main heading of Pocahontas No. 7-A mine, 120 feet from the mine mouth.

A composite sample was made by mixing face samples 14185, 14186, and 14187. An ultimate analysis of this sample is given under laboratory No. 14188.

Notes.—Both No. 6 and No. 7A mines are new and are opened by drift entries. The room-and-pillar system of mining is used. At the time of sampling the coal was undercut by electric machines and shot down with FFF black powder. The tipple was a temporary structure, the permanent plant being in course of construction. No. 6 mine had a daily output of 75 tons; No. 6B mine of 400 tons, and No. 7A was not shipping.

LECKIE. LECKIE NO. 1 AND NO. 2 MINES.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14271, 14272, 14273, 14274, 14275, 14276, 14277, 14278, 14279, 14280, 14281, and 14282 (pp. 116–117).

Mine.—Leckie No. 1 and No. 2; Norfolk & Western district; drift mines one-half mile east of Leckie on the Tug Fork branch of the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 3. Carboniferous (Pottsville) age, Pocahontas formation. Average thickness, 7 feet 6 inches; dip, about 1½ per cent northwest. Roof, dark shale about 9 inches thick, above which is a strong cap rock; floor, hard, smooth clay. The material from the floor did not become mixed with the coal when mining, but a small amount of the shale from the roof fell with the coal. Cover, 150 to 200 feet.

The bed was measured and sampled at five points in the No. 1 mine by G. H. Deike on July 6, 1912, as described below:

Sections of coal bed in Leckie No. 1 mine.

Section	A		B		C		D		E	
Laboratory No.	14271		14272		14273		14274		14275	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	10	0	10	0	10	0	11	0	10
"Sulphur" band	[a] 0	¾	[a] 0	1	[a] 0	1	[a] 0	1	[a] 0	1½
Coal	1	10½	1	10	1	½	1	3½	1	5
Coal, gray	..	..	..	..	0	6	0	1	..	..
Mother coal	..	..	..	..	..	..	..	..	0	1
Coal	..	..	..	..	0	5	0	1	0	4
Coal, bony	[a] 0	3½	[a] 0	6	[a] 0	2	[a] 0	3½	[a] 0	4
Coal	1	6	2	2	..	..	1	11	1	5
Coal, gray	0	½	0	½	1	7	..	..	..	..
Coal	..	..	0	8	0	11	..	..	..	..
Coal, gray	..	..	0	2	0	4	0	1	..	..
Coal	3	7	1	0	..	..	..	..	..	..
Mother coal	..	..	0	½	..	..	..	..	0	1
Coal	..	..	0	9½	2	0	2	6	2	9
Floor, clay.										
Thickness of bed	8	2½	8	1½	7	10½	7	3	7	4½
Thickness of coal sampled	7	10	7	6½	7	7½	6	10½	6	11½

[a] Not included in sample.

Section A (sample 14271) was measured at the face of room 1, off Monkey heading, 1,700 feet west of the mine mouth.

Section B (sample 14272) was measured at the face of Monkey heading, 1,900 feet west of the mine mouth.

Section C (sample 14273) was measured at the face of Kick-back heading, 1,250 feet northwest of the mine mouth.

Section D (sample 14274) was measured on a chain pillar of first right heading, 650 feet north of the mine mouth.

Section E (sample 14275) was measured on the pillar at the mouth of room 1, off second right heading, 950 feet northwest of the mine mouth.

A composite sample was made by mixing face samples 14271 to 14275, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14276.

The bed was measured and sampled at five points in the No. 2 mine by H. I. Smith on July 6, 1912, as described below:

Sections of coal bed in Leckie No. 2 mine.

Section	A		B		C		D		E	
Laboratory No.	14277		14278		14279		14280		14281	
Roof, "draw slate."	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Blocky coal	..	..	0	1	0	1	0	1	0	1
Coal	1	1	0	9	0	10	0	9	0	9½
"Sulphur" band	[a] 0	1	[b] 0	1	[b] 0	1	[b] 0	1	[b] 0	½
Coal	1	8	..	..	1	9	2	1	1	1
Coal, gray	..	..	0	3	..	..	..	..	0	4
Coal	..	..	1	6	..	..	..	..	0	6
Bone	[b] 0	5	[b] 0	3	[b] 0	5	[b] 0	4	[b] 0	6
Coal	4	6	1	6	1	8	1	8	1	10
Coal, gray	..	..	1	3	0	4	0	3	0	4
Coal	..	..	..	..	0	4	0	8	..	..
Coal, gray	..	..	..	..	..	..	0	4	..	..
Mother coal	..	..	..	..	0	1	..	..	..	..
Coal	..	..	1	10	2	5	1	8	2	10
Floor, clay.										
Thickness of bed	7	9	7	6	8	0	7	11	8	4
Thickness of coal sampled	7	4	7	2	7	6	7	6	7	9½

[a] Too brittle to exclude. [b] Not included in sample..

Section A (sample 14277) was measured at the face of room 17 off the first right air course, 100 feet from the outcrop.

Section B (sample 14278) was measured at the working face.

Section C (sample 14279) was measured at the face of the main straight entry, 600 feet southeast of the mine mouth.

Section D (sample 14280) was measured at the face of room 5 off the diagonal entry.

Section E (sample 14281) was measured at the face of the first left entry, 1,300 feet from the mine mouth.

A composite sample was made from face samples 14277 to 14281, inclusive. The result of an ultimate analysis of this sample are given under laboratory No. 14282.

Notes.—The Leckie No. 1 and No. 2 mines are opened by drift entries and the coal mined by the room-and-pillar system. At the time the mines were visited electric chain machines were used in No. 1 mine for undercutting the coal and permissible explosives for breaking the coal. Daily output, 450 tons. About 30 per cent of the coal was obtained from advance workings. There was approximately 25 acres of unmined coal tributary to this entry. Coal-cutting machines were not used in No. 2 mine. Permissible explosives were used for breaking the coal. The capacity of No. 2 mine was 450 tons per day. A large part of the coal was obtained from advance workings. There was approximately 35 acres of coal tributary to this mine. All the coal from both mines was shipped as run-of-mine. Pickers were employed on the cars in loading. There was one loading track with a capacity for 20 empty and 30 loaded cars.

RODERFIELD. DAVY-POCAHONTAS NO. 1 AND NO. 2 MINES.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14262, 14263, 14264, 14266, 14267, and 14268 (p. 117).

Mine.—Davy-Pocahontas No. 1 and No. 2; Norfolk & Western district; drift mines. No. 1 mine is 1¾ miles and No. 2 mine 2 miles east of Roderfield on the Spice Creek branch of the Norfolk & Western Railway.

Coal bed.—No. 1 mine is in the War Creek bed. Carboniferous age, Pottsville formation. Average thickness of bed, 6 feet. The roof is sandstone, to which a small amount of coal adheres. Floor, sandstone. No. 2 mine is in the Davy bed, Carboniferous age, Pottsville formation. Average thickness of bed, 3 feet 6 inches. Roof, smooth sandstone; floor, black carbonaceous shale which is hard and smooth. Material from the floor occasionally becomes mixed with the coal in mining. Cover at No. 2 mine, 200 to 300 feet.

The coal bed in No. 1 mine was measured and sampled at three points in the mine by H. I. Smith on July 5, 1912, and the bed in No. 2 mine was measured and sampled at three points by G. H. Deike on July 5, 1912, as described below:

Sections of coal bed in Davy-Pocahontas No. 1 mine.

Section	A		B		C	
Laboratory No.	14266		14267		14268	
	Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Roof, sandstone.						
Coal	0	9	1	1	0	3½
"Sulphur" band	..	..	..	..	[a]0	¾
Coal	..	..	..	..	0	4
Coal, gray	0	5	0	5	0	4
Coal	0	8	0	6½	0	1
Mother coal and "sulphur"	0	½	0	½	0	¼
Coal	1	6½	1	9	1	2
Mother coal and "sulphur"	..	..	..	..	0	¼
Coal	..	..	..	..	0	7¾
Coal, bony	[a]0	5	[a]0	7	[a]0	5
Coal, very soft	0	1	..	..	..	..
Coal	1	11	2	3½	2	0
Coal, bony	[a]0	3	[a]0	2	[a]0	4
Floor, sandy shale.						
Thickness of bed	6	1	6	10½	5	8½
Thickness of coal sampled	5	5	6	1½	4	10¾

[a] Not included in sample.

Section A (sample 14266) was measured at the face of room 3, off entry 2.

Section B (sample 14267) was measured at the face of entry 2, opposite room 9

Section C (sample 14268) was measured at the face of entry 7, at the mouth of room 15.

A composite sample was made by mixing face samples 14266, 14267, and 14268. The results of an ultimate analysis of this sample are given under laboratory No. 14269.

Sections of coal bed in Davy-Pocahontas No. 2 mine.

Section	A		B		C	
Laboratory No	14262		14263		14264	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	..	..	0	3	..	..
Mother coal	..	..	0	½	..	..
Coal	2	10½	0	11	1	1
Coal, gray	..	..	..	..	0	1
Mother coal	..	..	0	¾	..	..
Coal	..	..	0	3	0	4
Coal, gray	..	..	0	1	..	..
"Sulphur" band	..	..	..	..	[a] 0	⅜
Coal	..	..	0	7	1	2½
Shale, hard	[a] 0	2	[a] 0	1½	[a] 0	2
Coal	0	2	0	2	0	2
Shale, soft, block	[a] 0	6	[a] 0	6	[a] 0	6
Floor, sandy shale.						
Thickness of bed	3	8½	2	11¾	3	6⅞
Thickness of coal sampled	3	½	2	4¼	2	10½

[a] Not included in sample.

Section A (sample 14262) was measured at the face of room 5, left entry 1, 500 feet southwest of the mine mouth.

Section B (sample 14263) was measured at the face of the first left-air course, 600 feet southwest of the mine mouth.

Section C (sample 14264) was measured at the face of the main heading, 600 feet west of the mine mouth.

A composite sample was made by mixing face samples 14262, 14263, and 14264. The results of an ultimate analysis of this sample are given under laboratory No. 14265.

Notes.—No. 1 mine is opened by a drift entry. The coal is mined by irregular tunneling intended for the room-and-pillar system. The mine is not well laid out. At the time of sampling coal cutters were not used. No. 2 mine was new; the main entry was about 600 feet long. At the time of sampling the work was largely development. Electric chain coal cutters were used for undercutting the coal. Black powder was used for breaking the coal in both mines and also for taking up the floor. All of the coal from No. 1 mine was shipped as run-of-mine. The coal from No. 2 mine was passed over bar screens with spaces 1¼ inches wide. Sixty per cent of the coal passed through this screen. Pickers were employed on the cars in loading. The coal on the cars was very clean; the lumps were large and had a good appearance. There were two loading tracks with a capacity for 20 empty and 20 loaded cars. Daily output of No. 1 mine, 150 tons; of No. 2 mine, 80 tons. There was approximately 1,000 acres of unmined coal tributary to these mines.

ROLFE. ROLFE MINE.

Sample.—Semibituminous coal, Pocahontas field; analyses Nos. 14394, 14395, 14396, 14397, 14398, and 14400 (p. 117).

Mine.—Rolfe; Norfolk & Western district; a drift mine at Rolfe on the North Fork branch of the Norfolk & Western Railway, 3 miles from North Fork.

Coal bed.—Pocahontas No. 3. Carboniferous age, Pottsville formation. Average thickness, about 5 feet 2 inches. The coal bed is practically horizontal. The roof consists of a "draw-slate" 2 inches thick, which falls with the coal; above the "draw-slate" is a hard smooth shale. The floor is a hard smooth shale which does not become mixed with the coal in mining. There is a streak of bony coal about 1 foot from the top of the bed, which is rather persistent throughout the mine. This is excluded in the sections given below.

The coal bed was measured and sampled at six points in the mine by E. B. Sutton and H. I. Smith, June 25, 1912, as described below:

Sections of coal bed in Rolfe mine.

Section	A		B		C		D		E		F	
Laboratory No.	14394		14395		14396		14397		14398		14400	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	3	1	4	1	4	1	2	1	1½	1	2
Bone	[a]0	4	[a]0	3	[a]0	4	[a]0	2	[a]0	2½	[a]0	2
Coal	1	1	1	1	1	1	1	1	1	2	1	0
Coal, gray	0	4	0	4	0	4	..	..	..	..	..	..
Coal, gray, bony	..	..	..	..	..	..	[a]0	4	[a]0	4	[a]0	4
Coal	2	3	2	1	2	1	2	3¼	2	3½	2	2
Floor, shale.												
Thickness of bed	5	3	5	1	5	2	5	¼	5	1½	4	10
Thickness of coal sampled	4	11	4	10	4	10	4	6¼	4	7	4	4

[a] Not included in sample.

Section A (sample 14394) was measured at the face of B-2 butt entry off the B entry, 5,800 feet northwest of the mine mouth.

Section B (sample 14395) was measured at the face of B-1 butt entry off the B entry, 6,250 feet northwest of the mine mouth.

Section C (sample 14396) was measured at the face of the left boundary heading, 6,200 feet northwest of the mine mouth.

Section D (sample 14397) was measured at the face of room 15 off 3-A entry off right entry 1, 5,100 feet northeast of the mine mouth.

Section E (sample 14398) was measured at the face of room 18 off 2-A entry off right entry 2, 5,000 feet northeast of the mine mouth.

Section F (sample 14400) was measured on the pillar of room 4 off right entry 1, 4,200 feet east of the drift mouth.

A composite sample was made by mixing samples 14394 to 14398, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14399.

Notes.—The Rolfe mine is opened by drift entries and the coal mined by the room-and-pillar system. At the time the mine was visited the coal was undercut by electric chain cutters and broken down with FFFF black powder. FFF black powder was used for brushing the roof. All of the coal was shipped as run-of-mine and pickers were employed on the cars in loading. The lumps were fairly large, ranging from 6 to 12 inch cubes. There were two loading tracks with a capacity for 22 empty and 45 loaded cars. There were no storage bins. Daily output, 700 tons with a maximum day's run of 875 tons. All of the coal was from advance workings.

SUSANA. YUKON-POCAHONTAS.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14425, 14426, 14427, 14428, and 14429 (p. 118).

Mine.—Yukon-Pocahontas, Norfolk & Western district; a drift mine one-fourth mile west of Susana station (Yukon post office) on the Norfolk & Western Railway.

Coal bed.—War Creek. Carboniferous age, Pottsville formation. Average thickness, about 5 feet 3 inches. Dip, 1½ per cent to the northwest. Roof, shale; floor, bony coal 15 inches to 30 inches thick, underlain with sandstone.

The coal bed was measured and sampled at five points in the mine by H. I. Smith and E. B. Sutton, June 28, 1912, as described below:

Sections of coal bed in Yukon-Pocahontas mine.

Section	A		B		C		D		E	
Laboratory No	14425		14426		14427		14428		14429	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal and mother coal	[a]0	1½	..	..	..	..	[a]0	2	[a]0	3½
Coal	..	..	0	1	..	..	..	..	..	..
Bone	..	..	[a]0	½	..	..	..	..	..	..
Coal	0	11½	0	9	1	1	..	..	2	2
"Sulphur" band	Streak.		Streak.		Streak.		..	..	0	½
Coal	3	4	1	10½	2	7	4	1	1	9
Bone	[a]1	4	[a]1	3½	[a]1	8½	[a]0	10	[a]1	0
Floor, sandstone.										
Thickness of bed	5	9	4	½	5	4½	5	1	5	3
Thickness of coal sampled	4	3½	2	8½	3	8	4	1	3	11½

[a] Not included in sample.

Section A (sample 14425) was measured on a pillar of the third right cross-cut off the main entry, at the last break-through, 1,200 feet northwest of the old opening.

Section B (sample 14426) was measured at the face of room 24 off the main air course, 1,700 feet southwest of the main opening.

Section C (sample 14427) was measured at the face of the main entry, 1,850 feet southwest of the main opening.

Section D (sample 14428) was measured at the face of room 7 off the main air course, 850 feet southwest of the main opening.

Section E (sample 14429) was measured at the face of room 14 off the main air course, 1,100 feet southwest of the main opening.

A composite sample was made by mixing face samples 14426 to 14429, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14430.

Notes.—The Yukon-Pocahontas mine is opened by drift entries and the coal mined by the room-and-pillar system. At the time the mine was visited all of the coal was mined by hand. FFF black blasting powder was used for breaking the coal. None of the coal was shipped as run-of-mine. The screening plant was equipped with screens for producing 4-inch, 1½-inch, and ⅝-inch coal. Pickers were employed on the car in loading and also at a traveling belt. The coal lumps were large and had a good appearance. There were four loading tracks with a capacity for 40 empty and 35 loaded cars. The mine had a daily output of 400 tons, practically all of which was from advance workings. The maximum capacity of the mine was about 575 tons per day. There was about 4,000 acres of unmined coal tributary to this mine.

MAYBEURY. NORFOLK MINE.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 11247 and 11780 (p. 119).

Mine.—Norfolk, a drift mine 2 miles from Maybeury, on the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 3. Carboniferous (Pennsylvanian) age, Pocahontas formation. The bed is about 8½ feet thick and lies nearly horizontal.

Two car samples of coal from this mine were taken by E. R. Linkenhoker as the cars were being unloaded at the Bureau of Mines testing station at Pittsburgh, on November 28, 1910, and March 2, 1911, respectively.

Samples 11247 and 11780 each represent a car load of ¾-inch to 1½-inch nut coal.

For analyses and descriptions of other samples of coal from this mine, see Bureau of Mines Bull. 22, 1913, pp. 265, 1001.

WAR. JOHN'S BRANCH MINE.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14468, 14469, 14470, 14471, and 14472 (p. 119).

Mine.—John's Branch, Norfolk & Western district; a drift mine 1 mile south of War on the Dry Fork branch of the Norfolk & Western Railway.

Coal bed.—War Creek. Carboniferous age, Pottsville formation. Average thickness, 3 feet 10 inches. The coal bed is practically horizontal. Roof, 4 inches of "draw slate," which parts easily from the coal. Above the "draw slate" is a main cap rock. Floor, sandstone.

The coal bed was measured and sampled at four points in the mine by H. I. Smith and E. B. Sutton, June 29, 1912, as described below:

Sections of coal bed in John's Branch mine.

Section	A		B		C		D	
Laboratory No.	14468		14469		14470		14471	
Roof, "draw slate."	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	3½	0	3¼	..	..	..	..
Coal, gray	0	1	..	..	..	..	..	..
Coal, bony	..	..	0	¾	..	..	..	..
Coal	1	6	3	6	3	8	3	7
Mother coal and "sulphur"	0	½	..	..	..	..	..	..
Coal	0	10	..	..	..	..	..	..
Coal, bony	[a] 0	7	..	..	[a] 0	8	[a] 0	9
Floor, shale, coal, and sandstone.								
Thickness of bed	3	4	3	10	4	4	4	4
Thickness of coal sampled	2	9	3	10	3	8	3	7

[a] Not included in sample.

Section A (sample 14468) was measured at the face of the main entry, 800 feet from the mine mouth.

Section B (sample 14469) was measured at the face of heading 3 off the right entry.

Section C (sample 14470) was measured at the face of room 4 off the first left air course.

Section D (sample 14471) was measured at the face of the first right butt entry off left entry 2.

A composite sample was made by mixing face samples 14468 to 14471, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14472.

Notes.—The John's Branch mine is opened by drift entries and the coal mined by the room-and-pillar system. At the time of sampling all the mining was done by hand and the coal broken with FFF black powder. All of the coal was shipped as run-of-mine. There was no screening plant. Pickers were employed on the car in loading. The lumps of coal were large and had a good appearance. The loading tracks had a capacity of 15 empty and 15 loaded cars. Daily output, 125 tons.

WAR. WAR CREEK MINE.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14405, 14406, 14407, and 14408 (p. 119).

Mine.—War Creek, Norfolk & Western district; a drift mine at War, on the Dry Fork branch of the Norfolk & Western Railway.

Coal bed.—War Creek. Carboniferous age, Pottsville formation. Average thickness, 4 feet. Roof, smooth shale, 10 feet thick, which does not fall when the coal is mined. Floor, soft sandstone.

The coal bed was measured and sampled at three points in the mine by H. I. Smith and E. B. Sutton on June 30, 1912, as described below:

Sections of coal bed in War Creek mine.

Section	A		B		C	
Laboratory No.	14405		14406		14407	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	0	3½	0	2½	..	..
Coal	..	..	0	8	..	..
Mother coal and "sulphur"	..	..	0	½	..	..
Coal	1	9½	1	1	..	..
Mother coal and "sulphur"	0	½	0	½	..	..
Coal	1	1	1	3	3	0
Coal, bony	0	1½	0	7½	..	..
Floor, sandstone.						
Thickness of bed	3	4	3	11	3	0
Thickness of coal sampled	3	4	3	11	3	0

Section A (sample 14405) was measured at the face of the fourth left cross entry off the main entry.

Section B (sample 14406) was measured at the face of the main entry, 1,200 feet from the mine mouth.

Section C (sample 14407) was measured at the face of room 4 off the main air course, about 1,000 feet from the mine mouth.

A composite sample was made by mixing face samples 14405, 14406, and 14407. The results of an ultimate analysis of this sample are given under laboratory No. 14408.

Notes.—The War Creek mine is opened by drift entries and the coal mined by the room-and-pillar system. At the time of sampling the coal was undercut by hand and in some places was cut at the top of the bed. FFF black blasting powder was used both for breaking the coal and for brushing the roof. All of the coal was shipped as run-of-mine. Pickers were employed on the car in loading. The coal had a good appearance when loaded, the lumps being fairly large. There was one loading track with a capacity for a number of cars. Daily output, 50 tons. There was about 1,000 acres of land tributary to this mine. The War Creek mine is comparatively new.

WELCH. OREGON NO. 2 AND NO. 3 MINES.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14212, 14213, 14214, 14215, 14216, and 14217 (p. 120).

Mine.—Oregon No. 2 and No. 3; Norfolk & Western district; drift mines, 2 miles west of Welch, Hemphill Township, with railroad connections to the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 4. Carboniferous age, Pottsville formation. Average thickness, 3 feet 6 inches; the bed is practically horizontal. The roof is a dark, hard, smooth, sandy shale, and does not fall immediately after the coal is taken out. Floor, hard, smooth sandstone. Material from the roof and from the floor does not become mixed with the coal when mined. Cover, 150 to 450 feet.

The coal bed was measured and sampled at five points in the mines by G. H. Deike on June 30, 1912, as described below:

Sections of coal bed in Oregon No. 2 and No. 3 mines.

Section	A		B		C		D		E	
Laboratory No.	14212		14213		14214		14215		14216	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	11	1	0	1	0	0	10	1	0
Gray coal, hard	0	4	0	4	0	4	0	4	0	4
Coal	2	4	2	1	2	9	1	11	2	0
Floor, sandstone.										
Thickness of bed	3	7	3	5	4	1	3	1	3	4
Thickness of coal sampled	3	7	3	5	4	1	3	1	3	4

Section A (sample 14212) was measured at the face of main entry, 3,000 feet southwest of mouth of No. 3 mine.

Section B (sample 14213) was measured at the face of room 16 off fourth left entry, 2,000 feet south of mouth of No. 3 mine.

Section C (sample 14214) was measured on the pillar of room 1, off first left entry, 800 feet southeast of mouth of No. 3 mine.

Section D (sample 14215) was measured at the face of the main entry, 4,000 feet south of mouth of No. 2 mine.

Section E (sample 14216) was measured on the rib of room 11, off the air course, 100 feet from the face and 3,500 feet southeast of mouth of No. 2 mine.

A composite sample was made by mixing samples 14212 to 14216, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14217.

Notes.—The Oregon mines are opened by drift and worked by the room-and-pillar system of mining. At the time of sampling, the coal was undercut by compressed air punchers and shot down with FFF black powder. Permissible explosives were used for brushing the roof. The tipple was equipped with a shaking screen 5 by 24 feet with 3-inch holes. Fifty per cent of the coal passed through the screens. About 50 per cent of the coal was shipped as run-of-mine. The coal had a good appearance when loaded on the cars. There were two loading tracks with a capacity of 25 empty and 20 loaded cars. There was about 1,800 acres of unmined land tributary to this mine. Forty per cent of the coal was taken out in advance workings. The capacity of the mines was 450 tons a day with a maximum day's run of about 800 tons.

WELCH. STANDARD POCAHONTAS MINE.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 12512 and 12513 (p. 120).

Mine.—Standard Pocahontas; Norfolk & Western district; the mine is opened by a shaft 370 feet deep and is 2 miles west of Welch on the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 3 bed. Carboniferous age, Pocahontas formation. Thickness, 4 feet 7 inches to 6 feet. There is a characteristic bone parting 2 to 5 inches thick near the middle of the bed. Roof, "draw slate," overlain by 3 feet of black shale. Particles of the roof did not mix with the coal in loading. Floor, hard, smooth, sandy clay. Cover, about 375 feet.

The coal bed was measured and sampled at two points in the mine by J. T. Ryan on August 8, 1911, as described below:

Sections of coal bed in Standard Pocahontas mine.

Section	A		B	
Laboratory No.	12512		12513	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Coal laminated with mother coal	0	6½	0	3¼
"Sulphur"	0	¼	..	..
Coal	0	9½	1	1¼
Bone	[a]0	4	[a]0	2½
Coal	0	3½	..	..
Mother coal	0	1	..	..
Coal	2	10	3	4
Shale	0	¼	..	..
Coal	0	1½	..	..
Floor, sandy clay.				
Thickness of bed	5	½	4	11
Thickness of coal sampled	4	8½	4	8½

[a] Not included in sample.

Section A (sample 12512) was measured at the face of the main entry in No. 2 shaft, 85 feet from the bottom of the shaft.

Section B (sample 12513) was measured at the face of entry driven from No. 1 shaft to connect with No. 2 shaft, about 120 feet from the bottom of No. 1 shaft.

Notes.—At the time of sampling the coal was undercut by puncher machines and blasted down with permissible explosives. The machine cuttings were loaded out with the coal. The entire future output of the mine will be from advanced work as this is a new development.

MARION COUNTY.

MONONGAH. MINE NO. 63.

Sample.—Bituminous coal; analyses Nos. 11429, 11430, 11431, and 11432 (p. 121).

Mine.—No. 63 (formerly No. 8) Monongahela district; a slope mine at Monongah, on the main line of the Baltimore and Ohio Railroad.

Coal bed.—Pittsburgh. Carboniferous age, Monongahela formation; average thickness 7 feet; dip about 3° northwest; roof, dark shale 2 to 8 feet thick with sandstone cap rock; floor, fire clay; cover at points of sampling, 200 to 400 feet.

The bed was measured and sampled at four points in the mine by J. J. Rutledge on December 19, 1910, as described below:

Sections of coal bed in No. 63 mine, at Monongah.

Section	A		B		C		D	
Laboratory No.	11429		11430		11431		11432	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	0	1	11	1	7	1	10½
"Sulphur" band	..	..	0	½	ª0	½	0	¾
Coal	..	..	0	4½	..	..	0	4½
Coal, bony	ª0	¾	ª0	1	..	..	ª0	1
Coal	0	3¼	0	4	..	..	0	4
Coal, bony	ª0	¾	ª0	½	..	..	0	½
Coal	0	3	0	4	1	5	0	4
Coal, bony	ª0	1	ª0	1	ª0	1½	ª0	1
Coal	2	3	2	10	3	10	2	10
"Sulphur" band	ª0	½	ª0	½	..	..	0	¾
Coal	0	6	..	..	..	..	1	5¼
"Sulphur" band	0	½	..	..	..	..	..	..
Coal	1	6	1	7½	..	..	..	..
Floor, clay.								
Thickness of bed	7	¾	7	7½	7	0	7	5½
Thickness of coal sampled	6	9¼	7	5	6	10	7	8½

ª Not included in sample.

Section A (sample 11429) was measured at the face of room 14, off left entry 3 off north entry 2, about 2,700 feet northwest of the mouth of the main slope.

Section B (sample 11430) was measured at the face of room 14 off left entry 4 off north entry 2, about 3,200 feet northwest of the mouth of the main slope.

Section C (sample 11431) was measured at the face of room 18 off left entry 2 off north entry 2, about 2,650 feet from the mouth of the main slope.

Section D (sample 11432) was measured at the face of room 30 off left entry 1, off north entry 2, about 2,900 feet northwest from the mouth of the main slope.

A composite sample was made by mixing face samples 11429 to 11432, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 11433.

Notes.—Coal hard and blocky and used largely for firing locomotives, by-products and lake shipments; some coke was made from the screenings at the mine. There were three loading tracks with capacity for 60 to 75 cars. The coal was mined by the room-and-pillar system with modified panel system; permissible explosives were used exclusively; the coal was undercut with electric chain machines; daily output, 800 to 1,000 tons.

MONTANA STATION. PARKER RUN MINE.

Sample.—Bituminous coal; analyses Nos. 15067, 15068, 15069, 15070, 15071, and 15072 (p. 121).

Mine.—Parker Run; Monongahela district; a drift mine one-fourth of a mile north of Montana station on the Baltimore & Ohio Railroad.

Coal bed.—Known as the Sewickley. Carboniferous age, Monongahela formation. Thickness, 5 feet 6 inches to 6 feet; average, 5 feet 9 inches. The character of the seam is fairly regular, though there are some sulphur balls and occasional clay binders. Roof, smooth sandstone; floor, soft fire clay; particles of the roof and floor sometimes become mixed with the coal in loading.

The bed was measured and sampled at five points in the mine by G. H. Deike and Edwin Higgins on November 9, 1912, as described below:

Sections of coal bed in Parker Run mine.

Section	A		B		C		D		E	
Laboratory No	15067		15068		15069		15070		15071	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Black shale	[a]0	1	..	..	..	..	..	..	..	..
Coal, streaked, bony	[a]0	1½	..	..	..	..	..	..	[a]0	½
Coal, hard, bright	1	0	2	1	3	10½	3	5	3	9½
"Sulphur" streak	[a]0	1½	[a]0	1	[a]0	1	[a]0	2	..	..
Coal, hard, bright	2	5	1	8	2	½	1	9	..	..
Coal streaked with bone	..	..	[a]0	½	..	..	..	..	[a]0	2½
"Sulphur" streak	[a]0	1	..	..	..	..	..	..	..	..
Coal, hard, bright	2	0	2	2½	..	..	..	..	2	0
Coal, dirty, streaked	..	..	..	..	..	..	..	..	0	2
Bony coal and black 'rash"	[a]0	1½	[a]0	1	[a]0	1	[a]0	1	[a]0	1
Floor, fire clay.										
Thickness of bed	5	11½	6	2	6	1	5	5	6	3½
Thickness of coal sampled	5	5	5	11½	5	11	5	2	5	11½

[a] Not included in sample.

Section A (sample 15067) was cut from a pillar on the main entry at the mouth of right entry 7, 2,400 feet northeast of the mine mouth.

Section B (sample 15068) was measured at the face of room 8 off right entry 5.

Section C (sample 15069) was measured at the face of room 21 off right entry 3.

Section D (sample 15070) was cut from a pillar on the main air course at the mouth of left entry 2.

Section E (sample 15071) was measured at the face of the fourth left air course, 1,800 feet northeast of the mine mouth.

A composite sample was made by mixing the face samples 15068, 15069, and 15071. The results of an ultimate analysis of this sample are given under laboratory No. 15072.

Notes.—At the time of sampling the coal was undercut by hand and compressed air punchers and shot down with FFF black powder. Black powder was also used for brushing the roof and floor. All the coal was shipped as run-of-mine. Average daily output, 450 tons. Practically all of the coal was derived from advance workings. There was approximately 1,500 acres of unmined coal tributary to this opening. Probable life of mine, about 25 years.

WORTHINGTON. HUTCHINSON MINE.

Sample.—Bituminous coal; analyses Nos. 11152, 11153, 11154, and 11155 (p. 121).

Mine.—Hutchinson; Monongahela district; a slope mine 1 mile from Worthington, on the Baltimore & Ohio Railroad.

Coal bed.—Pittsburgh. Carboniferous age, Monongahela formation. Average thickness, 102 inches; dip, 1½° northwest; roof, firm, smooth, dark shale; floor, hard, smooth fire clay. There is a workable coal bed 120 feet above this bed.

The bed was measured and sampled at four points in the Hutchinson mine by A. A. Straub, on November 5, 1910, as described below:

Sections of coal bed in Hutchinson mine.

Section	A		B		C		D	
Laboratory No	11152		11153		11154		11155	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	10¼	2	8	3	[illegible]	2	8
Coal, bony	0	¼	0	¾	0	[illegible]	0	½
Coal	0	3¼	0	5¼	0	2¼	0	3¼
Coal, bony	0	¼	0	¼	0	[illegible]	0	[illegible]
Coal	0	3[illegible]	0	1¼	0	7¼	0	3¼
Coal, bony	0	[illegible]	0	[illegible]	0	[illegible]	0	[illegible]
Coal	4	5¼	4	2¼	1	6¼	4	2¼
"Sulphur" band	..	..	..	..	[a]0	[illegible]	..	..
Coal	..	..	..	..	1	0	..	..
"Sulphur" band	..	..	..	..	[a]0	[illegible]	..	..
Coal	..	..	..	..	1	7½	..	..
Floor, clay.								
Thickness of bed	8	¼	7	7¼	8	3¼	7	7¼
Thickness of coal sampled	8	¼	7	7¼	8	3¼	7	7¼

[a] Not included in sample.

Section A (sample 11152) was measured on pillar 15 on left heading 3, 2,500 feet south from the mouth of the main slope.

Section B (sample 11153) was measured on pillar 20 on left heading 1, 1,500 feet south of the mouth of the main slope.

Section C (sample 11154) was measured at the face of room 8 off the fourth left butt entry, 3,000 feet south of the mouth of the main slope.

Section D (sample 11155) was measured at the face of room 10 off left heading 2, 1,000 feet south of the mouth of the main slope.

A composite sample was made by mixing samples 11152 to 11155, inclusive. The result of an ultimate analysis of this sample is given under laboratory No. 11156.

Notes.—The coal was mined by the room-and-pillar system. At the time of sampling black powder was used for breaking down the coal after undercutting by electric chain cutters; permissible explosives were used for brushing the roof. None of the coal was shipped as run-of-mine; coal was screened over 1½ and ¾-inch bar screens; pickers were employed on the cars when loading; there were three loading tracks with capacity for 32 cars. Daily output of mine, 400 tons. About 60 per cent of the coal produced was from advance work. Recovery, 90 per cent when pillars are pulled. There was 151 acres of coal tributary to the main slope.

MARSHALL COUNTY.

MOUNDSVILLE. PANAMA MINE.

Sample.—Bituminous coal; analyses Nos. 14486, 14487, and 14488 (p. 122).

Mine.—Panama; Wheeling district; a shaft mine 1 mile south of Moundsville station, on the Baltimore & Ohio Railroad.

Coal bed.—Pittsburgh. Carboniferous age, Monongahela formation. Average thickness, 5 feet 6 inches. There is a distinct cleat extending north and south. Occasional clay veins are encountered. The immediate roof consists of a "draw slate" over which is a roof coal and shale with a limestone cap rock. Floor, underclay and limestone. Small amounts of the shale from the roof and underclay occasionally mix with the coal in mining. Cover at points of sampling, 400 feet.

The coal bed was measured and sampled at two points in the mine by G. H. Deike and J. T. Ryan on July 20, 1912, as described below:

Sections of coal bed in Panama mine.

Section	A		B	
Laboratory No	14486		14487	
Roof, "draw slate", roof coal, and limestone.	*Ft.*	*in.*	*Ft.*	*in.*
Coal, bony	..	..	0	3
Coal	0	7	0	5
Mother coal	0	⅛	0	⅛
Coal	0	1	0	6
Mother coal	0	⅛	0	¼
Coal	1	1	0	6½
Shale, black	..	..	[a] 0	1
Coal	..	..	0	5¼
Coal, bony	..	..	0	2
Mother coal or "sulphur"	[a] 0	1	0	½
Coal	2	0	0	9
Coal with streaks of mother coal and "sulphur"	0	4	0	7¾
"Sulphur" band	..	..	[a] 0	1¼
Coal streaked with bony coal	0	7	0	8
Floor, underclay and limestone.				
Thickness of bed	4	9¼	4	7⅝
Thickness of coal sampled	4	8¼	4	5⅜

[a] Not included in sample.

Section A (sample 14486) was measured at the face of the second south air course, 2,050 feet southeast of the shaft.

Section B (sample 14487) was measured at the face of room 3 off right entry 2 off north entry 2, at a point 1,700 feet northeast of the shaft.

A composite sample was made by combining samples 14486 and 14487. The results of an ultimate analysis of this sample are given under laboratory No. 14488.

Notes.—The Panama mine is opened by a shaft 138 feet deep. The coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut with electric coal cutting machines and broken down with black powder. The machine cuttings were loaded out with the coal. About 35 per cent of the coal passed through the screens. The screening plant consisted of bars 12 feet long with 1¼ and ¾ inch openings. No pickers were employed on the cars or belt. The lumps were small. There were two loading tracks with a capacity of 15 loaded cars. Approximately 3,000 acres remained to be mined. The probable lifetime of the mine was 30 years. The maximum day's run was 650 tons. Average daily output, 450 tons.

MERCER COUNTY.

CRYSTAL. GODFREY MINE.

Sample.—Semibituminous coal; Norfolk & Western field; analyses Nos. 14258, 14259, and 14261 (p. 122).

Mine.—Godfrey, Pocahontas district, a drift mine ½ mile southeast of Crystal on the Wenonah branch of the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 3. Carboniferous age, Pottsville formation. Average thickness, 4 feet 6 inches; dip, 1½ per cent northwest. Roof, smooth gray shale; floor, hard fire clay. Material from the roof and floor does not become mixed with the coal in mining. Cover, 150 to 200 feet.

The coal bed was measured and sampled at three points in the mine by D. J. Price, July 2, 1912, as described below:

Sections of coal bed in Godfrey mine.

Section	A		B		C	
Laboratory No.	14258		14259		14261	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	3	1	10	1	2
"Sulphur" band	[a]0	1	..	..	[a]0	1
Coal	0	5	..	..	0	6
Bone	[a]0	4	[a]0	3	[a]0	3
Coal	1	11	2	2	2	7
Floor, fire clay.						
Thickness of bed	4	0	4	3	4	7
Thickness of coal sampled	3	7	4	0	4	3

[a] Not included in sample.

Section A (sample 14258) was measured on the pillar of right entry 2, 1,100 feet northeast of the mine mouth.

Section B (sample 14259) was measured in room 3, left heading 2, off the old main entry, 700 feet southeast of the mine mouth.

Section C (sample 14261) was measured at the face of room 4, next to "Crystal."

A composite sample was made by mixing samples 14258 and 14259. The results of an ultimate analysis of this sample are given under laboratory No. 14260.

Notes.—The Godfrey mine is opened by drift entry and the coal mined by the room-and-pillar system.

At the time the mine was visited the coal was undercut by hand and shot down with FFF black powder. About 20 per cent of the coal mined was from advance workings and the remainder was from pillars. All of the coal was shipped as run-of-mine and pickers were employed on the cars in loading. The tipple was about 5,000 feet from the mouth of the mine. The distance from the foot of the incline to the tipple was about 2,000 feet. The loading track had a capacity for 14 empty and 17 loaded cars. Daily production, 100 tons. There was approximately 40 acres of unmined coal tributary to this entry.

GOODWILL. GOODWILL MINE.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14163, 14164, 14165, 14166, and 14167 (p. 123).

Mine.—Goodwill; Norfolk and Western district; a drift mine in Goodwill township, ½ mile northeast of Goodwill on a branch of the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 3. Carboniferous age, Pottsville formation. Average thickness 4 feet 8 inches; dip, 2 per cent northwest. Rolls occur in the bottom of the coal, but no faults or horsebacks are encountered. The roof is a "draw slate" 8 inches thick, overlain with sandstone; the floor is a smooth shale about 15 inches thick, below which there is a small amount of coal and sandstone. Particles of the roof do not mix with the coal, although a small amount of the shale bottom becomes mixed with the coal in mining. Cover at points of sampling, 300 to 400 feet.

The coal bed was measured and sampled at four points in the mine by D. J. Price on June 24 and 25, 1912, as described below:

Sections of coal bed in Goodwill mine.

Section	A		B		C		D	
Laboratory No.	14163		14164		14165		14166	
Roof, sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	1	1	1½	1	4	1	3
Bone and "sulphur"	[a] 0	5	[a] 0	5	[a] 0	5	[a] 0	4½
Coal	3	3	3	2½	2	9	2	10
Floor, shale.								
Thickness of bed	4	9	4	9	4	6	4	5½
Thickness of coal sampled	4	4	4	4	4	1	4	1

[a] Not included in sample.

Section A (sample 14163) was measured at the face of room 16, eighth left cross entry, 3,700 feet north of drift mouth No. 3.

Section B (sample 14164) was measured at the face of room 3, off second crosscut off Smith's entry, 3,000 feet northeast of drift mouth No. 3.

Section C (sample 14165) was measured at the face of the first crosscut off Nichol's entry, 2,850 feet east of drift mouth No. 2.

Section D (sample 14166) was measured on a pillar, 2,400 feet east of drift mouth No. 2.

A composite sample was made by combining pillar sample 14166 and face samples 14163, 14164, and 14165. The ultimate analysis of this sample is represented by laboratory No. 14167.

Notes.—The Goodwill mine is opened by 3 drift entries 2,388 feet above sea level. The coal is mined by the room-and-pillar method. At the time of sampling the coal was undercut by hand and electric short wall machines. The undercuttings were loaded out with the coal. The coal was broken down with FFF black blasting powder and a permissible powder was used for brushing the floor and roof. Although all the coal was shipped as run-of-mine, the tipple was equipped with screens as follows: Bars 8 by 12 feet with 3-inch openings, and 8 by 12 feet with ⅝-inch openings. Pickers were employed on the cars in loading. The screens were not washed or coked. The loading track had a capacity for 12 empty and 15 loaded cars. The mine was being run at its capacity, 350 tons per day. About 50 per cent of the coal came from advance workings and 50 per cent from pillars. There was about 400 acres of coal to be taken out from the three openings.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, Bureau of Mines, 1913, pp. 1023–1024.

GOODWILL. LOUISVILLE NO. 1, NO. 2, AND NO. 3 MINES.

Samples.—Semibituminous coal; Pocahontas field; analyses Nos. 14191, 14192, 14193, 14194, 14195, 14196, 14206, and 14207 (p. 123).

Mines.—Louisville Nos. 1, 2, and 3; Norfolk & Western district; drift mines in Goodwill township. Mines 1 and 2 are ½ mile northwest and No. 3 is 1 mile northwest of Goodwill station on the Flipping Creek Branch of the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 3. Carboniferous age, Pottsville formation. Average thickness, 4 feet 8 inches, dip 2 per cent northwest. The roof is a "draw slate" which is smooth and does not become mixed with the coal in mining. The "draw slate" is 8 inches to 4 feet thick, above which is a sandstone cap rock. The floor is a smooth, fairly hard shale in which are thin streaks of coal. Cover at points of sampling, 200 to 400 feet.

The coal bed was measured and sampled at one point in No. 1 mine, at four points in No. 2 mine, and at two points in No. 3 mine by G. H. Deike on June 24, 1912, as described below:

Sections of coal bed in Louisville No. 1, No. 2, and No. 3 mines.

Section	A		B		C		D		E		F		G	
Laboratory No.	14191		14192		14193		14194		14195		14206		14207	
Roof, sandstone.														
	Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, streaked with mother coal	1	1	1	1½	1	1½	1	2½	1	2	1	1½	1	0
"Sulphur" band	[a] 0	½	[a] 0	½	[a] 0	½	..	..	[a] 0	½	[a] 0	½	..	..
Coal	..	..	[a] 0	½	[a] 0	½	..	..	[a] 0	½	[a] 0	½	..	..
Coal, bony	[a] 0	4½	[a] 0	3	[a] 0	3	[a] 0	2½	[a] 0	3	[a] 0	3	[a] 0	5
Coal	..	..	1	4	0	5	1	0	0	8	2	6	..	..
Coal, gray	..	..	0	¼	0	½	0	½	0	¼	0	½	..	..
Coal	..	..	0	6	..	..	..	..	0	11	..	..	..	..
Coal, gray	..	..	0	½	..	..	..	..	0	½	..	..	..	..
Coal	3	2	1	2	2	7½	2	2½	1	11¼	0	9	3	5
Floor, shale.														
Thickness of bed	4	8	4	6¼	4	6½	4	8	5	1	4	9	4	10
Thickness of coal sampled	4	3	4	2¼	4	2½	4	5½	4	9	4	5	4	5

[a] Not included in sample.

Section A (sample 14191) was measured at the face of the main entry, 5,400 feet northeast of mouth of No. 2 mine.

Section B (sample 14192) was measured at the face of room 19, off left heading 9, 5,700 feet northeast of mouth of No. 2 mine.

Section C (sample 14193) was measured on the rib of room 14, off left heading 6, 4,500 feet northeast of mouth of No. 2 mine.

Section D (sample 14194) was measured on the rib of room 13, off left heading 4, 3,000 feet northeast of mouth of No. 2 mine.

Section E (sample 14195) was measured on the barrier pillar between the second left air course and second left pillar, 1,500 feet northwest of mouth of No. 3 mine.

Section F (sample 14206) was measured at the face, 1,950 feet northeast of mouth of No. 3 mine.

Section G (sample 14207) was measured on the pillar of main entry, 900 feet northeast of mouth of No. 1 mine.

A composite sample was made by mixing samples 14191, 14192, 14193, 14194, 14195, 14206, and 14207. The results of an ultimate analysis of this sample are shown under laboratory No. 14196.

Notes.—The mines are opened by drifts and the coal is mined by the room-and-pillar system. At the time the mine was visited electric coal cutters were used for undercutting the coal; FFF black powder was used for breaking down the coal and a permissible explosive for brushing the roof. About 20 per cent of the coal was shipped as run-of-mine and 25 per cent as slack. The screening plant had one shaking screen with 5-inch openings and one bar screen with 3-inch openings. The smaller coal was run over a bar screen with openings of 1 inch to 1¼ inches. Pickers were employed on the cars in loading. The washery had a daily capacity of 1,200 tons. The screenings were coked. The coking plant had a capacity of 125 tons per day. There were two loading tracks with a capacity for 50 empty cars and 20 loaded cars. The storage bins had a capacity for 150 tons. The coal when loaded on the cars had a good appearance and the lumps were large. There was approximately 400 acres of unmined coal tributary to these openings. About 40 per cent of the coal mined was from the advance workings. Daily output, 1,000 tons.

MATOAKA. PAWAMA NO. 1 AND NO. 2 MINES.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14231, 14232, 14233, 14234, 14235, 14236, and 14237 (p. 123).

Mine.—Pawama No. 1 and No. 2; Norfolk & Western district; drift mines at Matoaka, on the Norfolk & Western Railway and near the Virginia Railway.

Coal bed.—Pocahontas No. 3. Carboniferous age, Pottsville formation. Average thickness, 4 feet 4 inches; dip, 1½° northwest. There are no faults, rolls, or horsebacks. There is a characteristic bone parting about 3 inches thick near the middle of the bed. The roof consists of 12 inches of "draw slate," which was shot down after loading out the coal. Floor, soft smooth underclay. Cover at points of sampling, 150 to 175 feet.

The coal bed was measured and sampled at five points in the mines by J. J. Rutledge and D. J. Price on July 1, 1912, as described below:

Sections of coal bed in Pawama No. 1 and No. 2 mines.

	No. 2 mine.						No. 1 mine.			
Section	A		B		C		D		E	
Laboratory No	14231		14232		14233		14235		14236	
Roof, gray shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	0	1	0	0	10	0	10½	1	0
"Sulphur" and bone	[a]0	¼	[a]0	¼	[a]0	1	[a]0	3	[a]0	2
Coal	0	7	0	8½	0	7	0	9	0	9½
Bone	[a]0	2½	[a]0	2½	[a]0	4½	[a]0	4	[a]0	3
Coal	2	7	2	4½	2	7	2	5	2	2
Floor, underclay.										
Thickness of bed	4	4¾	4	3¾	4	5½	4	7½	4	5
Thickness of coal sampled	4	2	4	1	4	0	4	½	3	11½

[a] Not included in sample.

Section A (sample 14231) was measured at the face of the Roanoke heading, 2,060 feet southwest of the drift mouth of No. 2 mine.

Section B (sample 14232) was measured at the face of the main entry, 2,600 feet southwest of the drift mouth of No. 2 mine.

Section C (sample 14233) was measured at the face of room 20, off Norfolk entry, 2,300 feet southwest of the drift mouth of No. 2 mine.

A composite sample was made by combining and mixing samples 14231, 14232, and 14233. The results of an ultimate analysis of this sample are given under laboratory No. 14234.

Section D (sample 14235) was measured on pillar of room 14, off Walker heading, 1,000 feet north of the drift mouth of No. 1 mine.

Section E (sample 14236) was measured on pillar on Godfrey heading, 1,520 feet southwest of the drift mouth of No. 1 mine.

A composite sample was made by mixing pillar samples 14235 and 14236. The results of an ultimate analysis of this sample are given under laboratory No. 14237.

Notes.—The Pawama mines are opened by a drift about 2,600 feet above sea level and are worked by the room-and-pillar system. At the time of sampling the coal was undercut by hand and broken down with FFF black blasting powder. About 80 per cent of the coal was shipped as run-of-mine. The remainder was passed over bar screens 12 feet long with 4-inch openings. Pickers and trimmers were employed on the cars. There were two loading tracks with a capacity of 18 empty cars and 14 loaded cars. Daily output, 450 tons, with a maximum day's run of 700 tons.

MORA. CRANE CREEK NO. 1 AND NO. 2 MINES.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. W32517, W32518, W32519, W32520, W32600, W32601, W32602, and W32603 (pp. 124, 125).

Mine.—Crane Creek No. 1 and No. 2, Norfolk & Western district, slope mines, ½ mile west of Mora on the Pocahontas division of the Norfolk & Western Railway.

Coal bed.—Known as Pocahontas No. 3. Carboniferous (Pottsville) age, Pocahontas formation. Thickness, 4 feet to 6 feet 6 inches. Roof, sandstone and shale; floor, shale. A "rash" 1 to 4 inches thick overlies the coal; this is removed from all run-of-mine coal at the tipple by passing the coal over 3-inch screens. The coal from this bed is of very good quality, and can easily be cleaned from partings.

The coal bed was measured and sampled at eight points in the mine on July 19 and 20, 1912, by P. M. Riefkin, as described below:

Sections of coal bed in Crane Creek No. 1 mine.

Section	A		B		C		D	
Laboratory No.	W32517		W32518		W32519		W32520	
	Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Roof, sandstone and shale.								
Coal, hard, gray	..	..	..	..	0	4	..	..
Coal	0	9¾	0	9	0	7½	0	10
Bony coal to "sulphur"	[a] 0	2¼	[a] 0	2	[a] 0	½	[a] 0	1½
Coal	0	6	0	7	0	11½	3	5½
Coal, hard, gray	0	2½	0	3¾	..	..	..	..
Mother coal	..	..	..	..	0	¼	..	..
Coal	2	5¾	2	5	0	8	..	..
Shale	..	..	[a] 0	2½	..	..	..	..
Mother coal	0	2	..	..	0	¼	..	..
Coal	..	..	0	3¾	1	11½	..	..
Bony coal to shale	[a] 0	2¼	..	..	[a] 0	4	..	..
Coal	0	4½	..	..	0	8	..	..
Floor, shale.								
Thickness of bed	4	11	4	9	5	7½	4	5
Thickness of coal sampled	4	6½	4	4½	5	3	4	3½

[a] Not included in sample.

Section A (sample W32517) was measured at the face of the main heading.

Section B (sample W32518) was measured at the face of eighteenth cross air course.

Section C (sample W32519) was measured on the pillar beyond the second cut-through in room 11 off cross entry 16.

Section D (sample W32520) was measured on the pillar beyond the second cut-through in room 21 off cross entry 6, off the main heading.

Sections of coal bed in Crane Creek No. 2 mine.

Section	A		B		C		D	
Laboratory No.	W32600		W32601		W32602		W32603	
	Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Roof, sandstone and shale.								
Muck	[a] 0	2	..	..	..	..	[a] 0	2
Coal	0	5½	0	10½	0	9½	0	8
Mother coal (hard)	0	¼	..	..	..	..	0	1½
Coal	0	5	..	..	..	..	0	½
Coal, bony	[a] 0	1	[a] 0	3	[a] 0	1¾	[a] 0	3
Coal	..	..	2	0	0	6½	0	7
Coal, hard, gray	..	..	0	4	0	3	0	2
Coal	3	6	1	2	2	7½	1	8½
Mother coal, hard, to shale	[a] 0	1	[a] 0	2	Streak.		0	1
Coal	..	..	..	..	0	1	..	..
Mother coal, hard, bony	..	..	..	..	[a] 0	2	[a] 0	2½
Coal	[a] 0	4	0	5½	0	5½	0	8½
Floor, fire clay and shale.								
Thickness of bed	5	½	5	3	5	¾	4	8½
Thickness of coal sampled	4	4½	4	10	4	9	4	1

[a] Not included in sample.

Section A (sample W32600) was measured in a pocket above the second cut-through in room 4, off cross entry 3.

Section B (sample W32601) was measured at the face of room 14, off Allegheny entry.

Section C (sample W32602) was measured on a chain pillar between rooms 21 and 22, off Blue Ridge entry.

Section D (sample W32603) was measured at the face of room 13, off Bryant entry.

Notes.—The Crane Creek No. 1 and No. 2 mines dumped coal over the same tipple. At the time of sampling the coal in these mines was undercut by hand with picks in the bottom part of the seam, and shot down with black powder. The tipple was constructed of wood and steel, and equipped with a 3-inch bar screen, a 2-inch, 1½-inch, 1-inch and ⅝-inch revolving screen, and a picking table. The inspection and preparation of this coal was exceptionally good; all coal passing over the 4-inch screen was shipped as run-of-mine coal, and passed over a picking table before going into the railroad cars. The commercial sizes as shipped, besides run-of-mine, were egg and stoker coal. All of the coal which passed through the ⅝-inch mesh of the revolving screen was washed. The washed coal was not shipped, but was coked.

The average output of the two mines was 1,000 tons per day, with a daily capacity of 1,750 tons. The unmined area was about 3,800 acres.

For analyses and descriptions of other samples of coal from this mine, see Bull. 22, Bureau of Mines, 1913, pp. 1026–1027.

MORA. PINNACLE MINE.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. W32604, W32605, W32606, W32607, W32608, and W32609 (p. 125).

Mine.—Pinnacle; Norfolk & Western district; a slope mine; 1 mile north of Mora on the Pocahontas division of the Norfolk & Western Railway.

Coal bed.—Known as Pocahontas No. 3. Carboniferous (Pottsville) age, Pocahonta, formation. The coal in this mine is 4 to 5 feet 6 inches thick. Roof and floor goods a "rash" 1 to 4 inches thick overlies the coal; this is eliminated from run-of-mine coal at the tipple by bar screens. The coal from this bed is of a good quality and can easily be cleaned.

The coal bed was measured and sampled at six points in the mine by P. M. Riefkin on July 21, 1912, as described below:

Sections of coal bed in Pinnacle mine.

Section	A		B		C		D		E		F	
Laboratory No.	W32604		W32605		W32606		W32607		W32608		W32609	
Roof, shale and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Muck and "draw slate"	[a]0	1	[a]0	1½	[a]0	2½	[a]0	3	[a]0	1½	[a]0	1
Coal, gray	..	..	..	..	..	..	..	..	0	6½	..	..
Coal	1	2½	0	11¼	1	0	0	10	0	5	0	11
Mother coal	..	..	0	¼	..	..	0	½	..	..	0	½
"Sulphur"	..	..	..	..	..	..	..	..	[a]0	¾	[a]0	1½
Coal, bony	[a]0	2	..	..	[a]0	3	..	..	..	..	..	..
Coal	1	6¼	0	1¼	3	3	0	5½	0	5½	0	2
Mother coal, hard, bony	..	..	0	¼	..	..	..	..	..	..	..	..
Coal	..	..	0	2	..	..	..	..	..	..	..	..
Coal, bony	0	¼	[a]0	3	..	..	[a]0	2¾	[a]0	2	[a]0	2½
Coal	1	2¾	2	8½	..	..	3	0	2	11	3	4½
Mother coal	..	..	0	½	..	..	..	..	..	..	..	..
Floor, shale.												
Thickness of bed	4	2¾	4	4½	4	8½	4	9¾	4	8¼	4	11
Thickness of coal sampled	3	11¾	4	0	4	3	4	4	4	4	4	6

[a] Not included in sample.

Section A (sample W32604) was measured at the face of cross entry 2, north mine.

Section B (sample W32605) was measured at the face of the north main heading.

Section C (sample W32606) was measured at face of a pocket in the first pillar of room 25, cross entry 2, off main heading.

Section D (sample W32607) was measured at the face of cut-through off Cuba entry, off the second main heading.

Section E (sample W32608) was measured at the face of main heading 2.

Section F (sample W32609) was measured on a pillar of room 16, off cross entry 4, off the first main heading.

Notes.—At the time of sampling the coal in this mine was undercut in the bottom part of the seam by hand with picks, and shot down with black powder. The tipple was constructed of wood and equipped with a 3-inch bar screen and a 2-inch, 1½-inch, 1-inch, and ¾-inch revolving screen, washer, and picking table. The inspection and preparation of the coal for the market was exceptionally good. The coal was picked on a picking table and on the cars after passing over the 4-inch bar screen. The coal that passed through the 4-inch screen fell into the revolving screen, and the screened coal was shipped as egg and stoker coal. All the coal that passed through the ¾-inch mesh revolving screen was washed. Very little of the washed coal was shipped, practically all of it being coked. Average daily output, 1,000 tons, with a capacity of 1,600 tons per day. The unmined area was about 1,700 acres.

For analyses and descriptions of other samples of coal from this mine see Bulletin 22, Bureau of Mines, 1913, pages 1027–1028.

MORA. SAGAMORE NO. 1 AND NO. 2 MINES.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14250, 14251, 14252, 14253, 14254, 14255, 14256, and 14257 (p. 125).

Mine.—Sagamore No. 1 and No. 2; Norfolk and Western district; drift mines 1 mile west and 1 mile east of Mora, respectively, both on the Crane Creek branch of the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 3. Carboniferous (Pottsville) age, Pocahontas formation. Average thickness of bed in No. 1 mine about 5 feet, in No. 2 mine about 4 feet 6 inches. Roof, smooth, dark, soft, sandy shale, 10 to 24 inches thick. Floor, light-gray shale, smooth, and fairly hard. Occasionally a small amount of the roof material becomes mixed with the coal, but none of the floor is broken loose during mining operations. Cover at points of sampling, 250 to 400 feet.

The coal bed was measured and sampled at three points in No. 1 mine and at four points in No. 2 mine by H. I. Smith and G. H. Deike, July 2, 1912, as described below:

Sections of coal bed in Sagamore No. 1 mine.

Section	A		B		C	
Laboratory No.	14250		14251		14252	
Roof, sandy shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	9	0	9	0	10
Coal, bony	[a]0	4	[a]0	3½	[a]0	6
Coal	..	..	0	9	0	9
Coal, gray	..	..	0	1½	0	1½
Coal	..	..	0	6	..	..
Coal, gray	..	..	0	½	..	..
Coal	3	2	2	1	2	0
Mother coal	..	..	..	..	0	½
Coal	..	..	..	..	0	6
"Rash"	[a]0	9	[a]0	6	[a]0	6
Coal (streaked)	..	..	0	6	0	8
Floor, shale.						
Thickness of bed	5	0	5	6½	5	11
Thickness of coal sampled	3	11	4	9	4	11

[a] Not included in sample.

Section A (sample 14250) was measured at the face of the main straight entry H, 2,250 feet northwest of the mine mouth.

Section B (sample 14251) was measured at the face of G-1 heading, 2,250 feet west of the mine mouth.

Section C (sample 14252) was measured at the face of room 11 off G-2 heading, 1,900 feet southwest of the mine mouth.

Sections of coal bed in Sagamore No. 2 mine.

Section	A		B		C		D	
Laboratory No.	14253		14254		14255		14256	
Roof, sandy shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	4	1	0	0	3	0	11½
Coal, gray	0	2	..	..	0	2	..	..
"Sulphur" band	..	..	..	..	..	..	0	¼
Coal	0	6½	..	..	0	7	0	2¼
"Sulphur" band	0	½	a 0	½	a 0	½	..	..
Bone	a 0	4	a 0	3½	a 0	3½	0	½
Coal	..	..	0	7½	0	7	0	6½
Coal, gray	..	..	0	3½	0	3	0	3
Coal	2	3	2	2½	2	4	1	9
Mother coal	0	½	..	..	..	..	0	½
Coal	0	9½	..	..	..	..	0	8½
Floor, shale.								
Thickness of bed	4	6	4	5½	4	6	4	6
Thickness of coal sampled	4	2	4	1½	4	2	4	6

a Not included in sample.

Section A (sample 14253) was measured at the face of H-8 entry, 2,550 feet southeast of the mine mouth.

Section B (sample 14254) was measured at the face of G-6 entry, 2,300 feet southwest of the mine mouth.

Section C (sample 14255) was measured at the face of G-8 entry, 2,550 feet south of the mine mouth.

Section D (sample 14256) was measured at the face of the straight H air course, 2,350 feet southeast of the mine mouth.

A composite sample was made by mixing face samples 14250 to 14256, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14257.

Notes.—The Sagamore mines are opened by drift entries and the coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by electric chain machines and by hand. FFF black powder was used for breaking down the coal and a permissible explosive for brushing the roof. About 30 per cent of the coal was shipped as run-of-mine. The screening plant was equipped with shaking screens 5 by 12 feet with openings 4 by 6 inches. About 75 per cent of the coal passed through this screen. The smaller sizes of coal were run over shaking screens 5 by 20 feet, with holes 3 inches, 1½ inches, and 1 inch in diameter. There were two belt conveyors at which a number of pickers were employed. The screenings were washed, yielding coal of a maximum size of 1½ inches and producing nut and slack coal. The washery had a capacity of 600 tons per day. There were 3 loading tracks with a capacity of 40 empty and 50 loaded cars and storage bins for 200 tons. The tipple and washery equipment was very complete and the preparation for market was thorough. The coal being loaded was chiefly in large lumps which had a good appearance. Daily output of the two mines, 800 tons. There was about 3,000 acres of unmined coal, a large part of which was tributary to the two mines. About 30 per cent of the output was from advance workings and the remainder from pillars.

WEYANOKE. WEYANOKE NO. 1 AND NO. 2 MINES.

Sample.—Semibituminous coal; Pocahontas field; analyses Nos. 14224, 14225, 14226, 14227, 14228, 14229, and 14230 (p. 126).

Mine.—Weyanoke No. 1 and No. 2, Norfolk & Western district; drift mines about one-fourth mile north of Weyanoke station on the Wenonah branch of the Norfolk & Western Railway.

Coal bed.—Pocahontas No. 3. Carboniferous (Pottsville) age, Pocahontas formation. Average thickness, 4 feet. There are no faults or horsebacks, but there are occasional rolls and local dips. The cleat is well marked. The roof consists of about 8 inches of "draw slate," which is taken down after loading out the coal. Above the "draw slate" is a main cap rock. Particles of the roof sometimes got mixed with the coal. The floor consists of 8 to 10 inches of dark shale, under which is 4 inches of coal; under the coal is sandstone. Cover, 300 to 400 feet.

The coal bed was measured and sampled at 4 points in No. 1 mine and at 2 points in No. 2 mine by H. I. Smith and G. H. Deike on July 1, 1912, as described below:

Sections of coal bed in Weyanoke No. 1 and No. 2 mines at Weyanoke.

Section	A		B		C		D		E		F	
Laboratory No.	14224		14225		14226		14227		14228		14230	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Cannel (cube) coal	[a]0	½	[a]0	1	[a]0	½	[a]0	1	[a]0	1	[a]0	1
Coal	1	0	0	10	1	0	1	0	0	9	1	0
Bone, gray	..	..	[a]0	1½	[a]0	2½	0	⅛	0	¼	[a]0	4
Coal	..	..	0	4	..	..	0	2	0	2	1	9
Coal, gray	[a]0	2	0	4	..	..	0	⅛	[a]0	2½	0	½
Coal	..	..	..	..	..	..	0	7	1	4	..	..
Mother coal	..	..	..	..	..	..	0	⅛	0	¼	..	..
Coal	2	11	2	4	3	3	2	1⅝	1	8½	1	1½
Floor, dark shale.												
Thickness of bed	4	1½	4	½	4	6	4	0	4	3½	4	4
Thickness of coal sampled	3	11	3	10	4	3	3	11	4	0	3	11

[a] Not included in sample.

Section A (sample 14224) was measured at the face of room 11, off left entry 9, off main heading of No. 1 mine.

Section B (sample 14225) was measured at the face of the main air course of No. 1 mine.

Section C (sample 14226) was measured at the face of room 22, off eighth left cross heading of No. 1 mine.

Section D (sample 14227) was measured at the face of right entry 3, off main entry of No. 2 mine.

Section E (sample 14228) was measured at the face of room 6, off left entry 6 of No. 1 mine.

Section F (sample 14230) was measured on pillar of last crosscut in room 7, off second left butt entry of No. 2 mine.

A composite sample was made by combining and mixing samples 14224, 14225, 14226, 14227, and 14228. The results of an ultimate analysis of this sample are given under laboratory No. 14229.

Notes.—The Weyanoke mines are opened by drifts and the coal is mined by the room-and-pillar system. At the time of sampling the coal was top cut by electric chain machines and broken out with FFF black blasting powder; permissible explosives were used for brushing the floor and roof. The machine cuttings were loaded out with the coal. None of the coal was shipped as run-of-mine, and about 25 per cent passed through bar screens 16 feet long having 1¼-inch openings. Twelve men were employed as pickers on the belt. The lumps on the car were large and had a good appearance. There were two loading tracks, with a capacity of 25 empty cars and 30 loaded cars. Most of the coal was shipped for domestic purposes. The daily output of both mines was 600 tons, with a maximum day's run of 1,000 tons. Ten per cent of the coal came from advance workings and the remainder from pillars. About 1,000 acres remained to be mined through the existing openings.

MONONGALIA COUNTY.

RICHARD. ELKINS NO. 1 MINE.

Sample.—Bituminous coal; analyses Nos. W12204, W19475, W19476, W19477, W19478, and W19479 (pp. 126, 127).

Mine.—Elkins No. 1; Monongahela district; a slope mine ½ mile northeast of Richard, on the Morgantown & Kanawha Railroad, with connections to the Baltimore & Ohio Railroad at Rowlesburg.

Coal bed.—Upper Freeport. Carboniferous age, Allegheny formation. Average thickness, 4 feet. Roof, low-grade coal about 3½ feet thick, overlain by sandstone; floor, hard clay with smooth surface.

The bed was measured and sampled at five points in the mine by G. S. Pope, April 22, 1911, as described below:

Sections of coal bed in Elkins No. 1 mine.

Section	A		B		C		D		E	
Laboratory No	W19475		W19476		W19477		W19478		W19479	
	Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Roof, coal and sandstone.										
Coal	0	8¾	2	7	2	6½	2	8	2	8½
Mother coal	0	¾	..	..	0	1	..	..	..	..
Coal	2	1½	..	..	0	4	..	..	..	..
Shale	[a]0	1½	[a]0	1¼	[a]0	¾	[a]0	2¼	[a]0	2¼
Coal	0	11¼	0	3	1	2	1	1	0	6
"Sulphur"	..	..	[a]0	⅜	..	..	..	..	..	..
Coal	..	..	0	3½	..	..	..	..	..	..
Mother coal	..	..	0	½	..	..	..	..	0	¼
Coal	..	..	0	4	..	..	..	..	0	7½
Coal and parting	[a]0	1	..	..	..	..	..	..	..	..
Floor, hard clay.										
Thickness of bed	4	¾	3	7⅝	4	2¼	3	11¼	4	½
Thickness of coal sampled	3	10¼	3	6	4	1½	3	9	3	10¼

[a] Not included in sample.

Section A (sample W19475) was measured at the face of the fifth right "flat," 4,200 feet northeast of the drift mouth.

Section B (sample W19476) was measured at the face of butt entry 8 off fifth right entry, about 700 feet from the main entry.

Section C (sample W19477) was measured at the face of first right heading, 4,200 feet from the drift mouth.

Section D (sample W19478) was measured at the face of butt entry 11, off first right entry, 750 feet from right entry 1.

Section E (sample W19479) was cut from butt entry 5, off fifth right "flat," 1,400 feet from fifth "flat."

A composite sample was made by mixing samples W19475, W19476, W19477, W19478, and W19479. The results of an ultimate analysis of this sample are given under laboratory No. W12204.

Notes.—The Elkins No. 1 mine is opened both by drift entries and a slope. At the time of sampling the coal was undercut by hand and machines and shot down with black powder. There were two loading tracks with a capacity for 20 cars. Pickers were employed on the cars in loading. The mine was equipped to produce 700 tons per day. Average daily output 425 tons, the larger part of which was from advance workings. There was about 500 acres of coal tributary to this and other openings.

PRESTON COUNTY.

MASONTOWN. ELKINS NO. 6 MINE.

Sample.—Bituminous coal; analyses Nos. W19493, W19494, W19540, and W19541 (p. 127).

Mine.—Elkins No. 6; Monongahela district; a drift mine at Masontown, on the Morgantown & Kanawha Railroad with connections to the Baltimore & Ohio Railroad at Rowlesburg.

Coal bed.—Upper Freeport. Carboniferous age, Allegheny formation. Average thickness, 4 feet. Roof, shale, bone, and dirty coal, overlain by sandstone; floor, smooth hard fire clay.

The bed was measured and sampled at four points in the mine by G. S. Pope, April 24, 1911, as described below:

Sections of coal bed in Elkins No. 6 mine.

Section	A		B		C		D	
Laboratory No.	W19493		W19494		W19540		W19541	
Roof, shale, bone, dirty coal and sandstone.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	2½	1	9½	2	0	2	8
"Sulphur"	0	⅜	..	..	..	..	..	..
Coal	0	6	..	..	..	..	..	..
Mother coal	0	⅜	0	¼	0	½	..	..
Coal	0	10	0	8	0	7⅝	..	..
Shale	[a]0	1½	[a]0	2¼	[a]0	2	[a]0	2¼
Coal	0	3½	0	3½	0	4	0	3½
Bone	0	⅝	0	⅝	0	½	0	⅝
Coal	0	8½	0	9¾	0	7⅜	0	9
Shale	0	¼	0	¾	0	⅜	0	⅜
Coal	0	3¼	0	3¼	0	3	0	3⅜
Floor, hard, smooth fire clay.								
Thickness of bed	4	⅞	4	1⅞	4	1⅜	4	3⅛
Thickness of coal sampled	3	11⅜	3	11⅝	3	11⅜	4	⅞

[a] Not included in sample.

Section A (sample W19493) was measured at the face of the first butt heading, 600 feet from the third "flat," 150 feet from the main entry.

Section B (sample W19494) was measured at the face of the third right entry, 50 feet from the main entry.

Section C (sample W19540) was measured at the face of the second butt heading, 1,000 feet from the main entry.

Section D (sample W19541) was measured at the face of the second butt entry, 700 feet from the second "flat," 300 feet from the main entry.

A composite sample was made by mixing samples W19493, W19494, W19540, and W19541. The results of an ultimate analysis of this sample are given under laboratory No. 12225.

Notes.—The Elkins No. 6 mine is opened by a drift entry. The mine is comparatively new, and only a few side entries have been started. At the time of sampling the coal was undercut by hand and broken down with black powder. There was one loading track at the tipple with a capacity of 15 cars. There were no screens, the entire production being shipped as run-of-mine. Pickers were employed on the cars in loading. All of the coal was used for coking purposes. There was 1,200 acres of unmined area tributary to this opening.

RALEIGH COUNTY.

HOTCOAL. BIG STICK MINE.

Sample.—Semibituminous coal; New River field; analyses Nos. 14375, 14376, 14377, and 14378 (p. 127).

Mine.—Big Stick; Kanawha-New River district; a drift mine at Hotcoal, with railroad connections to the Virginian Railway.

Coal bed.—Beckley. Carboniferous age, Pottsville formation. Average thickness, 5 feet 6 inches; dip, 2½° northeast. The roof consists of 3 inches of "draw slate," above which is 2 feet of dark carbonaceous shale mixed with coal, above which is sandstone cap rock. Floor, hard, smooth shale. Material from the roof and floor does not become mixed with the coal in mining. Cover, 150 to 250 feet.

The coal bed was measured and sampled at 3 points in the mine by J. J. Rutledge, July 8, 1912, as described below:

Sections of coal bed in Big Stick mine.

Section	A		B		C	
Laboratory No	14375		14376		14377	
Roof, "draw-slate," shale	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	[a]0	1	..	..	[a]0	¾
Shale, black	[a]0	1	..	..	[a]0	1
Coal, dirty	[a]0	¾	..	..	[a]0	¾
Coal	3	3	2	6	3	4
Mother coal	..	..	0	½	..	..
Coal	..	..	0	10	..	..
Cannel coal	0	½	..	..	0	3
Coal, bony	..	..	0	¾	..	..
Coal	2	0	1	10	2	6
Floor, fire clay.						
Thickness of bed	5	6	5	3¼	6	3½
Thickness of coal sampled	5	3½	5	3¼	6	1

[a] Not included in sample.

Section A (sample 14375) was measured at the face of left heading 2, 1,600 feet east of the mouth of the straight main entry.

Section B (sample 14376) was measured at the face of left entry 3, off the straight main entry, 1,400 feet northeast of the mine mouth.

Section C (sample 14377) was measured at the face of the straight main entry, 1,900 feet northeast of the mine mouth.

A composite sample was made by mixing face samples 14375, 14376, and 14377. The results of an ultimate analysis of this sample are given under laboratory No. 14378.

Notes.—The Big Stick mine is opened by two entries and the coal mined by the room-and-pillar system. At the time of sampling the coal was undercut by hand and shot down with FF black blasting powder. A permissible explosive was also used in some parts of the mine. All of the coal was shipped as run-of-mine. Pickers were employed on the car in loading. There was no screening plant. The mine was new having been in operation only about 1 year at the time of sampling. All of the coal was obtained from advance workings. Daily output, 450 tons per day, with a maximum capacity of 500 tons.

McQuade. Blue Jay No. 4 Mine.

Sample.—Semibituminous coal; New River field; analyses Nos. 14288, 14289, 14290, and 14291 (p. 128).

Mine.—Blue Jay No. 4; Kanawha-New River district; a drift mine at McQuade station 2 miles southeast of Raleigh, on the Piney branch of the Chesapeake & Ohio Railway.

Coal bed.—Beckley. Carboniferous age, Pottsville formation. Average thickness, 5 feet; slight dip, northwest. Roof, smooth shale 1 to 2 feet thick; floor, very soft fire clay; material from the roof and the floor does not become mixed with the coal in mining. Cover at points of sampling 75 to 100 feet.

The coal bed was measured and sampled at three points in the mine by D. J. Price, July 10, 1912, as described below:

Sections of coal bed in Blue Jay No. 4 mine.

Section	A		B		C	
Laboratory No	14288		14289		14290	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	4	1	2	1	2
Bone and shale	[a]0	9	[a]0	3	..	..
Coal, bony	..	..	..	..	0	3
Coal	2	10	3	0	3	0
Floor, soft fire clay.						
Thickness of bed	4	11	4	5	4	5
Thickness of coal sampled	4	2	4	2	4	5

[a] Not included in sample.

Section A (sample 14288) was measured on a pillar of the main heading about 600 feet east of the mine mouth.

Section B (sample 14289) was measured at the face of the second right heading, 400 feet south of the mine mouth.

Section C (sample 14290) was measured on the pillar of the first right heading, 250 feet south of the mine mouth.

A composite sample was made by mixing samples 14288, 14289, and 14290. The result of an ultimate analysis of this sample are shown under laboratory No. 14291.

Notes.—The Blue Jay No. 4 mine is opened by drift entries and the coal mined by the room-and-pillar system. At the time of sampling the coal was undercut by hand. As the mining operations were confined entirely to pillars, no powder was used in breaking the coal. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. Daily output, about 150 tons. There was one loading track with a capacity for 10 empty and 7 loaded cars.

RALEIGH. RALEIGH NO. 1 AND NO. 3 MINES.

Samples.—Semibituminous coal; New River field; analyses Nos. 10624, 10625, 10626, 10627, 10628, and 10630 (p. 128).

Mine.—Raleigh No. 1 and No. 3; Kanawha-New River district; slope mines at Raleigh, on the Chesapeake & Ohio Railway.

Coal bed.—Beckley. Carboniferous (Pottsville) age, Quinnimont formation. Average thickness, 52 inches; dip about 2° N. 50° W.; roof, shale 2 feet thick with a sandstone capping 90 feet thick; floor, hard, smooth, gray shale.

The bed was measured and sampled at five points in the mines by J. J. Rutledge on June 25, 1910, as described below:

Sections of coal bed in Raleigh No. 1 and No. 3 mines.

Section	A		B		C		D		E	
Laboratory No.	10624		10625		10626		10627		10630	
	Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Roof, shale.										
Coal	1	4	1	4	1	0	..	..	1	3
Coal, soft	0	⅛	0	½	0	⅛	..	..	0	¼
Coal	3	10½	2	9½	3	1¾	3	7½	3	10½
Floor, shale.										
Thickness of bed	5	2⅝	4	2	4	1⅞	3	7½	5	1¾
Thickness of coal sampled	5	2⅝	4	2	4	1⅞	3	7½	5	1¾

Section A (sample 10624) was measured on the left rib of left entry 7 opposite the mouth of right heading 1 in No. 3 mine.

Section B (sample 10625) was measured at the face of right entry 15 in No. 3 mine.

Section C (sample 10626) was measured at the last crosscut in the main entry in No. 3 mine.

Section D (sample 10627) was measured at the face of room 11 off right heading 12 in No. 3 mine.

Section E (sample 10630) was measured at the neck of the last break-through off right entry 6 in No. 1 mine.

A composite sample was made by mixing samples 10624 to 10627, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 10628.

Notes.—No. 1 and No. 3 mines are connected and in reality form but a single mine, opened by two drift entries; double entry, room-and-pillar system of mining. At the time of sampling black powder was used for breaking coal after undercutting with electric chain cutters; bar screens; capacity, 1,000 tons a day; two loading tracks with capacity for 40 cars; unmined area about 2,000 acres.

SLAB FORK. SLAB FORK NO. 1, NO. 2, NO. 3, NO. 4, AND NO. 5 MINES.

Sample.—Semibituminous coal; New River field; analyses Nos. 14313 to 14318, and 14325 to 14337 (pp. 129, 130).

Mines.—Slab Fork No. 1, No. 2, No. 3, No. 4, and No. 5; Kanawha-New River district; drift mines; Nos. 1, 2, 3, and 4 are about three-fourths of a mile northeast of Slab Fork; No. 5 is one-fourth of a mile south of Slab Fork, on the Virginian Railway.

Coal bed.—Beckley. Carboniferous (Pottsville) age, Quinnimont formation. Thickness, 3 feet 5 inches to 4 feet 7 inches. Dip slightly irregular, but the main direction is northwest. Roof, sandy shale containing about 3 inches of roof coal. The shale and roof coal combined vary in thickness from 18 inches to 3 feet, above which there is a sandstone cap rock. Floor, hard smooth clay. Material from the roof and floor did not become mixed with the coal in mining. Cover at points of sampling, 100 to 400 feet.

The coal bed was measured and sampled at 15 points in the five mines by G. H. Deike and H. I. Smith July 10 and 11, 1912, as described below:

Sections of coal bed in Slab Fork No. 1 and No. 4 mines.

Section	No. 4		No. 1	
Laboratory No.	14336		14387	
Roof, sandy shale.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	0	11	[a]0	1½
Shale	..	..	[a]0	½
Mother coal	0	¼	..	..
Coal	0	8½	2	7
"Sulphur"	..	..	Streak.	
Coal, gray	0	2	..	..
Coal	0	2½	..	..
Coal, gray	0	1	..	..
Coal	1	8	1	4
Coal, bony	[a]0	3	[a]0	4
Coal	..	..	0	11½
Floor, clay.				
Thickness of bed	4	¼	5	4½
Thickness of coal sampled	3	9¼	4	10½

[a] Not included in sample.

Section A (sample 14336) was measured at the face of the main entry, about 1,450 feet northwest of the mine mouth.

Section B (sample 14337) was measured on the pillar of room 7, off second right entry, 1,200 feet northeast of the mine mouth.

Sections of coal bed in Slab Fork No. 2 mine.

Section	A		B		C		D		E	
Laboratory No.	14325		14326		14328		14329		14330	
Roof, sandy shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	[a]0	3	[a]0	3	[a]0	3	[a]0	2	[a]0	2½
Shale	[a]0	1½	[a]0	3	[a]0	3	[a]0	1	[a]0	3½
Coal	0	5	0	2	0	2	0	1½	0	2
Coal, gray	0	2	0	4	0	4	0	4½	0	4
Coal	3	5	3	6	1	3½	2	2½	3	6
Mother coal	0	1	..	..	0	½	0	¼	..	..
Coal	0	9	..	..	1	10	1	4	..	..
Bone	[a]0	4	..	..	..	..	[a]0	1	..	..
Floor, clay.										
Thickness of bed	5	6½	4	6	4	2	4	4¾	4	6
Thickness of coal sampled	4	10	4	0	3	8	4	¾	4	0

[a] Not included in sample.

Section A (sample 14325) was measured on the pillar of third left entry, between rooms 7 and 8, 1,800 feet southwest of the mine mouth.

Section B (sample 14326) was measured on the pillar between rooms 8 and 10, off seventh left entry, 3,000 feet southwest of the mine mouth.

A composite sample was made by mixing pillar samples 14325 and 14326. The results of an ultimate analysis of this sample are shown under laboratory No. 14327.

Section C (sample 14328) was measured on the face of the first panel off the second drift, 4,200 feet south of the mine mouth.

Section D (sample 14329) was measured at the face of first right entry, off third left entry, 1,200 feet southeast of the mine mouth.

Section E (sample 14330) was measured at the face of fourteenth left entry, 150 feet from the straight main entry, 5,000 feet southwest of the mine mouth.

A composite sample was made by mixing face samples 14328, 14329, and 14330. The results of an ultimate analysis of this sample are shown under laboratory No. 14331

Sections of coal bed in Slab Fork No. 3 mine.

Section	A		B		C	
Laboratory No	14332		14333		14334	
Roof, sandy shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	..	..	[a]0	1½	0	1½
Shale	..	..	[a]0	1	[a]0	1
Coal, impure	..	..	[a]0	2	..	..
Coal	0	4	0	9½	1	½
Coal, hard	..	..	0	2	0	2
Coal, soft	..	..	1	2	..	..
Mother coal	..	..	0	½	..	..
Coal, bony	0	1½	..	..	..	..
Coal	..	..	0	4½	..	..
Mother coal	..	..	0	1	..	..
Coal	3	5	0	4½	2	9
Mother coal	..	..	[a]0	1	..	..
Shale, bony	[a]0	4	..	..	[a]0	3
Coal	1	0	0	8½	..	..
Coal, bony	[a]0	1	..	..	..	..
Floor, clay.						
Thickness of bed	5	3½	4	2	4	5
Thickness of coal sampled	4	10½	3	8½	4	1

[a] Not included in sample.

Section A (sample 14332) was measured at the face of the main air course, 2,450 feet southeast of the mine mouth.

Section B (sample 14333) was measured at the face of A entry, 2,350 feet southeast of the mine mouth.

Section C (sample 14334) was measured at the face of room 3, off the first right entry, 2,600 feet southeast of the mine mouth.

A composite sample was made by mixing face samples 14332, 14333, and 14334. The results of an ultimate analysis of this sample are given under laboratory No. 14335.

Sections of coal bed in Slab Fork No. 5 mine.

Section	A		B		C		D		E	
Laboratory No	14313		14314		14315		14316		14318	
Roof, sandy shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, gray	..	..	..	..	0	4	..	..	..	..
Coal	[a]0	2	..	..	0	6½	..	..	..	..
Coal with "sulphur"	..	..	..	..	..	..	[a]0	2½	..	..
Shale	[a]0	3	..	..	Streak.		[a]0	1	..	..
Coal	1	0	0	10	0	6	0	11½	1	0
Coal, gray	..	..	..	..	0	2½	..	..	..	..
Shale	[a]0	2	[a]0	½	..	..	0	¼	0	½
Coal	..	..	0	7	0	2½	..	..	..	..
Coal, gray	..	..	0	2	0	2	..	..	..	..
Coal	2	10	1	11½	1	8	2	9	2	9
Floor, hard clay.										
Thickness of bed	4	5	3	7	3	7½	4	¼	3	9½
Thickness of coal sampled	3	10	3	6½	3	7½	3	8¾	3	9½

[a] Not included in sample.

Section A (sample 14313) was measured at the face of the main heading, 3,200 feet southwest of the mine mouth.

Section B (sample 14314) was measured at the face of room 15, off the sixth left entry, 2,000 feet southwest of the mine mouth.

Section C (sample 14315) was measured at the last cut-through, off the third left entry, 1,900 feet southeast of the mine mouth.

Section D (sample 14316) was measured on the pillar of room 10, off the second left entry, 800 feet south of the mine mouth.

Section E (sample 14318) was measured on the pillar of fourth left entry, between rooms 13 and 14, 1,250 feet south of the mine mouth.

A composite sample was made by mixing samples 14313 to 14316, inclusive. The results of ultimate analysis of this sample are shown under laboratory No. 14317.

Notes.—All of these mines are opened by drift entries and the coal mined by the room-and-pillar system. At the time of sampling, electric chain machines were used for undercutting the coal in Nos. 2, 3, and 4 mines. Machines were not used in the other mines. A permissible explosive was used for breaking the coal and for brushing the roof. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. No. 5 mine was equipped with 2 loading tracks with a capacity for 15 empty and 15 loaded cars; the other mines have 2 loading tracks with a capacity for 40 empty and 40 loaded cars.

The capacity of the mines was as follows: No. 1 mine, 30 tons a day; No. 2 mine, 350 tons a day; No. 3 mine, 170 tons a day; No. 4 mine, 60 tons a day, and No. 5 mine, 425 tons, or a total capacity of 1,035 tons a day. About 35 per cent of the coal was obtained from advance work. There was approximately 1,800 acres of unmined coal tributary to the various openings.

WINDING GULF. LYNWIN MINE.

Sample.—Semibituminous coal; New River field; analyses Nos. 14319, 14320, 14321, 14322, 14323, and 14324 (p. 130).

Mine.—Lynwin; Kanawha-New River district; a drift mine, one-fourth of a mile south of Winding Gulf on the Winding Gulf branch of the Virginian Railway.

Coal bed.—Beckley. Carboniferous (Pottsville) age, Quinnimont formation. Average thickness 5 feet; dip, slight, to the northwest. Roof, light blue shale to which the coal adheres more or less; floor, hard smooth shale that does not become mixed with the coal in mining. Cover, 400 feet.

The coal bed was measured and sampled at five points in the mine by D. J. Price on July 11, 1912, as described below:

Sections of coal bed in Lynwin mine.

Section	A		B		C		D		E	
Laboratory No	14319		14320		14321		14322		14323	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	3	0	2	9	2	10	2	8	2	9
Shale and bone	[a] 0	4	[a] 0	5	[a] 0	5	[a] 0	7	[a] 0	6½
Coal	2	0	1	7	1	11	1	11	2	0
Floor, shale.										
Thickness of bed	5	4	4	9	5	2	5	2	5	3½
Thickness of coal sampled	5	0	4	4	4	9	4	7	4	9½

[a] Not included in sample.

Section A (sample 14319) was measured at the break-through to the air course at left entry 6, about 2,600 feet from the mine mouth.

Section B (sample 14320) was measured at the face of right heading 6, about 2,800 feet from the mine mouth.

Section C (sample 14321) was measured at the face of room 2, right heading 8, about 3,300 feet from the mine mouth.

Section D (sample 14322) was measured at the face of right heading 8, 3,400 feet from the mine mouth.

Section E (sample 14323) was measured at the face of the main heading, 3,200 feet from the mine mouth.

A composite sample was made by mixing face samples 14319 to 14323, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 14324.

Notes.—The Lynwin mine is opened by a drift entry and the coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by hand and broken down with FFF black powder and a permissible explosive. All of the coal was shipped as run-of-mine. The screening plant had bar screens 6 by 12 feet with 3-inch openings. Pickers were employed on the car in loading. The run-of-mine coal contained large lumps and presented a good appearance on the cars. There were two loading tracks with a capacity for 22 empty and 22 loaded cars. There was approximately 600 acres of unmined coal tributary to this opening. A large percentage of the coal was obtained from advance work. Daily output, 450 tons, with a capacity of 550 tons.

WINDING GULF. WINDING GULF NO. 1 AND NO. 2 MINES.

Sample.—Semibituminous coal; New River field; analyses Nos. 14368, 14369, 14370, 14371, 14372, 14373, and 14374 (p. 131).

Mine.—Winding Gulf No. 1; Kanawha-New River district; at Winding Gulf on the Winding Gulf branch of the Virginian Railway. No. 2 mine is 1 mile from Winding Gulf on Winding Gulf branch of the Chesapeake & Ohio Railway.

Coal bed.—Beckley. Carboniferous (Pottsville) age, Quinnimont formation. Average thickness, 4 feet 6 inches. Dip about 2° northwest. Roof, smooth gray shale; floor, hard fire clay. Material from the roof and floor does not become mixed with the coal in mining. Cover at points of sampling, 60 to 400 feet.

The coal bed was measured and sampled at three points in No. 1 mine and at two points in No. 2 mine by D. J. Price July 9, 1912, as described below:

Sections of coal bed in Winding Gulf No. 1 mine.

Section	A		B		C	
Laboratory No.	14368		14369		14370	
Roof, shale	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	2	11	2	10	2	11
Coal, bony	[a] 0	3	[a] 0	3	[a] 0	4
Coal	1	2	1	3	1	6
Floor, fire clay.						
Thickness of bed	4	4	4	4	4	9
Thickness of coal sampled	4	1	4	1	4	5

[a] Not included in sample.

Section A (sample 14368) was measured at the face of room 14, off left entry 1, off the main right heading.

Section B (sample 14369) was measured at the face of the air course of the west entry.

Section C (sample 14370) was measured at the face of the main right heading.

A composite sample was made by mixing face samples 14368, 14369, and 14370. The results of an ultimate analysis of this sample are given under laboratory No. 14371.

Sections of coal bed in Winding Gulf No. 2 mine.

Section	A		B	
Laboratory No.	14372		14373	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	3	0	3	2
Coal, bone	[a] 0	4	[a] 0	3
Coal	0	9	1	6
Floor, fire clay.				
Thickness of bed	4	1	4	11
Thickness of coal sampled	3	9	4	8

[a] Not included in sample.

Section A (sample 14372) was measured on the pillar at the break-through to right heading 1.

Section B (sample 14373) was measured at the face of left heading 2.

A composite sample was made by mixing samples 14372 and 14373. The results of an ultimate analysis of this sample are given under laboratory No. 14374.

Notes.—Both of these mines were comparatively new at the time of sampling. Both mines are opened by a slope and the coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by electric machines and broken down with permissible explosives. All of the coal was shipped as run-of-mine. Pickers were employed on the cars in loading. The average daily output of No. 1 mine was 400 tons.

WOODBAY. McALPIN NO. 1 MINE.

Sample.—Semibituminous coal, New River field; analyses Nos. 14301, 14302, 14303, 14304, 14305, 14306, 14307, 14308, and 14309 (p. 131).

Mine.—McAlpin No. 1; Kanawha-New River district; a drift mine at Woodbay Station on the Winding Gulf branch of the Virginian Railway.

Coal bed.—Beckley. Carboniferous (Pottsville) age, Quinnimont formation. Average thickness, 5 feet; dip, 1½° northwest. Roof, shale 2 to 3 feet thick, containing streaks of coal, above the shale which is a cap rock; floor, a dark soft, smooth carbonaceous shale. Material from the roof and floor did not become mixed with the coal in mining. Cover at points of sampling, about 400 feet.

The bed was measured and sampled at eight points in the mine by J. J. Rutledge and D. J. Price, July 6, 1912, as described below:

Sections of coal bed in McAlpin No. 1 mine.

Section	A		B		C		D	
Laboratory No.	14301		14302		14303		14304	
	Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Roof, shale.								
Coal	3	1	2	7	2	3	2	9
Mother coal	0	⅛	..	..	..	..	0	¼
Coal	0	9⅝	1	3	1	4	0	11
Cannel coal	0	3	0	2	0	2	0	3
Coal	1	3	1	3	2	6	1	4
Floor, shale.								
Thickness of bed	5	4¾	5	3	6	3	5	3¼
Thickness of coal sampled	5	4¾	5	3	6	3	5	3¼

Sections of coal bed in McAlpin No. 1 mine.

Section	E		F		G		H	
Laboratory No.	14305		14306		14307		14308	
	Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Roof, shale.								
Coal	3	3	3	3	3	2	3	3
Coal, bony	[a]0	4	[a]0	4½	[a]0	3½	[a]0	4½
Coal	0	11½	0	8	0	10	0	9
Floor, shale.								
Thickness of bed	4	6½	4	3½	4	3½	4	4½
Thickness of coal sampled	4	2½	3	11	4	0	4	0

[a] Not included in sample.

Section A (sample 14301) was measured at the face of the second panel, off left heading 3, off second main entry.

Section B (sample 14302) was measured at the face of the second main air course.

Section C (sample 14303) was measured at the face of room 10, second panel, left entry 2.

Section D (sample 14304) was measured at the face of room 3, first panel, left entry 3.

Section E (sample 14305) was measured on the pillar of main heading 1, between right entries 3 and 4.

Section F (sample 14306) was measured at the face of the second panel, off right heading 2.

Section G (sample 14307) was measured at the face of the fourth panel, off right heading 2.

Section H (sample 14308) was measured on the pillar of the air course, near right entry 2.

A composite sample was made by mixing samples 14301 to 14308, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 14309.

Notes.—The McAlpin No. 1 Mine is opened by drift entries and the coal mined by a combination of the room-and-pillar and panel systems. At the time of sampling the coal was undercut by electric punchers and shot down with a permissible explosive. A permissible explosive was also used for brushing the roof. All of the coal was shipped as run-of-mine. The mine was equipped with a screening plant in which the bar screens had 1½-inch openings. The coal was hand-picked prior to loading on a conveying table about 170 feet long. The coal presented a good appearance when loaded on the cars. There were two loading tracks with a capacity for 31 empty and 36 loaded cars. All of the coal was obtained from advance work. Output, 1,000 tons day, with a maximum output of about 1,500 tons.

RANDOLPH COUNTY.

MILL CREEK. TOLBERT AND SPIKER MINE.

Sample.—Bituminous coal; analysis No. 13561 (p. 132).

Mine.—Tolbert and Spiker; Monongahela district; on Mill Creek, 6 miles south of the town of Mill Creek.

Coal bed.—Sewell (?). Carboniferous age, Pottsville formation. Average thickness about 4 feet; dip, 8° west.

The bed was measured and sampled at one point by E. R. Lloyd on March 19, 1912, as described below:

Section of coal in Tolbert and Spiker mine.

Laboratory No.	13561	
Roof, dark shale.	*Ft.*	*in.*
Coal	1	7
Bone	[a]0	5
Coal	2	0
Shale.		
Thickness of bed	4	0
Thickness of coal sampled	3	7

[a] Not included in sample.

The sample was collected about 100 feet north of the mouth of the mine about 40 feet from the outcrop, under about 75 feet of cover. Average output, about 2½ tons a day. The mine is located on the northeast face of Rich Mountain, at an elevation of about 2,750 feet.

SPRUCE. HOPKINS MINE.

Sample.—Bituminous coal; analysis No. 13601 (p. 132).

Mine.—Hopkins; Monongahela district; about 9 miles southwest of Cheat Bridge, on Cheat River lumber road of West Virginia Pulp & Paper Co., and about 5 miles from Spruce.

Coal bed.—Sewell (?). Carboniferous age, Pottsville formation. Thickness at point where sampled, 4 feet 7 inches clear coal, which is about the average thickness within the mine. Roof and floor, shale. The mine is near the top of Cheat Mountain at an elevation of over 4,000 feet. Sample was collected in the mine 1,075 feet southwest of the mine mouth by E. R. Lloyd on March 20, 1912.

WYOMING.

CONVERSE COUNTY.

Douglas. Outcrop.

Sample.—Subbituminous coal; Glenrock field; analysis No. 11442 (p. 132).

Mine.—Outcrop in sec. 33, T. 40 N., R. 72 W., 50 miles north of Douglas, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Lower Burned. Tertiary age, Fort Union formation. Thickness, 6 feet 5 inches. Roof, carbonaceous shale; floor, carbonaceous clay.

The bed was measured and sampled by V. H. Barnett on September 25, 1910, as described below:

Section of coal bed in outcrop, 50 miles north of Douglas.

Laboratory No	11442	
Roof, carbonaceous shale.	*Ft.*	*in.*
Coal with thin carbonaceous lenses	[a]0	10
Coal	5	7
Floor, shale.		
Thickness of bed	6	5
Thickness of coal sampled	5	7

[a] Not included in sample.

Glenrock. Country Bank.

Sample.—Subbituminous coal; Glenrock field; analysis No. 11048 (p. 132).

Mine.—Country bank in sec. 5, T. 37 N., R. 75 W., 20 miles north of Glenrock, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Not named. Tertiary age, Fort Union formation. Thickness, 12 feet 8 inches; sandstone cap rock.

The bed was measured and sampled by C. H. Wegemann on September 29, 1910, as described below:

Section of coal bed in country bank, 20 miles north of Glenrock.

Laboratory No	11048	
Roof, sandstone.	*Ft.*	*in.*
Coal	5	0
Coal, bony	[a]1	2
Bone	[a]0	2
Coal, clean	[a]3	10
Coal with 2 bone partings	[a]2	6
Floor, not stated.		
Thickness of bed	12	8
Thickness of coal sampled	5	0

[a] Not included in sample.

Note.—The coal was mined intermittently from an open pit.

Inez. Diamond Mine.

Sample.—Subbituminous coal; Glenrock field; analysis No. 14781 (p. 132).

Mine.—Diamond; Inez district; a slope mine in sec. 16, T. 33 N., R. 73 W., 1½ miles south of Inez, and of the Chicago & North Western Railway.

Coal bed.—Not named. Tertiary age, Fort Union formation. Thickness, 4½ to 5 feet; roof, fair shale; floor, good shale; dip, 17° NE.

The bed was measured and sampled on September 24, 1912, by V. H. Barnett, as described below:

Section of coal bed in Diamond mine.

Laboratory No.	14781	
Roof, shale.	*Ft.*	*in.*
Coal	4	3
Clay	0	1
Coal	0	4
Floor, shale.		
Thickness of bed	4	8
Thickness of coal sampled	4	8

The sample was obtained at the face of an entry 450 feet from the mine mouth.

INEZ. INEZ MINE.

Sample.—Subbituminous coal; Glenrock field; analysis No. 14754 (p. 132).

Mine.—Inez; Inez district; a drift mine in the NW. ¼ sec. 16, T. 33 N., R. 72 W., 1 mile south of Inez and of the Chicago & North Western Railway.

Coal bed.—Not named. Tertiary age, Fort Union formation. Average thickness, about 5 feet; dip, 17° NE.

The bed was measured and sampled by F. A. Herald on September 19, 1912, as described below:

Section of coal bed in Inez mine.

Laboratory No.	14754	
Roof, brown, carbonaceous shale.	*Ft.*	*in.*
Coal	5	3
Shale, carbonaceous, brown	[a] 0	5
Shale, gray	[a] 0	11
Shale, brown, carbonaceous	[a] 0	6
Coal	0	7
Floor, brown, carbonaceous shale.		
Thickness of bed	7	8
Thickness of coal sampled	5	10

[a] Not included in sample.

The sample was taken at the end of the entry, 620 feet northwest of the mouth of the slope.

LOST SPRING. FRED HAYNES PROSPECT.

Sample.—Subbituminous coal; Lost Spring field; analysis No. 10711 (p. 132).

Prospect.—Fred Haynes, a prospect opening in sec. 11, T. 32 N., R. 69 W., 9 miles west of Lost Spring, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Unnamed. Tertiary age, Fort Union formation. Thickness, 4 feet 2 inches. Roof, shale; floor, shale.

The coal bed was sampled at one point in the entry by D. E. Winchester on August 1, 1910. The sample was taken at the face of the main entry, about 50 feet from the mouth of the entry.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 471, p. 491.

LOST SPRING. HARNEY CREEK MINE.

Sample.—Lignite verging on subbituminous coal; Lost Spring field; analysis No. 10911 (p. 132).

Mine.—Harney Creek, a drift mine in sec. 13, T. 34 N., R. 67 W., 17 miles northeast of Lost Spring, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Unnamed. Tertiary age, Fort Union formation. Thickness, 5 feet 6 inches. Roof, sandstone; floor, sandstone. The coal bed was sampled at one point in this mine by D. E. Winchester, September 10, 1910, as described below:

Section of coal bed in Harney Creek mine.

Laboratory No	10911	
Roof, sandstone.	*Ft.*	*in.*
Coal	2	10
Shale, carbonaceous	[a]0	1
Coal	2	1
Shale, carbonaceous	[a]0	6
Floor, sandstone.		
Thickness of bed	5	6
Thickness of coal sampled	4	11

[a] Not included in sample.

The above sample was taken at the end of the main entry, 80 feet from the main mouth.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 471, p. 505.

Lost Spring. Onyon Mine.

Sample.—Subbituminous coal; Lost Spring field; analysis No. 10740 (p. 133).

Mine.—Onyon, a drift mine in sec. 36, T. 33 N., R. 68 W., about 2 miles northwest of Lost Spring, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Unnamed. Tertiary age, Fort Union formation. Thickness, 5 feet 9 inches; roof, shale; floor, shale.

The bed was sampled at one point in the mine by D. E. Winchester on August 4, 1910. The sample was taken in room 15 off north entry 1, 1,800 feet from the mine mouth.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 471, p. 495.

Lost Spring. Prospect.

Sample.—Subbituminous coal; Lost Spring field; analysis No. 11447 (p. 133).

Prospect.—Prospect in sec. 8, T. 37 N., R. 68 W., 35 miles north of Lost Spring, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—D. Tertiary age, Fort Union formation. Thickness, 4 feet 4 inches; cap, sandstone; floor, shale.

The bed was sampled by D. E. Winchester on October 21, 1910. The bed had been opened as a stripping pit. Some coal had been taken from the prospect during previous winters.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 471, p. 513.

Lost Spring. Rosin Mine.

Sample.—Subbituminous coal; Lost Spring field; analysis No. 10775 (p. 133).

Mine.—Rosin, a drift mine in sec. 35, T. 34 N., R. 68 W., 8 miles north of Lost Spring, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Unnamed. Tertiary age, Fort Union formation. Thickness, 5 feet 1 inch; roof, shale; floor, shale.

The bed was measured and sampled by D. E. Winchester on August 17, 1910, as described below:

Section of coal bed in Rosin mine.

Laboratory No	10775	
Roof, shale.	*Ft.*	*in.*
Coal, soft and bright	1	1
Shale, soft, carbonaceous	[a]0	3
Coal, bright and hard	1	8
Shale, carbonaceous	[a]0	1
Coal, bright and hard	2	0
Floor, shale.		
Thickness of bed	5	1
Thickness of coal sampled	4	9

[a] Not included in sample.

The sample was taken at the face of left entry 1 off main entry, about 1,000 feet from the mine mouth.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 471, p. 501.

LOST SPRING. SUNSET MINE.

Sample.—Subbituminous coal; Lost Spring field; analysis No. 10835 (p. 133).

Mine.—Sunset, a drift mine in sec. 35, T. 34 N., R. 68 W., 8 miles northwest of Lost Spring, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Unnamed. Tertiary age, Fort Union formation. Average thickness about 5 feet; roof, bony coal; floor, shale.

The coal bed was sampled by D. E. Winchester August 20, 1910, as described below:

Section of coal bed in Sunset mine.

Laboratory No	10835	
Roof, bony coal.	*Ft.*	*in.*
Coal	0	4
Bony coal	[a]1	4
Coal	0	6
Bony coal	[a]0	½
Coal	0	7
Shale	[a]0	3
Coal	2	3
Floor, shale.		
Thickness of bed	5	3½
Thickness of coal sampled	3	8

[a] Not included in sample.

This sample was taken at the face of the main entry, 500 feet from the mouth.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 471, p. 502.

CROOK COUNTY.

SUNDANCE. BELSHE MINE.

Sample.—Bituminous coal; Sundance field; analysis Nos. 15713, 15714, and 15715 (p. 133).

Mine.—Belshe; SE. ¼ NW. ¼ sec. 24, T. 51 N., R. 64 W., 7½ miles west of Sundance.

Coal bed.—No name. Lower Cretaceous age, Lakota formation. The bed is nearly horizontal and is variable in thickness and character. Both roof and floor are shale.

The bed was measured and sampled by M. W. Ball on January 2 and 3, 1913, as described below:

Sections of coal bed in Belshe mine.

Section	A		B		C		D	
Laboratory No.	15713				15714 15715			
Roof, section A, black, carbonaceous shale with some coal; section B, black, "slaty" shale; section C, dark-gray shale, "slaty" immediately above coal.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal, somewhat dirty, pyritiferous	0	2	..	..	..	..	..	..
Coal, bright, clean	0	3¾	0	6½	0	2	0	2
Coal, rather dull, but clean	0	2	..	..	..	..	..	..
Coal and pyrite	..	..	0	½	..	..	..	..
Coal, slightly dull, with half inch streaks of lustrous coal	..	..	..	..	0	11¾	..	..
Coal, bright, with a few sulphurous streaks	..	..	..	..	1	1	..	..
Coal, dull	..	..	..	..	0	2	..	..
Coal, bright	1	1	0	5½	0	7	..	..
Coal, moderately bright, rather hard	2	2	..	..	..	..	..	..
Splint and good coal, intermixed	[a] 0	2¼	..	..	..	..	..	..
Coal, rather dull	..	..	0	7½	..	..	..	..
Coal, bright	..	..	1	4	..	..	..	..
Coal, rather dull and hard	..	..	0	6½	..	..	..	..
Coal, good	..	..	..	..	..	..	3	2
Splint	[a] 0	2¼	0	5½	[b] 0	4½	0	2½
Coal, moderately bright, with a little pyrite	1	0	..	..	..	..	..	..
Coal	..	..	..	..	[b] 0	3	0	7
Splint	..	..	..	..	[b] 0	2	0	1½
Coal, bright	..	..	0	3	0	6	..	..
Coal, somewhat dull	..	..	0	4	..	..	..	..
Coal	..	..	0	3½	..	..	0	3½
Floor, section A, black shale; section B, "slaty" shale, with some coal; section C, "slaty" shale; section D, dark-gray shale.								
Thickness of bed	5	3¼	4	10½	4	3¼	4	6½
Thickness of coal sampled	4	10¾	4	10½	([c])		4	6½

[a] Not included in sample.
[b] Not included in sample 15714 but included in sample 15715.
[c] Sample 15714, 3 feet 5¾ inches; sample 15715, 4 feet 3¼ inches.

Section A (sample No. 15713) was taken at end of room turned east from back entry, 275 feet from mouth of mine.

Section B was taken at the end of the back entry.

Section C (samples Nos. 15714 and 15715) was taken in the last crosscut, 48 feet from the front entry.

Section D was taken at the end of the front entry.

FREMONT COUNTY.

HUDSON. HICKEY MINE.

Sample.—Subbituminous coal; Hudson field of the Wind River region; analysis No. 17247 (p. 133).

Mine.—Hickey, a slope mine in sec. 11, T. 1 S., R. 2 E., 1 mile west of Hudson and one-fourth of a mile from the main line of the Chicago & North Western Railway.

Coal bed.—Unnamed. Cretaceous age, Mesaverde formation. Average thickness about 3 feet 8 inches; dip 13° E., 15° N. Faults, rolls, and horsebacks are uncommon. Roof, hard gray shale, above which is sandstone cap rock; floor, hard smooth shale; particles of the roof and floor do not get mixed with the coal in mining. The cover at the point where sample was taken is about 100 feet.

The coal bed was measured and sampled by J. C. Roberts on May 16, 1913. The sample included a 3½ foot cut of coal. The roof was shale, over which was sandstone; the floor was shale.

The sample was taken in room 3, off the south entry, 300 feet from the mouth of the mine.

Notes.—The Hickey mine is opened by a slope and the coal is mined by the room-and-pillar system. The coal was undercut by hand and broken down with black blasting powder. The coal cuttings were loaded with the coal. The screening equipment consisted of bar screens 12 feet long with 1½-inch and ½-inch openings. No men were employed to pick shale out of the cars at the tipple. Eight men were employed at this mine.

For a description of the geologic relation of this bed see U. S. Geol. Survey Bull. 471, p. 545.

HUDSON. INDIAN MINE.

Sample.—Subbituminous coal; Hudson field; analyses Nos. 10838, 10839, 10840, 10841, 10842, 10843, 13178, and 13179 (p. 133).

Mine.—Indian; slope mine near Hudson, in the NE. ¼ sec. 2, T. 2 S., R. 2 E., in the Shoshone Indian Reservation, on a spur of the Wyoming & North Western Railway.

Coal bed.—The bed has not been designated by name, because of the lenticular nature of the beds in this region. Cretaceous age; Mesaverde formation. Thickness in the mine, variable; dip, 13° S. where sample 13178 was taken, and 18° S. where sample 13179 was taken.

The bed was measured and sampled at five points on September 3, 1910, by H. M. Wolflin, and at two points on December 11, 1911, by Max A. Pishel, as shown below:

Sections of coal bed in Indian mine.

Section	A		B		C		D		E	
Laboratory No	10838		10839		10840		10841		10843	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
"Draw slate"	..	..	[a]0	10	[a]0	8	[a]0	8	..	..
Clean coal	7	7½	3	10	..	..	3	1½	..	..
Slaty coal	..	..	[a]0	6½	..	..	..	..	..	..
Bony coal	..	..	..	..	0	7	0	2½	..	..
Clean coal	..	..	2	5	6	7	3	2	7	6
Floor, clay.										
Thickness of bed	7	7½	7	7½	7	10	7	2	7	6
Thickness of coal sampled	7	7½	6	3	7	2	6	6	7	6

Section	F		G	
Laboratory No.	13178		13179	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	6	..	..
Coal, with bony streaks	..	..	0	10
Bone	[a]0	4	[a]0	4
Coal	..	..	2	0
Shale, bony	[a]1	2	[a]1	0
Coal	1	6	0	8
Bone	[a]0	4	[a]0	2
Shale, bony	[a]1	2	..	..
Coal	0	6	2	0
Floor, sandstone.				
Thickness of bed	6	6	7	0
Thickness of coal sampled	3	6	5	6

[a] Not included in sample.

Section A (sample 10838) was measured at the face of room 17, off the 00 entry.

Section B (sample 10839) was measured at the face of west entry 4.

Section C (sample 10840) was measured in a room (about No. 30), off west entry 2.

Section D (sample 10841) was measured at the face of room 4, off east entry 3.

Section E (sample 10843) was measured at the face of room 14½, off west entry 2.

Section F (sample 13178) was measured in the fifth west back entry, one-fourth of a mile southwest of the mine mouth.

Section G (sample 13179) was taken in the main entry, 900 feet from the mine mouth.

A composite sample was made by mixing samples 10838, 10839, 10840, and 10841. The results of an ultimate analysis of this sample are given under laboratory No. 10842.

Notes.—At the time of sampling in 1911 the mine had been abandoned several months and a number of men were engaged in removing the hoisting engine. It seemed probable that in the future the mine would be worked on a small scale, as there was still considerable coal left in the mine as pillars. The probable lifetime of the mine was 10 years.

HUDSON. MITCHELL MINE.

Sample.—Subbituminous coal; Hudson field; analysis No. 17248 (p. 133).

Mine.—Mitchell, a slope mine in the Shoshone Indian Reservation, in the SW. ¼ sec. 22, T. 1 S., R. 2 E., 5 miles from Hudson and 3½ miles from the Indian mine spur off the Chicago & North Western Railway.

Coal bed.—The bed has not been designated by name, chiefly because most of the beds in this region are lenticular. Cretaceous age, Mesaverde formation. Average thickness, 3 feet; dip, 20° S. 40° E. No cleats, rolls, or horsebacks have been encountered. Roof, excellent shale, above which is sandstone; floor, hard, smooth shale. The cover at the point where the sample was taken is about 100 feet thick.

The coal bed was measured and sampled at one point in the mine by J. C. Roberts, on May 16, 1913, as described below:

Section of coal bed in Mitchell mine.

Laboratory No.	17248	
Roof, sandstone over shale.	*Ft.*	*in.*
Coal	1	2
Shale parting	[a] 0	1½
Coal	2	½
Floor, shale.		
Thickness of bed	3	4
Thickness of coal sampled	3	2½

[a] Not included in sample.

The sample was taken in the last room off the main slope, 40 feet from the slope and 500 feet from the mouth of the slope.

Notes.—The Mitchell mine is opened by a slope which is 5,050 feet above sea level. Although the long-wall method of mining was started, the workings at the time of sampling consisted mostly of rooms and gangways. The screening plant consisted of bar screens with 1-inch openings. There were no railway connections and the coal was transported by wagons. The mine was operated only when the operator had an order for coal for local consumption. All workings were advance.

HUDSON. PROPOSIA MINE.

Sample.—Lignite; analyses Nos. 17244, 17245, and 17246 (p. 133).

Mine.—Proposia, a slope mine 4 miles from Hudson on a spur of the Chicago & North Western Railway.

Lignite bed.—Lander. Cretaceous age, Mesaverde formation. Average thickness, 12 feet, ranging from 8 to 17 feet; dip, 14° N. 17° E. There are no cleats, faults, rolls, or horsebacks in the bed. Roof, shale with a sandstone cap rock; floor, hard smooth shale. Particles of the roof and floor do not mix with the coal in mining. The roof at the points where the samples were taken varies from 100 to 300 feet in thickness.

The lignite bed was measured and sampled at two points by J. C. Roberts on May 18, 1913, as described below:

Sections of lignite bed in Proposia mine.

Section	A		B	
Laboratory No.	17244		17245	
Roof, shale under sandstone.	*Ft.*	*in.*	*Ft.*	*in.*
Clean coal	[a] 7	6	..	..
Do.	9	6	8	0
Floor, shale.				
Thickness of bed	17	0	8	0
Thickness of coal sampled	9	6	8	0

[a] Not included in sample.

Section A (sample 17244) was measured at the face of the second north entry, 1,800 feet from the mine mouth.

Section B (sample 17245) was measured at the face of the fifth south entry, 1,200 feet from the mine mouth.

A composite sample was made by mixing samples 17244 and 17245. The results of an ultimate analysis of this sample are given under laboratory No. 17246.

Notes.—The Proposia mine is opened by a slope. The coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by hand and shot down with FF black blasting powder. The cuttings were loaded with the coal. None of the coal was shipped as run-of-mine. The screening equipment consisted of shaking screens 32 feet long by 8 feet wide; diameter of holes ½, 2, 4, 5, 6, and 8 inches. Twenty-five per cent of the coal was lump; 30 per cent grate; 33 per cent nut, and 12 per cent slack. One picker was employed on the screens. The lump coal was large and had an excellent appearance on the cars. There were three loading tracks with a capacity for 50 empty cars and 50 loaded cars. There was approximately 600 acres of unmined coal to be taken out. The mine had an estimated life of 10 years. About 75 per cent of the future tonnage was to be derived from advance work and 25 per cent from pillars. The daily output at the time of sampling (May, 1913) was 1,100 tons, with a maximum day's run of 1,300 tons.

HOT SPRINGS COUNTY.

CROSBY. BIG HORN MINE.

Sample.—Subbituminous coal; analyses Nos. 15201 to 15212, 15214 to 15219, 15234 to 15237 (pp. 134–136).

Mine.—Big Horn, a slope mine at Crosby, 2 miles from Kirby, on the Chicago, Burlington & Quincy Railroad.

Coal bed.—Gebo. Cretaceous age, Laramie formation. Thickness of coal, 5 feet 6 inches to 8 feet 6 inches, not including the top coal, which is about 2½ feet thick and is left up for roof; dip, 17° N. 15° E. No faults, rolls, or horsebacks are encountered. Roof, clay and shale with a sandstone cap rock; floor smooth, medium-hard, shaly clay. Neither roof nor floor particles mixed with the coal in mining. Machine cuttings were loaded with the coal. The cover at the points where the samples were taken varies from 85 to 300 feet in thickness.

The coal bed was measured and sampled at five points in the mine and four samples of slack coal were taken, one in the mine and three from wagons, by O. U. Bradley, December 6 to 17, 1912, as described below:

Sections of coal bed in Big Horn mine.

Section	A		B		C		D		E	
Laboratory No.	15201 15202		15209 15210		15214 15215		15216 15217		15218 15219	
Roof, clay and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	..	..	1	7	..	..	..	..	..	..
Bony coal	..	..	0	1	..	..	..	..	..	..
Coal	7	6	1	0	8	1	2	4	5	3
Bony coal	..	..	[a]0	3	..	..	[a]0	3	..	..
Coal	..	..	5	6	..	..	5	11	..	..
Floor, underclay.										
Thickness of bed	7	6	8	5	8	1	8	6	5	3
Thickness of bed sampled	7	6	8	2	8	1	8	3	5	3

[a] Not included in sample.

Section A (duplicate samples 15201 and 15202) was measured on the east side of room 5, off butt entry 3 off west entry 1.

Section B (duplicate samples 15209 and 15210) was cut from east side of room 5, 50 feet from second butt dip entry off west entry 2.

Section C (duplicate samples 15214 and 15215) was cut from a pillar between rooms 6 and 7, off butt entry 2 off west entry 1.

Section D (duplicate samples 15216 and 15217) was cut from the face of room 6, off first dip butt entry off west entry 2.

Section E (duplicate samples 15218 and 15219) was cut from the face of room 9, off east entry 2, 45 feet from the entry.

A composite sample was made by mixing samples 15205, 15207, and 15211. The results of an ultimate analysis of this sample are shown under laboratory No. 15234.

A composite sample was made by mixing samples 15206, 15208, and 15212. The results of an ultimate analysis of this sample are shown under laboratory No. 15235.

A composite sample was made by mixing samples 15201, 15210, 15216, and 15218- The results of an ultimate analysis of this sample are shown under laboratory No. 15236.

A composite sample was made by mixing samples 15202, 15209, 15217, and 15219. The results of an ultimate analysis of this sample are shown under laboratory No. 15237.

Duplicate samples 15203 and 15204 represent slack coal taken from the mine.

Duplicate samples 15205 and 15206 represent slack from 17 wagonloads averaging 3,400 pounds each.

Duplicate samples 15207 and 15208 represent slack coal taken from 20 wagon-loads averaging 3,400 pounds each.

Duplicate samples 15211 and 15212 represent slack coal taken from 18 wagon-loads averaging 3,400 pounds each.

Notes.—The Big Horn mine is opened by a slope. The coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by three electric coal-cutting machines and four air-punching machines. It was shot down with black blasting powder. Twenty-six per cent of the output passed through the screens. Grades of coal produced were: Lump coal, that passing 3, 4, and 8 inch openings; egg nut, that passing 4-inch to 1¼-inch openings; nut, that passing 3-inch to 1¼ inch openings; pea, that passing 1¼-inch to ¾-inch openings. About 70 per cent of the coal was shipped under the first three classifications. There was little demand for the pea coal and the excess was thrown in with the slack and run out to the dump, except a small amount burned in the company's boilers. The lumps were large and had a good appearance. There were four loading tracks with a capacity of 15 empty and 15 loaded cars. There was approximately 490 acres to be taken out from the opening. The daily average output was 800 tons, with a maximum day's run of 815 tons.

KIRBY. GEBO MINE.

Sample.—Subbituminous coal; Bighorn Basin field; analyses Nos. 15277, 15278, 15279, 15280, 15281, 15282, 15283, 15342, 15343, 15344, 15345, 135346, 15347, 15348, 15349, 15475, 15482, 15687, and 16102 (pp. 136, 137).

Mine.—Gebo, a slope mine 2 miles southwest of Kirby in the S. ½ sec. 11, T. 44 N., R. 95 W., on a spur of the Chicago, Burlington & Quincy Railroad.

Coal bed.—Gebo. Cretaceous age, Eagle sandstone. Average thickness 10 feet 8 inches, dip 20½° N. 24° E.; cleat N. 57° E. No faults, rolls or horsebacks are encountered. Roof, clay and shale, with a sandstone cap rock; floor, hard and smooth sandstone. Particles of the roof and floor seldom mix with the coal in mining. Machine cuttings were loaded with the coal. The cover at the points where the samples were taken varies from 100 to 410 feet in thickness.

The coal bed was measured and sampled at 10 points in the mine, and 12 samples of slack were taken as it was being loaded on the car by O. U. Bradley, December 13 to 27, 1912, and 2 samples were taken from the cars of slack at Pittsburgh, Pa., by E. R. Linkenhoker on January 15 and 31, 1913, as described below:

Section of coal bed in Gebo mine.

Section	A	
Laboratory No	15277	
Roof, top coal, clay, and shale.	*Ft.*	*in.*
Clean coal	7	2
Coal, slightly bony	0	2
Coal	0	6
Floor, clay.		
Thickness of bed	7	10
Thickness of coal sampled	7	10

Section A (sample 15277) was measured at the face of the fourth east entry, 225 feet from the second butt entry.

Sections B, C, D, E, F, G, H, I, and J were cut as described below. Sections B to E had a roof of top coal, clay, and shale, and a floor of clay; sections F to J had a roof of top coal, clay, "slate," and sandstone, and a floor of clay.

Section B (sample 15278) represented an 8-foot 9-inch cut of coal. It was measured at the face of room 8 east, second butt entry off the fifth entry.

Section C (sample 15279) represented a 9-foot 8-inch cut of coal. It was measured at the face of room 7 east, off first butt entry off the fifth level.

Section D (sample 15280) represented a 9-foot 2-inch cut of coal. It was measured at the face of the third east entry, 400 feet from the second butt entry.

Section E (sample 15281) represented a 7-foot 11-inch cut of coal. It was measured at the face of the third west entry, 300 feet from the main slope.

Section F (sample 15282) represented a 6-foot 9-inch cut of coal. It was measured on a pillar between room 1 west and the back entry off second level.

Section G (duplicate samples 15342 and 15343) represented a 5-foot 9-inch cut of coal. It was measured on a pillar between the second west entry and the back entry, 375 feet beyond the third butt entry, 1,750 feet from main slope.

Section H (duplicate samples 15344 and 15345) represented a 6-foot 1-inch cut of coal. It was measured on a pillar between rooms 1 and 2 west, off the first butt entry off the second east entry, about 400 feet from main slope.

Section I (duplicate samples 15346 and 15347) represented a 6-foot 5-inch cut of coal. It was measured on a pillar between pitch rooms 10 and 11, off the first west entry.

Section J (duplicate samples 15348 and 15349) represented a 6-foot 7-inch cut of coal. It was measured near the face on a pillar between the first west entry and the back entry, 2,100 feet from the slope.

A composite sample was made by mixing samples 15277, 15278, 15279, and 15280. The results of ultimate analysis of this sample are shown under laboratory No. 15283.

A composite sample was made by mixing six separate samples taken from the slack as it was being loaded into the cars for shipment. The results of an ultimate analysis of this sample are given under laboratory No. 15475.

A composite sample was made by mixing six samples taken from the slack as it was being loaded into the cars for shipment. The results of an ultimate analysis of this sample are given under laboratory No. 15482.

Samples were collected from each of two cars of slack shipped to Bureau of Mines, Pittsburgh, Pa., for a test as to most efficient utilization. For the results of analysis, see laboratory Nos. 15687 and 16102.

Notes.—The Gebo mine is opened by a drift. The coal is mined by the room-and-pillar system. At the time of sampling it was undercut by electric chain machines and compressed-air punching machines and was broken down with black blasting powder. The machine cuttings were loaded out with the coal. About 12 per cent of the coal was shipped as run-of-mine. The tipple was equipped with a shaking screen 42 by 7 feet, 15 feet having 1-inch holes, 4 feet, 1½-inch holes, 4 feet, 2½-inch holes, 4 feet, 5-inch holes, 4 feet, 8-inch holes, and a ⅞-inch screen for slack. No men were employed on the cars or belts to pick shale from the coal. The lumps being loaded were large and had a good appearance. There were four loading tracks with a capacity of 75 empty cars and 75 loaded cars. Approximately 1,000 acres of coal remained to be taken out. The mine was expected to have a lifetime of 50 years; future tonnage was to be derived almost wholly from advance workings. The daily output was 750 tons, with a maximum day's run of 1,026 tons. The daily output was to be increased to 1,000 tons or more.

MEETEETSE. DICKIE NO. 1 PROSPECT.

Sample.—Subbituminous coal; Bighorn Basin field; analysis No. 14679 (p. 137).

Mine.—Dickie No. 1; a prospect drift on the northwest side of Grass Creek Basin, in sec. 2, T. 46 N., R. 99 W., 18 miles southeast of Meeteetse.

Coal bed.—No name. Cretaceous age, Gebo formation (Eagle sandstone). Dip, 16° N.

The bed was measured and sampled by D. F. Hewett, August 23, 1912, as described below:

Section of coal bed in Dickie No. 1 prospect.

Laboratory No.	14679	
Roof, brown shale.	*Ft.*	*in.*
Coal	0	1
Shale	0	½
Coal	0	5
Coal, bony	0	2
Coal	2	6
Shale	0	½
Coal	0	9
Floor, shale.		
Thickness of bed	4	0
Thickness of coal sampled	4	0

The sample was taken from the east rib of the western of two short drifts at a point 25 feet from the mouth. As the mine had been idle since 1910, the coal was weathered.

MEETEETSE. MAYFIELD NO. 2 PROSPECT.

Sample.—Subbituminous coal; Bighorn Basin field; analysis No. 14756 (p. 137).

Mine.—Mayfield No. 2. A prospect 2 miles southwest of Grass Creek Basin, in E. ½ sec. 26, T. 46 N., R. 99 W., 19 miles southeast of Meeteetse.

Coal bed.—No name. Tertiary age, Fort Union formation. Dip 5° SE.

The bed was measured and sampled by D. F. Hewett on August 24, 1912, as described below:

Section of coal bed in Mayfield No. 2 prospect.

Laboratory No.	14756	
Roof, gray shale.	*Ft.*	*in.*
Coal	[a] 24	0
Coal, bench sampled and mined	6	0
Floor, sandstone.		
Thickness of bed	30	0
Thickness of coal sampled	6	0

[a] Not included in sample.

The sample was taken at the face of the new drift, 30 feet from the mouth.

Notes.—This prospect was opened in 1911 on the west side of a ravine, 500 feet southwest of an old opening on the east side, where sample No. 5770 was collected by E. G. Woodruff in 1907.

Only the lower part of the bed was mined. The mine was operated only intermittently, but the sample represented unweathered coal.

JOHNSON COUNTY.

BUFFALO. PROSPECT.

Sample.—Subbituminous coal; Sussex field (Basin No. 2); analysis No. 10683 (p. 137).

Mine.—This sample was taken from a small prospect which was being operated by ranchers in sec. 17, T. 47 N., R. 81 W. The mine is 18 miles southeast of Buffalo and about 50 miles south of Regis. which is the nearest station on the Chicago, Burlington & Quincy Railroad

Coal bed.—B. Tertiary age, Fort Union formation. Thickness, about 11 feet; sandstone capping.

The bed was measured and sampled by C. H. Wegemann on July 20, 1910, as described below:

Section of coal bed in prospect, 18 miles southeast of Buffalo.

Laboratory No	10683	
Roof, white sandstone.	*Ft.*	*in.*
Coal	[a]0	6
Shale, gray	[a]2	0
Coal	[a]2	5
Shale	[a]0	1
Coal	3	4
Shale	[a]0	½
Coal	[a]2	5
Floor, not stated.		
Thickness of bed	10	9½
Thickness of coal sampled	3	4

[a] Not included in sample.

For a description of the geologic relations of this bed, see U. S. Geol. Survey Bull. 471, p. 454.

CASPER. PROSPECT.

Sample.—Subbituminous coal; Sussex field (Basin No. 3); analysis No. 10728 (p. 137).

Mine.—Prospect in sec. 23, T. 44 N., R. 81 W., about 75 miles north of Casper, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Unnamed. Tertiary age, Fort Union formation. Thickness, 6 feet 3 inches; cap rock, sandstone.

The coal bed was sampled by C. H. Wegemann on July 29, 1910, as described below:

Section of coal bed in prospect, 75 miles north of Casper.

Laboratory No	10728	
Roof, sandstone.	*Ft.*	*in.*
Shale	[a]0	6
Coal	3	7
Coal, bony	[a]0	2
Coal	[a]2	0
Floor, not stated.		
Thickness of bed	6	3
Thickness of coal sampled	3	7

[a] Not included in sample.

Note.—Coal had been mined at this prospect to some extent for local use.

For a description of the geologic relations of this coal bed, see U. S. Geol. Survey Bull. 471, p. 458.

CASPER. PROSPECT.

Sample.—Subbituminous coal; Sussex field (Basin No. 4); analysis No. 10804 (p. 137).

Mine.—Prospect in sec. 36, T. 43 N., R. 79 W., about 60 miles north of Casper, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Unnamed. Tertiary age, Fort Union formation. Thickness, about 14 feet. It was sampled by C. H. Wegemann, August 16, 1910, as described below:

Section of coal bed in prospect, about 60 miles north of Casper.

Laboratory No.	10804	
Roof, not stated.	*Ft.*	*in.*
Shale	[a]0	6
Coal	[a]0	8
Shale	[a]1	2
Coal	2	5
Coal, bony	[a]0	5
Coal	1	2
Shale	[a]0	6
Coal	[a]1	1
Shale	[a]1	0
Coal	[a]1	2
Coal, bony	[a]0	7
Coal	[a]2	3
Floor, not stated.		
Thickness of bed	12	11
Thickness of coal sampled	3	7

[a] Not included in sample.

For a description of the geologic relations of this bed, see U. S. Geol. Survey Bull. 471, p. 462.

CASPER. PUGGSLEY MINE.

Sample.—Subbituminous coal; Sussex field (Basin No. 4); analysis No. 10827 (p. 138).

Mine.—Puggsley, a mine in sec. 20, T. 42 N., R. 77 W., 65 miles north of Casper, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Unnamed. Tertiary age, Fort Union formation. Thickness, 20 feet 5 inches. It was sampled at one point in this mine by Carroll H. Wegemann, on August 26, 1910, as described below:

Section of coal bed in Puggsley mine.

Laboratory No.	10827	
Roof, not stated.	*Ft.*	*in.*
Coal	[a]0	10
Bone	[a]0	4
Coal	[a]3	0
Shale and coal	[a]1	10
Coal	5	0
Coal, bony	[a]0	3
Coal	[a]0	8
Bone	[a]0	7
Coal	[a]4	2
Shale	[a]0	3
Coal	[a]3	6
Floor, not stated.		
Thickness of bed	20	5
Thickness of coal sampled	5	0

[a] Not included in sample.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 471, 1912, p. 461.

CASPER. PROSPECT.

Sample.—Subbituminous coal; Sussex field; analysis No. 11059 (p. 138).

Mine.—Prospect in sec. 32, T. 41 N., R. 78 W., 50 miles north of Casper, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Unnamed. Tertiary age, Fort Union (?) formation. Bed, 3 feet 10 inches thick, with a sandstone cap rock 12 feet thick, and a sandstone floor. The coal bed was sampled by C. J. Hares on October 6, 1910, as described below:

Section of coal bed in prospect, 50 miles north of Casper.

Laboratory No	11059	
Roof, sandstone.	*Ft.*	*in.*
Coal	2	2
Sandstone	[a]0	8
Shale	[a]1	0
Floor sandstone.		
Thickness of bed	3	10
Thickness of coal sampled	2	2

[a] Not included in sample.

Note.—The coal had been mined to some extent for local use.

KAYCEE. ABANDONED MINE.

Sample.—Bituminous coal; Sussex field (Basin No. 3); analysis No. 10694 (p. 138).

Mine.—An abandoned mine in sec. 24, T. 44 N., R. 81 W., 6 miles northeast of Kaycee and about 50 miles north of Bucknum, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Unnamed. Cretaceous age, Fort Union formation.

The bed was sampled at one point in the mine by O. B. Hopkins. The sample represented the entire face section of the coal bed.

For a description of the geologic relations of this coal bed see U. S. Geol. Survey Bull. 471, 1912, p. 459.

NATRONA COUNTY

CASPER. RED ASH MINE.

Sample.—Subbituminous coal; Wind River field; analysis No. 10671 (p. 138).

Mine.—Red Ash, a drift mine in sec. 14, T. 31 N., R. 82 W., Platte River district, about 25 miles southwest of Casper, which is the nearest station on the Chicago & North Western Railway.

Coal bed.—Not named. Cretaceous age, Fort Union formation. Thickness, 6 feet 3 inches. The bed is overlain with 4 inches of carbonaceous shale capped by sandstone; floor, carbonaceous shale.

The bed was sampled at one point in the mine by D. E. Winchester on July 9, 1910. The sample was taken at the face of the second and last room off the main entry, about 300 feet from the mine mouth.

Note.—The mine had been worked periodically to supply local use, but was idle at the time of sampling.

PARK COUNTY.

CODY. McGUFFEY MINE.

Sample.—Subbituminous coal; Bighorn Basin field; analysis No. 12886 (p. 138).

Mine.—McGuffey; a slope mine on the north side of Oregon Basin, in sec. 17, T. 52 N., R. 100 W., 9 miles southeast of Cody.

Coal bed.—Unnamed. Cretaceous age; Gebo formation (Eagle sandstone). Dip, 14° NE.

The bed was measured and sampled by D. F. Hewett on August 18, 1911, as shown below:

Section of coal bed in the McGuffey mine.

Laboratory No.	12886	
Roof, blue shale.	*Ft.*	*in.*
Coal	0	5
Coal, high in pyrite	0	1
Coal	0	7
Shale	[a]0	5
Coal	1	7
Floor, shale.		
Thickness of bed	3	1
Thickness of coal sampled	2	8

[a] Not included in sample.

The sample was taken from the east rib of a room 20 feet above entry 3, at a point 70 feet east of the slope. The coal was unweathered.

Note.—The mine had been opened in 1910 and operated throughout the year.

MEETEETSE. BLACK DIAMOND MINE.

Sample.—Subbituminous coal; Bighorn Basin field; analysis No. 12887 (p. 138).

Mine.—Black Diamond; a slope mine in sec. 21, T. 49 N., R. 100 W., 4 miles north of Meeteetse.

Coal bed.—No name. Tertiary age, Fort Union formation. Thickness, 42 inches; roof, sandstone overlain with shale; floor, shale; dip, 18° NE.

The bed was measured and sampled by D. F. Hewett on September 11, 1911. The sample represented 3½ feet of coal, including the entire bed.

The sample was taken from the breast of a room 30 feet above entry 6, at a point 200 feet west of the slope.

Note.—This is the same mine sampled by E. G. Woodruff in 1907 (No. 5768). Under No. 5768 the location is incorrectly given as sec. 28, T. 49 N., R. 100 W. The mine was operated throughout the year.

MEETEETSE. GREYBULL OR WILSON MINE.

Sample.—Subbituminous coal; Bighorn Basin field; analysis No. 12888 (p. 138).

Mine.—Greybull or Wilson; a slope mine in the SE. ¼ sec. 13, T. 48 N., R. 101 W., 3½ miles southwest of Meeteetse and 40 miles south of Cody.

Coal bed.—No name. Cretaceous age, Gebo formation (Eagle sandstone). Dip, 5° NE.; thickness, 5½ to 6 feet.

The bed was measured and sampled by D. F. Hewett on September 12, 1911, as shown below:

Section of coal bed in Greybull or Wilson mine.

Laboratory No.	12888	
Roof, sandstone.	*Ft.*	*in.*
Coal	[a]0	6
Shale	[a]0	3
Coal	2	1
Shale	[a]0	4
Coal	2	5
Floor, shale.		
Thickness of bed	5	7
Thickness of coal sampled	4	6

[a] Not included in sample.

The sample was taken from the face of a room 200 feet northwest of the slope, at a point 400 feet from the mine mouth.

Notes.—This mine lies on the north side of Greybull River and adjoins the Erskine mine, sampled by E. G. Woodruff in 1907 (sample No. 5769). The Erskine mine and the Wilson mine operate the same bed, though the coal sections in the respective mines differ considerably. The mine was operated throughout the year.

WILEY. EAGLE MINE.

Sample.—Subbituminous coal; Bighorn Basin field; analysis No. 12885 (p. 139).

Mine.—Eagle. A slope mine east of Oregon Basin, in the SW. ¼ sec. 2, T. 51 N. R. 100 W., 2 miles northeast of Wiley, 13 miles southeast of Cody.

Coal bed.—No name. Cretaceous age, Meeteetse formation. Dip, 34° E.

The bed was measured and sampled by D. F. Hewett on August 16, 1911, as shown below:

Section of coal bed in Eagle mine.

Laboratory No.	12885	
Roof, soft gray shale.	*Ft.*	*in.*
Coal	1	4
Coal, bony	1	0
Shale	[a]0	1
Coal	0	11
Shale	[a]0	2
Coal	2	2
Floor, brown shale.		
Thickness of bed	5	8
Thickness of coal sampled	5	5

[a] Not included in sample.

The sample was take from the west rib of the north entry, 6 feet from the bottom of the shaft. The coal was unweathered.

Note.—The mine had been idle since 1910.

WILEY. EAST WILEY MINE.

Sample.—Subbituminous coal; Bighorn Basin field; analysis No. 12884 (p. 139).

Mine.—East Wiley, a slope mine on the east side of Oregon Basin, in the SE. ¼ sec. 10, T. 51 N., R. 100 W., 6 miles east of Wiley and 14 miles southeast of Cody.

Coal bed.—No name. Cretaceous age, Gebo (Eagle sandstone) formation. Dip, 22° E. Roof, shale; floor, brown shale on massive sandstone.

The bed was measured and sampled by D. F. Hewett on August 14, 1911, as shown below:

Section of coal bed in East Wiley mine.

Laboratory No.	12884	
Roof, shale.	*Ft.*	*in.*
Coal	1	6
Shale	[a]0	5
Coal	2	1
Floor, brown shale on massive sandstone.		
Thickness of bed	4	0
Thickness of coal sampled	3	7

[a] Not included in sample.

The sample was taken from the south rib of the slope at a point 180 feet from the mouth. The coal was unweathered.

Notes.—This mine was sampled by E. G. Woodruff in 1907 (sample No. 5762). The several benches of the bed vary greatly in thickness in the workings, but the total thickness of the bed is nearly uniform. The mine was operated intermittently.

WILEY. WEST WILEY (THOMPSON) MINE.

Sample.—Subbituminous coal; Bighorn Basin field; analysis No. 12883 (p. 139).

Mine.—West Wiley. A drift mine on the southwest side of the Oregon Basin, in sec. 34, T. 51 N., R. 101 W., 4 miles southwest of Wiley and 15 miles southeast of Cody. Operated intermittently.

Coal bed.—No name. Cretaceous age; Gebo (Eagle sandstone) formation. Dip, 28° SW.

The bed was measured and sampled by D. F. Hewett, June 30, 1911, as shown below:

Section of coal bed in West Wiley mine.

Laboratory No.	12883	
Roof, shale.	*Ft.*	*in.*
Coal, bony	[a]0	9
Shale	[a]0	4
Coal, bony	[a]0	4
Coal	1	7
Shale	[a]0	3
Coal	1	8
Shale, black, soft	[a]0	4
Coal, extremely variable	[a]0	4
Floor, sandstone.		
Thickness of bed	5	7
Thickness of coal sampled	3	8

[a] Not included in sample.

The sample was taken from the face of a room 300 feet from the mouth of the drift at a point 40 feet below the surface. The coal was unweathered.

Notes.—This mine was sampled by E. G. Woodruff in 1907 (sample No. 5767), but at the time of sampling the slope had been abandoned and the coal was mined from a drift. The several benches of the bed vary greatly in thickness throughout the workings, the total thickness varying from 4 feet 4 inches to 6 feet.

SHERIDAN COUNTY.

ACME. ACME NO. 1 MINE.

Sample.—Subbituminous coal, Sheridan field; analyses Nos. 10823, 10824, 10826, and 12687 (p. 139).

Mine.—Acme No. 1, a drift mine at Acme, in the SE. ¼ sec. 13, T. 57 N., R. 85 W., on the main line of the Chicago, Burlington & Quincy Railroad.

Coal bed.—Carney. Tertiary age, Fort Union formation. Average thickness, 16 feet; dip, 1° SE.; roof, soft shale or soapstone; floor, shaly clay, fairly hard.

The bed was measured and sampled at three points by H. M. Wolflin, on August 29, 1910, and at one point by O. B. Hopkins, September 30, 1911, as described below:

Sections of coal bed in Acme No. 1 mine.

Section	A		B		C		D	
Laboratory No.	10823		10824		10826		12687	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Dirty coal	..	..	..	..	[a]0	10	..	..
Soft gray shale	..	..	..	..	[a]1	0	..	..
Coal	..	..	..	..	..	..	[a]4	0
Coal and shale	..	..	..	..	[a]0	2	..	..
Shale	..	..	..	..	..	..	[a]1	3
Coal	..	..	..	..	7	7½	8	0
Gray shale	[a]0	10	[a]1	0	[a]0	5	[a]0	7
Bony coal	..	..	[a]0	4	..	..	..	..
Bottom coal, clean	6	2	6	10	[a]7	0	[a]7	0
Floor, clay.								
Thickness of bed	7	0	8	2	17	½	20	10
Thickness of coal sampled	6	2	6	10	7	7½	8	0

[a] Not included in sample.

Section A (sample 10823) was measured on the pillar of room 9 off east entry 5, off the main entry. The sample was from the bottom coal.

Section B (sample 10824) was measured on the pillar of room 1, off west entry 3, off the main north entry, and represented bottom coal.

Section C (sample 10826) was measured on the pillar near the face of west entry 3, off the main north entry; it represented top coal.

The point of measurement of section D was not recorded.

Notes.—The room-and-pillar system of mining was used. At the time of sampling black powder was used for breaking the coal when undercut and for brushing the top and taking up the bottom. None of the coal was shipped as run-of-mine. It was all passed over bar screens set 1½, 3, 4, and 5 inches apart. The screenings from the bar screens passed through revolving screens with openings of 1 and 2 inches. The coal was clean and free from rock but had a tendency to slack. There was only one loading track, with a capacity of 30 cars. The output of the mine was 350 tons per day.

ACME. ACME No. 2 MINE.

Sample.—Subbituminous coal; Sheridan field; analyses Nos. 10807, 10808, 10609, 10811, and 12683 (p. 139).

Mine.—Acme No. 2, a drift mine at Acme, in sec. 13, T. 57 N., R. 85 W., on the main line of the Chicago, Burlington & Quincy Railroad.

Coal beds.—Carney and Masters. Tertiary age, Fort Union formation. Total thickness of Carney bed, 8 feet; thickness of workable portion, 6 feet; dip, 1° southwest; thickness of Masters bed, 7 to 9 feet. Roof, Carney bed, soft shale above roof coal; followed by soft sandstone; Masters bed, shale. Floor, Carney bed, black shaly clay; Masters bed, shale.

The Carney bed was measured and sampled at four points in the mine by H. M, Wolflin, on August 24, 1910, as described below:

Sections of Carney bed in Acme No. 2 mine.

Section	A		B		C		D	
Laboratory No.	10807		10808		10809		10811	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Roof coal	..	..	..	..	..	..	..	..
Clean coal	2	10	2	8	2	6	2	8
Woody coal	1	4	1	4	1	4	1	2
Clean coal	2	2	2	2	2	3½	2	8½
Floor, black shaly clay.								
Thickness of bed	6	4	6	2	6	1½	6	6½
Thickness of coal sampled	6	4	6	2	6	1½	6	6½

Section A (sample 10807) was measured at the face of room 13 off south entry 2 off west entry 3.

Section B (sample 10808) was measured at the face of south entry 5 off west entry 3.

Section C (sample 10809) was measured at the face of room 1 off north entry 1 off west entry 5.

Section D (sample 10811) was measured on a pillar of west entry 1.

The Masters bed was measured and sampled on September 30, 1911, by O. B. Hopkins, as described below:

Section of Masters bed in Acme No. 2 mine.

Laboratory No.	12683	
Roof, shale.	*Ft.*	*in.*
Coal	[a] 1	6
Coal	7	0
Floor, shale.		
Thickness of bed	8	6
Thickness of coal sampled	7	0

[a] Not included in sample.

Sample 12683 was taken 2,000 feet northeast of the mine mouth.

Notes.—The coal is subbituminous merging into lignite. About 48 per cent of the output was shipped as run-of-mine. About 10 per cent was disposed of as screenings. The screening plant was the same one described under Acme No. 1 mine. The room-and-pillar system of mining was used; 60 per cent of the coal was produced from advance workings. The ultimate recovery was expected to be 90 per cent. Black powder was used for breaking down the coal, brushing the top, and taking up the bottom.

Carneyville. Carney Mines.

Samples.—Subbituminous coal, Sheridan field; analyses Nos. 12684, 15677, 15678, 15679, 15680, 15681, 15682, 15683, 15684, 15685, and 15686 (pp. 140, 141).

Mines.—Carney, two drift mines north of Carneyville, in secs. 16 and 17, T. 57 N., R. 84 W., on the Chicago, Burlington & Quincy Railroad.

Coal bed.—Known as the Carney. Tertiary age; Fort Union formation. Thickness, fairly uniform, ranging from 16 to 18 feet, 10 to 12 feet of which is mined. Dip, 1° to 2° SE. The coal is mined to a clay parting 1½ to 6 inches thick; in some places it is rather soft and in falling becomes mixed with the coal. Above the parting is 4 to 4½ feet of coal of good grade; it makes an excellent roof and no timbering is required in the rooms or entries. Above this roof coal there is 1 foot to 1 foot 2 inches of clay shale which falls easily. Floor, rather smooth, soft underclay. In places a streak of bony coal 2 to 6 inches thick appears at the bottom of the coal bed next to the underclay. Where this streak was present the undercutting was done above it. The cover ranges from 200 to 225 feet.

The bed was measured and sampled at one point by O. B. Hopkins, on October 4, 1911, as described below:

Section of coal bed in Carney mine.

Laboratory No	12684	
	Ft.	*in.*
Coal	*a* 5	0
Shale	*a* 1	0
Coal	10	0
Thickness of bed	16	0
Thickness of coal sampled	10	0

a Not included in sample.

The section (sample 12684) was measured 1,000 feet northwest of the mine mouth.

The coal bed was also measured and sampled at four points in No. 1 mine and five points in No. 2 mine by O. U. Bradley on January 1 to 9, 1913, as described below:

Section A (sample 15677) was measured at the face of room 10, off north entry 4, off east entry 2. The sample represented a 10-foot 6-inch cut of coal.

Section B (sample 15678) was measured 25 feet in room 9, north entry 2, off west entry 4. The sample represented an 11-foot 2-inch cut of coal.

Section C (sample 15679) was measured at the face of the main entry. The sample represented a 10-foot 10-inch cut of coal.

Section D (sample 15680) was measured at the face of room 12, off north entry 4, off west entry 3, 75 feet from north entry 4. The sample represented a 10-foot 1-inch cut of coal.

These samples were from No. 1 mine; those from No. 2 mine follow.

Section A (sample 15681) was measured at the face of room 5, off north entry 2, off west entry 2. The sample represented an 11-foot 10-inch cut of coal.

Section B (sample 15682) was measured 85 feet in room 16 at face, off north entry 3, off east entry 1. The sample represented an 11-foot 1-inch cut of coal.

Section C (sample 15683) was measured 125 feet in room 15 at face, off north entry 2, off east entry 2. The sample represented an 10-foot 11-inch cut of coal.

Section D (sample 15684) was measured at the face of the main north entry. The sample represented an 11-foot 7-inch cut of coal.

Section E (sample 15685) was measured 65 feet in room 9 at face, off north entry 4, off west entry 1. The sample represented a 10-foot 5-inch cut of coal.

A composite sample was made by mixing samples 15677, 15678, 15679, 15680, 15681, 15682, 15683, 15684, and 15685. The results of an ultimate analysis of this sample are given under laboratory No. 15686.

Notes.—The coal was mined by the panel method, room-and-pillar system. At the time of sampling the coal was undercut by chain breast machines in the bottom part of the bed and shot down with black powder. One wooden tipple was in constant use at the mines and a new steel tipple had been built as a reserve when heavy orders were received. About 22 per cent of the coal was shipped in run-of-mine form, and about 68 per cent passed through the screens. The wooden tipple had bar screens 7 feet long, 3½ feet wide, with 5½-inch spaces; shaker screen, 7½ feet by 8 feet, with holes 2½ inches in diameter, and a revolving screen 28 feet long divided into two sizes, as follows: 12 feet with 1-inch holes and 16 feet with ½-inch holes. No pickers were employed. The tipple was rather high, and the coal fell such a distance in screening that an unusually large percentage of fine coal was produced. All slack coal, which at times included all sizes below 1½ inches, was loaded direct into box or flat cars, hauled out, and dumped. During certain months of the year, when demand for the fine coal was small, this waste amounted to 2,000 to 3,000 tons per month. The loaded lumps were medium large and had a good appearance on the cars. Three loading tracks were available, with a capacity of 160 loaded and 160 empty cars. About 1,800 acres remained to be mined. Average daily output, 1,800 tons, with a maximum day's run of 2,200 tons.

DIETZ. DIETZ NO. 4 MINE.

Sample.—Subbituminous coal; Powder River field; analyses Nos. 16221, 16222, 16223, 16224, 16225, 16226, 16227, 16228, 13498, 13499, 13537 (p. 141).

Mine.—Dietz No. 4, a slope mine in sec. 34, T. 57 N., R. 84 W., at Dietz, on the main line of the Chicago, Burlington & Quincy Railroad.

Coal bed.—Dietz No. 2 (?). Tertiary age, Fort Union formation. Average thickness, 8 feet. The bed is practically horizontal, and very few faults, rolls, or horsebacks are encountered. Roof, bony coal about 6 inches thick, overlain by gray shale. The floor is a shaly clay, and in many places is wet and soft, and particles of it mix with the coal in loading. The cover at the points where the samples were taken varies from 100 to 220 feet in thickness.

The coal bed was measured and sampled at two points in the mine by H. M. Wolflin, January 12, 1912, as described below:

Sections of coal bed in Dietz No. 4 mine.

Section	A		B	
Laboratory No.	13498		13499	
Roof, coal and shale.	*Ft.*	*in.*	*Ft.*	*in.*
Clean coal	2	10	3	9
Bony coal	0	1	0	2
Clean coal	5	6¼	3	9
Floor, shaly clay.				
Thickness of coal mined	8	5¼	7	8
Thickness of bed sampled	8	5¼	7	8

Section A (sample 13498) was measured at the face of room 3, off west entry 4.

Section B (sample 13499) was measured at the face of room 1, off west entry 12, off the main north entry.

A composite sample was made by mixing face samples 13498 and 13499; the results of an ultimate analysis of this sample are shown under laboratory No. 13537.

The bed was also measured and sampled at seven points in the mine by O. U. Bradley, January 31 to February 3, 1913, as described below:

Section A (sample 16221) was measured at the face of room 9, off west entry 6 off north entry 6. The sample represented a 7½ foot cut of coal, the thickness of the coal mined.

Section B (sample 16222) was measured at the face of room 2, off west entry 12 off north entry 4. The sample represented a 7-foot 7-inch cut of coal, the thickness of the coal mined.

Section C (sample 16223) was measured at the face of north entry 4 off west entry 4. The sample represented a 7½-foot cut of coal, the thickness of the coal mined.

Section D (sample 16224) was measured at the face of room 10, off east entry 5 off the main north entry. The sample represented a 7-foot 5-inch cut of coal, the thickness of the coal mined.

Section E (sample 16225) was measured at the face of the main north entry. The sample represented a 7-foot 1-inch cut of coal, the thickness of the coal mined.

Section F (sample 16226) was measured on pillar in room 9, off east entry 14. The sample represented an 8-foot 7-inch cut of coal, the thickness of the coal mined.

Section G (sample 16227) was measured at the face of main entry, 150 feet from opening. The sample represented a 6-foot 1-inch cut of coal, the thickness of the coal mined.

A composite sample was made by mixing the pillar and face samples 16221 to 16227, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 16228.

Notes.—The Dietz No. 4 mine is opened by a slope. At the time of sampling the coal was mined by the double-entry room-and-pillar system. Part of the coal was undercut by hand and part was shot off the solid. The cuttings from mining were loaded with the coal. The coal was shot down with black blasting powder. The screening plant consisted of shaker screens as follows: Screen 6 by 8 feet with 7-inch holes; screens 2½ by 8 feet with 2½-inch holes, 3½ by 8 feet with 2-inch holes; 6 by 8 feet with 1¼-inch holes; and 25 by 8 feet with ⅞-inch holes. None of the coal was shipped in run-of-mine form. No pickers were employed. The lump coal had a firm and fairly clean appearance on the cars and was medium large. The coal screenings were fairly clean and well sized. There were five loading tracks with a capacity of 48 empty cars and 48 or more loaded cars. The mine had a probable lifetime of about six years; the tonnage was to be derived from both advance and pillar work. The daily output of the mine was 1,000 tons, with a maximum day's run of 1,500 tons; 1,000 tons was about the capacity of the plant.

KOOI. HUGHEY PROSPECT.

Sample.—Subbituminous coal; Sheridan field; analysis No. 12642 (p. 141).

Prospect.—Hughey; sec. 27, T. 57 N., R. 85 W., 1½ miles south of Kooi.

Coal bed.—Monarch (?). Tertiary age, Fort Union formation. Roof and floor, shale. The bed was measured and sampled by O. B. Hopkins on September 11, 1911. The sample represented a 4-foot cut of coal, the thickness of the bed.

KOOI. KOOI MINE.

Sample.—Subbituminous coal; Sheridan field; analyses Nos. 10818, 10819, 10820, 10821, 12641, 16060, 16061, 16062, 16063, 16064, 16065, 16066, and 16067 (pp. 141, 142).

Mine.—Kooi; a slope mine in sec. 23, T. 57 N., R. 85 W.; at Kooi on a branch of the Chicago, Burlington & Quincy Railroad, about three-fourths of a mile from the main line.

Coal bed.—Known as the Monarch. Average thickness of the coal as mined, 8 feet 6 inches. The bed dips 1° S. 4° E. with a pronounced east-and-west cleat. The roof is coal of variable thickness; above the roof coal is a thick bed of alternate coal and shale. Floor, hard clay, which becomes soft when wet. The cover, at the points where the samples were taken, ranges from 50 to 210 feet.

The bed was measured and sampled at four points by H. M. Wolfin, on August 26, 1910, and at one point by O. B. Hopkins, on September 11, 1911, as described below:

Sections of coal bed in Kooi mine.

Section	A		B		C		D		E	
Laboratory No.	10818		10819		10820		10821		12641	
Roof, coal and shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Top coal	..	..	[a] 20	0	[a] 25	0	[a] 25	0	..	..
Shaly coal	[a] 0	4	[a] 0	3	[a] 0	4	..	..	..	..
Clean coal	1	10	1	3	1	3½	1	0	..	..
Shaly coal	[a] 0	5½	[a] 0	2½	[a] 0	1½	[a] 0	2	..	..
Coal	..	..	..	..	..	..	..	..	[a] 4	0
Shale	..	..	..	..	..	..	..	..	[a] 3	0
Coal	..	..	..	..	..	..	..	..	[a] 7	0
Shale	..	..	..	..	..	..	..	..	[a] 0	½
Coal, clean	6	7	7	3	7	6	7	½	9	0
Floor, clay.										
Thickness of bed	..	..	28	11½	34	3	33	2½	23	½
Thickness of coal sampled	8	5	8	6	8	9½	8	½	9	0

[a] Not included in sample.

Section A (sample 10818) was measured at the face of room 9, off west entry 5 off the main entry.

Section B (sample 10819) was measured at the face of the main air course which is also a manway.

Section C (sample 10820) was measured at the face of room 3, off west entry 7.

Section D (sample 10821) was measured at the face of room 6, off east entry 5 off the main entry.

Section E (sample 12641) was measured in the east entry, 2,100 feet from the mine mouth.

The bed was also measured and sampled at seven points by O. U. Bradley on January 23 to 25, 1913, as described below:

Sections of coal bed in Kooi mine.

Section	A		B		C		D		E		F		G	
Laboratory No.	16060		16061		16062		16063		16064		16065		16066	
Roof, 4 to 6 feet of coal.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Clean coal	6	10	9	0	6	9	6	9	7	6	8	4	7	1
Bony coal	[a] 0	4	..	..	[a] 0	3	[a] 0	5	[a] 0	6	..	..	[a] 0	8
Clean coal	1	1	..	..	1	2	1	0	1	0	..	..	1	1
Floor, underclay.														
Thickness of coal mined	8	3	9	0	8	2	8	2	9	0	8	4	8	10
Thickness of coal sampled	7	11	9	0	7	11	7	9	8	6	8	4	8	2

[a] Not included in sample.

Section A (sample 16060) was measured at the face of room 20, off the fifth east entry, first panel.

Section B (sample 16061) was measured at the face of room 9, off the fifth east entry, fourth panel.

Section C (sample 16062) was measured at the face of the fifth east entry, 1,900 feet from the main slope.

Section D (sample 16063) was measured at the last crosscut, eleventh east entry, 165 feet from main slope.

Section E (sample 16064) was measured at the face of room 4, off tenth east entry, fourth panel, room just turned.

Section F (sample 16065) was measured at the face of room 7, off tenth east entry, second panel.

Section G (sample 16066) was measured at the face of the twelfth west entry, 185 feet from main slope.

A composite sample was made by mixing the face samples 16060 to 16066, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 16067.

Notes.—The Kooi mine is opened by a slope 3,550 feet above sea level and is worked by the room-and-pillar panel system. At the time of sampling the coal was undercut by puncher and electric chain breast machines and broken down with black blasting powder. The machine cuttings were loaded with the coal. The screening plant consisted of shaking screens 6 by 8 feet with 6-inch openings, 6 by 9 feet with 2½-inch openings, 6 by 7 feet with 1¼-inch openings, and 6 by 7 feet with ¾-inch openings. Thirty per cent of the coal passed over the 6-inch screen. The lumps on the car were large and had a good appearance. There were five loading tracks with a capacity of 60 empty cars and 60 loaded cars. Twenty-seven per cent of the coal was shipped in run-of-mine form. Estimated probable life of the mine, 15 to 20 years. The estimated average output at time of sampling was 1,000 tons a day; maximum day's run 1,600 tons.

MONARCH. MONARCH MINES.

Sample.—Subbituminous coal; Sheridan field; analyses Nos. 12005, 12006, 12008, 12685, 15830, 15831, 15832, 15833, 15834, 15835, 35836, and 15837 (p. 143).

Mines.—Monarch and New Monarch, drift mines. The Monarch is at Monarch and the New Monarch is in the SE. ¼ sec. 24, T. 57 N., R. 85 W., west of Monarch, with railroad connections to the main line of the Chicago, Burlington & Quincy Railroad.

Coal bed.—Monarch. Tertiary age, Fort Union formation. Average thickness, 30 feet, about 10 feet of which is mined. The bed dips slightly to the southeast. The upper part of the bed consists of coal mixed with bands of shale which are nonpersistent; mining is confined to the lower part. Floor, dark-gray underclay which becomes very soft and swells when wet. The Carney bed is 90 feet below the Monarch. The cover at the points where samples were taken ranges from 40 to 300 feet.

The bed was measured and sampled at three points in the old mine by H. M. Wolflin, on March 29, 1911; and at one point in the new mine by O. B. Hopkins, on October 4, 1911.

Sections of coal bed in Monarch mines.

Section	A		B		C		D	
Laboratory No	12005		12006		12008		12685	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	..	..	..	..	..	..	[a]6	0
Shale	..	..	..	..	..	..	[a]3	0
Coal	..	..	..	..	..	..	[a]6	0
Clean bright coal	7	10	5	4	8	2	10	0
Dull hard coal	..	..	0	9	..	..	..	..
Clean bright coal	..	..	2	1	..	..	..	..
Floor, underclay.								
Thickness of coal sampled	7	10	8	2	8	2	10	0

[a] Not included in sample.

Section A (sample 12005) was measured on the pillar of butt entry 3, off west entry 9, near room 70.

Section B (sample 12006) was measured at the face of room 70, near west entry 7.

Section C (sample 12008) was measured on a pillar in west entry 9, near room 70.

Section D (sample 12685) was measured 800 feet southwest of the mine mouth.

The bed was also measured and sampled at seven points in the new mine by O. U. Bradley on January 11 to 17, 1913, as described below:

Section A (sample 15830) was measured at the face of east entry 6, 250 feet from main haulage slope. The sample represented a 9-foot 2-inch cut of clean coal.

Section B (sample 15831) was measured at the face of west entry 8.

Section C (sample 15832) was measured at the face of west entry 13, off old slope. The sample represented an 8-foot 11-inch cut of clean coal.

Section D (sample 15833) was measured in west entry 4, opposite butts 5 and 6, 800 feet from haulage way. The sample represented a 10-foot 1-inch cut of clean coal.

Section E (sample 15834) was measured at the face of room 8, off butt entry 4, off west entry 4. The sample represented a 10-foot 4-inch cut of clean coal.

Section F (sample 15835) was measured on a pillar between rooms 24 and 25, off west entry 12. The sample represented a 9-foot cut of clean coal.

Section G (sample 15836) was measured on a pillar between rooms 5 and 6, off butt entry 2, off west entry 3. The sample represented a 9-foot 9-inch cut of clean coal.

A composite sample was made by mixing samples 15830 to 15836, inclusive. The results of an ultimate analysis of this sample are given under laboratory No. 15837.

Notes.—The Monarch mine is opened by a drift and the coal is mined by the double-entry room-and-pillar panel system. At the time of sampling the coal was undercut by electric chain coal-cutting machines and broken down with FF black powder. Machine cuttings were loaded out with the coal. No impurities mixed with the coal in loading. About one-third of the coal was shipped as run-of-mine and about 68 per cent of the remainder passed through the screens. The tipple was equipped with bar screens, one 12 feet long with 5-inch openings, one 12 feet long with 4-inch openings, and shaker screens, one 12 by 6 feet with 2½ and 1¼ inch openings, and one 12 feet long with 1-inch openings. No pickers were employed on the cars. The mine had a storage bin with a capacity of 280 tons. The lumps and slack appeared fairly good on the cars. Eight loading tracks were available with a capacity of 100 empty and 100 loaded cars. Fifty per cent of the coal mined was from advance workings. The mine had approximately 300 acres unmined. Average daily ouput, 1,600 tons.

NEW ACME. NEW ACME MINE.

Sample.—Subbituminous coal; Sheridan field; analyses Nos. 16055, 16056, 16057, 16058, and 16059 (p. 143).

Mine.—New Acme, a drift mine in sec. 22, T. 57 N., R. 84 W., at New Acme, on a spur of the Chicago, Burlington & Quincy Railway, three-fourths of a mile from the main line.

Coal bed.—Known as the Monarch. Tertiary (Eocene) age, Fort Union formation. The thickness of the coal bed ranges from 16 to 30 feet, of which 6½ to 10 feet is mined. Coal was left for roof as the overlying strata was weak. Many fires originated in the roof coal, particularly when the pillars were being pulled. The floor was coal in all places examined or sampled. The floor of the Monarch seam as exposed elsewhere is composed chiefly of dark carbonaceous shale and squeezes and swells when wet.

There is about 50 to 100 feet of cover at the points where the samples were collected.

The coal bed was measured and sampled at four points in the mine on January 20 and 21, 1913, by O. U. Bradley, as described below:

Section A (sample 16055) was measured at the face of west entry 7, 150 feet from the main slope. The sample represented a 6-foot 8-inch cut of clean coal.

Section B (sample 16056) was measured at the face of room 4, off east entry 5. The sample represented a 7-foot 2-inch cut of clean coal.

Section C (sample 16057) was measured at the face of room 13, off east entry 3. The sample represented a 7-foot 7-inch cut of clean coal.

Section D (sample 16058) was measured at the face of room 11, off east entry 2. The sample represented a 6-foot 9-inch cut of clean coal.

A composite sample was made by mixing samples 16056 to 16058, inclusive. The results of an ultimate analysis of this sample are shown under laboratory No. 16059.

Notes.—The New Acme mine is opened by a drift. The coal was mined by the room-and-pillar and later by the panel system. At the time of sampling the coal was undercut by electric breast machines and broken down with black powder. About 35 per cent of the coal was shipped as run-of-mine, 70 per cent of the coal going to the screen passed through. The screening equipment consisted of bars 4 by 8 feet with 5-inch openings; shaker screens 8 by 12 feet, the diameter of holes being 1¼, 2½, 4, and 6 inches; and revolving screens 24 feet long divided into sections as follows: 8 feet with ½-inch holes, 8 feet with 1-inch holes, 4 feet with 2½-inch holes, and 4 feet with 3½-inch holes. No pickers were employed on the cars. The mine had a storage bin of 500 tons capacity. The lump coal on the cars appeared dull; the lumps were small. The mine had three loading tracks with a capacity for 86 empty cars and 86 loaded cars. A runaround for switching permitted loading while cars were being shifted or loads taken out. As the tipple was rather high, there was more or less breakage in the handling of the coal. There was approximately 1,000 acres to be mined. The daily output averaged about 1,000 tons, with a maximum day's run of 1,420 tons. The estimated life of the mine was 50 years.

SWEETWATER COUNTY.

GUNN. "B" MINE.

Sample.—Subbituminous coal; Rock Springs field; analyses Nos. 11507, 11508, 11509, and 11510 (p. 144).

Mine.—Gunn-Quealy "B," a slope mine in sec. 8, T. 19 N., R. 104 W., at Gunn, 4 miles from Baxter, on a branch line from the Union Pacific Railroad.

Coal bed.—Locally known as No. 11. Cretaceous age, Mesaverde formation. Average thickness, 6 feet 1 inch; dip, 7½° S., 80° W.; cleat S. 70° E. Roof, shale. about 16 feet thick; floor, shaly sandstone; cover, at points where samples were taken, 350 to 360 feet thick.

The bed was measured and sampled at three points in the mine by S. S. Smith on December 22, 1910, as described below:

Section A (sample 11507) was measured at the face of room 15, off entry 5. The sample represented a 6-foot 2-inch cut.

Section B (sample 11508) was measured at the face of room 25, off entry 4. The sample represented a 6-foot 1-inch cut.

Section C (sample 11509) was measured at the face of room 8, off entry 8. The sample represented a 6-foot 2-inch cut.

Notes.—The room-and-pillar system of mining was employed. At the time of sampling the coal was undercut with electric chain cutters. Black powder was used for breaking down the coal and dynamite for brushing the roof. About 30 per cent of the coal was shipped as run-of-mine; the remainder was sized by shaking and revolving screens. No hand picking was done. There were four loading tracks with capacity for 100 cars.

The daily production was 1,000 tons, 80 per cent being obtained from advance work. About 375 acres of unmined coal was tributary to this slope.

SUPERIOR. B MINE.

Sample.—Bituminous coal; Rock Springs field; analyses Nos. 11461, 11462, 11463, and 11464 (p. 144).

Mine.—B, a drift mine in sec. 29, T. 21 N., R. 102 W., 2 miles northeast of Superior, on a branch of the Union Pacific Railroad from Thayer Junction and only a few miles north of the main line.

Coal bed.—No. 1. Cretaceous age, Mesaverde formation. Locally known as No. 7 of the Laramie. Average thickness, 6 feet 6 inches; dip, 4½° NE. The roof is a smooth black shale 15 feet thick with a sandstone cap rock; the floor is a hard smooth shale; the covering at the point of sampling is 400 feet.

The bed was measured and sampled at four points in the mine by S. S. Smith on December 14, 1910, as described below:

Sections of coal bed in the B mine.

Section	A		B		C		D	
Laboratory No.	11461		11462		11463		11464	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	..	..	1	1	1	6	..	..
Rock parting	..	..	0	¼	0	¼	..	..
Coal	7	2	6	3½	5	9½	6	3½
Floor, shale.								
Thickness of bed	7	2	7	4¾	7	3¾	6	3½
Thickness of coal sampled	7	2	7	4¾	7	3¾	6	3½

Section A (sample 11461) was measured at the face of room 27, off the inside dip entry.

Section B (sample 11462) was measured at the face of south entry 2.

Section C (sample 11463) was measured at the face of room 20, off south entry 3.

Section D (sample 11464) was measured at the face of room 11, off north entry 3.

Notes.—The coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut with electric chain cutters; black powder and a permissible explosive was used for breaking down the coal; 75 per cent of the coal produced was from advance workings; daily output 1,200 tons; unmined area tributary to this opening, 250 acres.

SUPERIOR. SUPERIOR C MINE.

Sample.—Subbituminous coal; Rock Springs field; analyses Nos. 11456, 11457 11458, and 11459 (pp. 144, 145).

Mine.—C, a slope mine in sec. 21, T. 21 N., R. 102 W., at Superior, on a branch line of the Union Pacific Railroad from Thayer Junction.

Coal bed.—Unnamed. Cretaceous age, Mesaverde formation, known locally as No. 1 of the Laramie. Average thickness, 8 feet 6 inches; dip 4½° NE. The cleat is well defined and extends east and west. Roof, black shale 4 to 8 feet thick, above which is a heavy capping of sandstone; floor, hard, smooth shale. The cover at points where samples were taken is about 250 feet.

The bed was measured and sampled at four points by S. S. Smith on December 14, 1910, as described below:

Sections of coal bed in the Superior C mine.

Section	A		B		C		D	
Laboratory No.	11456		11457		11458		11459	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	1	10	2	3	..	..	2	3
Rock	[a]0	1½	[a]0	1¾	..	..	[a]0	1½
Coal	6	8	6	3	6	7	6	2
Floor, shale.								
Thickness of bed	8	7½	8	7¾	6	7	8	6½
Thickness of coal sampled	8	6	8	6	6	7	8	5

[a] Not included in sample.

Section A (sample 11456) was measured at the face of room 15, off south entry 6.

Section B (sample 11457) was measured at the face of room 25, off south entry 5.

Section C (sample 11458) was measured at the face of room 1, off north entry 7.

Section D (sample 11459) was measured at the face of room 38, off south entry 4.

Notes.—The mine is opened by a slope in the coal bed and is worked by the room-and-pillar system. At the time of sampling 90 per cent of the coal produced was from advance work; black powder and a permissible explosive were used for breaking the coal after undercutting by electric machines. Daily production of mine, 1,300 to 1,400 tons; unmined area tributary to this opening, 1,200 acres.

UINTA COUNTY.

ELKOL. ELKOL MINE.

Sample.—Bituminous coal; Kemmerer field; analyses Nos. 11730 and 11731 (p. 145).

Mine.—Elkol, a slope mine in sec. 11, T. 20 N., R. 117 W., near Elkol, 8 miles southwest of Diamondville on the Oregon Short Line Railroad.

Coal bed.—Locally known as No. 1. Cretaceous age, Frontier formation. The coal bed is 51 feet thick and at the points where the samples were taken is 1,600 feet below the surface. The coal bed was sampled at 2 points in the mine by S. S. Smith on February 9, 1911, as described below:

Sections of coal bed in Elkol mine.

Section	A		B	
Laboratory No.	11730		11731	
Roof, not stated.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	[a]27	6	[a]27	6
Coal, soft and black	8	6	8	6
Coal	[a]15	0	[a]15	0
Floor, not stated.				
Thickness of bed	51	0	51	0
Thickness of coal sampled	8	6	8	6

[a] Not included in sample.

Section A (sample 11730) was measured at the face of north entry 3, 1,900 feet from the slope mouth.

Section B (sample 11731) was measured at the face of room 1, off north entry 3.

Notes.—The State mine inspector's report for the fiscal year ended September 30, 1910, gives the total production for that year as 50,000 tons, which was shipped to points in Utah, Idaho, Oregon, Washington, and California.

FRONTIER. KEMMERER NO. 1 MINE.

Sample.—Bituminous coal; Kemmerer field; analyses Nos. 11565, 11566, 11567, and 11568 (p. 145).

Mine.—No. 1, a slope mine in sec. 12, T. 21 N., R. 116 W., at Frontier, on a branch of the Oregon Short Line Railroad.

Coal bed.—Cretaceous age, Frontier formation. Locally known as No. 1 of the Laramie. Average thickness, 7 feet; dip, 13° west; well-defined cleat. Roof, draw slate, 3 feet 6 inches thick, which stands fairly well, overlain by shaly sandstone; floor, hard, smooth fire clay. Cover at points where samples were taken varies from 175 to 400 feet.

The bed was measured and sampled at four points in the mine by S. S. Smith on January 11, 1911, as described below:

Sections of coal bed in the Kemmerer No. 1 mine.

Section	A		B		C		D	
Laboratory No.	11565		11566		11567		11568	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal (hard)	3	7	4	9	1	6	1	6
Soft rock parting	[a]0	½	[a]0	4	[a]0	6	[a]0	10
Coal (soft)	4	9	3	4	3	10	3	8
Floor, fire clay.								
Thickness of bed	8	4½	8	5	5	10	6	0
Thickness of coal sampled	8	4	8	1	5	4	5	2

[a] Not included in sample.

Section A (sample 11565) was measured at the face of room 16, off south entry 13, and 400 feet below the surface.

Section B (sample 11566) was measured on the pillar of room 18, off north entry 10, and 240 feet below the surface.

Section C (sample 11567) was measured at the face of room 3, off north entry 7, and 175 feet below the surface.

Section D (sample 11568) was measured on the pillar of room 62, off south entry 7, and 200 feet below the surface.

Notes.—The mine is worked by the room-and-pillar system. At the time of sampling the coal was shot from the solid with FFF black powder. The coal was shipped as run-of-mine; the lumps were large and bright, but the slack was dull. Pickers sorted out inferior coal and shale as the coal was loaded on the car. There were 5 loading tracks with capacity for 150 cars. The daily production was 600 tons.

For analyses and descriptions of other samples of coal from this mine see Bull. 22, p. 1150.

KEMMERER. PROSPECT.

Sample.—Subbituminous coal; Kemmerer field, analysis No. 11740 (p. 145).

Mine.—A prospect in sec. 33, T. 22 N., R. 116 W., 2½ miles north of the Oregon Short Line Railroad and 1½ miles west of Kemmerer.

Coal bed.—Cretaceous age, Frontier formation. Thickness, 16 feet; roof, sandstone; floor, fire clay.

The bed was sampled by S. S. Smith on February 12, 1911, at the face of the first entry, 200 feet from the mouth of the slope. The vertical depth at which the sample was taken was 150 feet.

SUSIE. KEMMERER NO. 4 MINE.

Sample.—Bituminous noncoking coal; analyses Nos. 13517, 13518, and 13519 (p. 145).

Mine.—Kemmerer No. 4, a slope mine at Susie, 4 miles north of Kemmerer, on a spur of the Oregon Short Line Railroad, secs. 25 and 26, T. 22 N., R. 116 W.

Coal bed.—Kemmerer No. 1. Cretaceous age, Frontier formation. Average thickness, 3 feet 7 inches to 5 feet 6 inches; dip 19° W.; no faults but a few rolls and horsebacks; cleat runs north and south. Roof, shale; in places there is as much as 3 feet of "draw slate," which comes down with the coal. The floor consists of a hard rough shale. Particles of the roof and floor did not get mixed with the coal in loading. The thickness of the cover at the points where the samples were taken varies from 700 to 750 feet.

The coal bed was measured and sampled at two points in the mine by J. C. Roberts on January 23, 1912, as described below:

Sections of coal bed in Kemmerer No. 4 mine.

Section	A		B	
Laboratory No.	13517		13518	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*
Coal	3	6	4	3
Clay parting	0	2	..	..
Coal	1	6	..	..
Floor, shale.				
Thickness of bed	5	2	4	3
Thickness of sampled	5	2	4	3

Section A (sample 13517) was measured 1,000 feet from mouth of slope.

Section B (sample 13518) was measured at the face of the second north entry, 1,300 feet from mouth of slope.

A composite sample was made by mixing samples 13517 and 13518. The results of an ultimate analysis of this sample are shown under laboratory No. 13519.

Notes.—The No. 4 mine was opened by a slope and the coal was mined by the room-and-pillar system. The coal was shot off the solid with black powder. Dynamite was used in brushing the floor and roof. The screening equipment consisted of shaking screens 32 feet long and 6 feet wide—16 feet with 3-inch openings and 16 feet with ¾-inch openings. No pickers were employed on the cars. There were five loading tracks with a capacity of 30 empty cars and 20 loaded cars. The average daily output was 500 tons a day with a maximum day's run of 700 tons; all of the coal was shipped as run-of-mine. About 75 per cent of the coal was taken from advance workings.

WESTON COUNTY.

CAMBRIA. ANTELOPE NO. 3 AND NO. 4 MINES.

Sample.—Subbituminous to cannel coal; Black Hills region (Wyoming No. 2) analyses Nos. 16412, 16413, 16414, 16415, 16428, 16429, 16430, and 16431 (p. 146).

Mine.—Antelope No. 3 and No. 4, drift mines at Cambria, in secs. 19 and 20, T. 46 N., R. 61 W., at end of the spur of the Chicago, Burlington & Quincy Railroad, 6½ miles from Newcastle.

Coal bed.—Not named, probably a local bed. Cretaceous age, Lakota formation. Average thickness, 5 feet, varying from 2 feet 6 inches to 8 feet. Many mud and sand pockets occur in parts of the mine. Sandstone, shale, and bony coal partings appear in the coal at varying intervals. The coal itself varies greatly in character from subbituminous to cannel coal. The coal bed is uneven; one room may be going to the dip and a parallel room going to the rise. Roof, a black sandy shale over which is 1 to 2 feet of granular sandstone which gives considerable trouble if not closely watched. Floor is rough and uneven; small metallic concretions and large hard nodules called "nigger heads" are common. It is for the most part hard black shale and sandstone. Particles of the roof and bottom did not mix with the coal in loading. The thickness of the cover at the points where the samples were taken varies from 25 feet to 225 feet.

The coal bed was measured and sampled at six points in the mine by O. U. Bradley on February 11 and 12, 1913, as described below:

Sections of coal bed in the Antelope No. 3 and No. 4 mines.

Section	A		B		C		D		E		F	
Laboratory No.	16412		16413		16414		16428		16429		16430	
Roof, shale.	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*
Coal	..	..	3	2	2	3	..	..	1	0	2	1
Rock or splint	..	..	[a]0	8	[a]1	9	..	..	[a]0	6	[a]0	6
Coal	0	8	0	11	1	4	..	..	1	0	..	..
Rock	[a]1	0	[a]0	2	[a]1	0	..	..	[a]0	3	..	..
Coal	2	3	1	2	2	7	[b]3	9	4	6	4	10
Floor, black shale and sandstone.												
Thickness of bed	3	11	6	1	8	11	3	9	7	3	7	5
Thickness of coal sampled	2	11	5	8	6	2	3	9	6	6	6	11

[a] Not included in sample.

[b] Contains two streaks of bony coal ½ to ⅝ inch thick, which is loaded with the coal.

Section A (sample 16412) was measured at the face of east entry 1, No. 4 mine.

Section B (sample 16413) was measured at the face of north entry 1, off east entry 1, No. 4 mine.

Section C (sample 16414) was measured at the face of room 29, off east entry 3, No. 4 mine.

A composite sample was made by mixing samples 16412, 16413, and 16414. The results of an ultimate analysis of this sample are given under laboratory No. 16415.

Section D (sample 16428) was measured at the face of room 35, off east entry 5, No. 3 mine.

Section E (sample 16429) was measured at the face of west entry 8, No. 3 mine.

Section F (sample 16430) was measured at the face of west entry 7, No. 3 mine.

A composite sample was made by mixing samples 16428, 16429, and 16430. The results of an ultimate analysis of the sample are given under laboratory No. 16431.

Notes.—The coal is mined by the room-and-pillar system. At the time of sampling the coal was undercut by compressed-air punching machines and broken down with black blasting powder. Dynamite was used for brushing the floor and roof. The machine cuttings were loaded with the coal. The entire output of the mines was run through a rock crusher to facilitate picking. It was then sized and resized and loaded into cars. The Chicago, Burlington & Quincy Railroad bought the entire output for steaming purposes. The coal burns well, especially under forced draft, and is particularly suited to railroad work.

MOORCROFT. LOCAL MINE.

Sample.—Subbituminous coal; Upper Belle Fourche River field; analysis No. 12446 (p. 146).

Mine.—Local drift mine, 12 miles south of Moorcroft and of the Chicago, Burlington & Quincy Railroad.

Coal bed.—Not named, a local bed.

The bed was measured and sampled by A. W. Stickney on July 17, 1911, as described below:

Section of coal bed in local mine, about 12 miles south of Moorcroft.

Laboratory No.	12446	
	Ft.	in.
Roof, sandy shale.		
Bone	[a]0	1
Coal	2	11
Floor, carbonaceous shale.		
Thickness of bed	3	0
Thickness of coal sampled	2	11

[a] Not included in sample.

The sample was taken from the face of the entry, 100 feet from the entrance.

MOORCROFT. PROSPECT.

Sample.—Subbituminous coal; Upper Belle Fourche River field; analysis No. 12392 (p. 146).

Location.—Prospect opened by sampler about 12 miles southwest of Moorcroft and of the Burlington Railroad, in the SE. ¼ sec. 29, T. 48 N., R. 68 W.

Coal bed.—Local lense in Lance formation. Tertiary or Cretaceous age. Thickness, 2 feet 4 inches; roof, clay; floor, carbonaceous shale.

The bed was measured and sampled by A. W. Stickney on June 27, 1911. The sample represented a 2-foot 4-inch cut of coal, the thickness of the bed.

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INDEX.

This index gives the names of places at or near which the samples of coal mentioned in this bulletin were collected. In addition, it gives the names of many coal beds, including names of geological significance as well as some that are merely local, and the names of most of the mines.

Attention is called to the fact that many beds opened by prospect pits, country banks, or even mines, especially beds in the Rocky Mountain province, have no names, even local ones. Moreover, such local names as "A," "B," "1," "2," "Upper," "Lower," have been omitted in the case of beds that have not been correlated with other beds similarly designated in the same State, the purpose in omitting such names being to reduce the length of the index and to avoid the confusion that would result from applying the same designation to different beds. Mines that in the text have no other title than a number, as "Mine No. 1," are not included, for the same reasons.

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Bulletin 86

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, ACTING DIRECTOR

SOME ENGINEERING PROBLEMS OF THE PANAMA CANAL IN THEIR RELATION TO GEOLOGY AND TOPOGRAPHY

PUBLISHED WITH THE APPROVAL OF THE
GOVERNOR OF THE PANAMA CANAL

BY

DONALD F. MACDONALD

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SOME ENGINEERING PROBLEMS OF THE PANAMA CANAL IN THEIR RELATION TO GEOLOGY AND TOPOGRAPHY.

By DONALD F. MACDONALD.

INTRODUCTION.

This report aims to discuss, from the viewpoint of the mining geologist, the bearing of topographic and geologic conditions on certain problems that arose in the construction of the Panama Canal. It is published by the Bureau of Mines as a contribution to engineering literature because it presents information that shows how geology and topography must be considered by the mining engineer in planning excavations and in removing loose material and solid rock in the safest and most efficient manner.

The printing of this bulletin has been approved by the governor of the Panama Canal, as it was impracticable for the Panama Canal to give this information to the general public in any of its publications.

The "sword cut" of Goethals and his men did not sever the Isthmian barrier; it merely clove the hills to a depth of 40 feet above sea level. This cut and the great areas of low valleys at both ends of it have been covered with fresh water to the 85-foot level. This high-level lake held by dams at either end, the 85-foot locks, by which ships enter it, and the dredged approaches from the two oceans constitute the Panama Canal.

In the preparation of accurate estimates concerning the cost of large excavations information regarding the geologic conditions, as obtained by a study of natural features, is necessary. Drill holes and test pits will furnish supplementary geologic information, but without a general knowledge of the geologic conditions it will not be possible for the constructing engineer to answer such questions as the following:

(1) What rock units will be encountered in the excavation and in what proportion? (2) What will be the cost of drilling each for blasting? (3) How far apart and how deep must the drill holes be to efficiently break the rock of each to the proper size—not too fine nor yet too coarse? In this connection jointing, fissuring, bedding, toughness, and other physical characteristics must be considered (4) How steep will the slopes of the excavation stand, and to what extent will they wash and trench on exposure to the atmosphere?

(5) What proportion of the material to be excavated is of such physical character that it may be utilized in any other part of the project, such as in concrete construction, road making, wharves, breakwaters, riprap work, etc.?

ACKNOWLEDGMENTS.

The first work of considerable scope bearing on isthmian geology was that of Hill[a] in 1895.

In 1899 Bertrand and Zurcher[b] published a brief report on the geology of the Isthmus for the New French Company (Compagnie Nouvelle du Canal de Panamá).

In July, 1906, Howe[c] was sent to the Isthmus by the new American canal commission to report on such phases of engineering geology as lock and dam foundations and the natural resources of the country. Again, from January to April, 1907, he spent some time on the Canal Zone. His reports are most interesting. Other publications bearing on isthmian geology are listed in the bibliography given at the end of this bulletin. Hill, Howe, and the earlier workers must be congratulated on the results they obtained, in view of the shortness of the time at their disposal, the few and small excavations then made, the thick soil, and the jungle that obscured much of the land at that time.

In 1910, as Culebra Cut was deepened, great masses of earth and rock began to crush down from its slopes. It was thought well to have these and other phases of the work that had a geological bearing studied by a geologist. To make recommendations on this question C. W. Hayes,[d] then chief geologist of the United States Geological Survey, was, at the instance of the Secretary of War and Secretary of the Interior, sent to the Canal Zone. He realized that certain geological principles underlay the slides and recommended that a geologist be appointed to study them.

For this work the writer, who had been assistant geologist of the United States Geological Survey, was sent to the Isthmus, January, 1911, as geologist to the Isthmian Canal Commission. To this commission and to members of the engineering staff, but particularly to Col. George W. Goethals, chairman and chief engineer, the writer is indebted for wide opportunity to make geological studies and for suggestion and interest in results.

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[a] Hill, R. T., Geological history of the Isthmus of Panama and portions of Costa Rica: Bull. Museum Comparative Zoology of Harvard College, vol. 28, 1898.

[b] Bertrand, M., and Zurcher, P., Étude géologique sur l'Isthme de Panama; Rapport de la Commission. Compagnie Nouvelle du Canal de Panama, vol. 1, 1899, pp. 85-120.

[c] Howe, Ernest, Canal Commission Ann. Report 1907, Appendix E, pp. 108-138; Isthmian geology of the Panama Canal: Econ. Geol., vol. 2, 1907, pp. 639-658; Geology of the Isthmus of Panama: Am. Jour. Sci., vol. 26, ser. 4, 1908, pp. 212-237.

[d] Hayes, C. W., Notes on the geology and slides of Culebra Cut: Canal Record, vol. 4, Dec. 7, 1910, p. 115.

RELATION OF DIFFERENT FACTORS TO ENGINEERING PROBLEMS.

TOPOGRAPHIC TYPES.

What topographic conditions confronted the builders of the Panama Canal? The surface configurations of that part of the Isthmian land barrier that is within the Canal Zone fall under two chief types. They are: (1) The hill type of topography, developed in the interior and locally on the Pacific side and consisting chiefly of irregular angular hills, short ridges, and crooked unsymmetrical valleys and basins (Pls. I and II); and (2) the coastal-plain type, developed as low hills and swampy flats in the vicinity of Colon and locally on the Panama side (Pl. III)

HILL TYPE.

The hill type of topography is well developed in the central and southern parts of the Canal Zone. Cerro Gorda, near Culebra, and Cerro Balboa, near the old site of Gorgona, stand approximately 1,000 feet above sea level, and below this elevation are many lesser peaks. Seen from a position of commanding view, this topography is most striking in its irregularity and presents some likeness, in its green jungle-covered irregular hills and hollows, to enormous cross waves at sea. It is a type of land form not uncommon in the Republic of Panama.

Now, what relation does this type of topography have to canal construction? In building the Panama Railroad and in building some of the dirt-train and other branch lines connected with canal construction the numerous hills had, of course, to be avoided, with the result that the sinuosity of the tracks added difficulties both to their construction and operation. The new line of the Panama Railroad is relatively little curved, but it encountered some heavy cuts and fills. The canal passes between Gold Hill, 662 feet high, and Contractors Hill, 450 feet high, and a part of each had to be removed in the digging of Culebra Cut, thus adding to the amount of excavation. Conditions had to be studied in choosing the route, so as to strike the proper balance between minimum excavation and minimum curvature of the canal channel. On the other hand, most of the hills are composed of tough hard rock which could not readily be worn away and cut into valleys by the streams. This stronger rock forms Gold and Contractors Hills, and these are strengthening pillars which hold back slides of weaker rocks. Ancon Hill, another high point of hard rock, furnished a convenient quarry for stone to be crushed and used in the concrete of the Pacific locks. The height and steepness of Ancon Hill is such that the whole rock product could be handled by gravity from the quarry, through the crushing plant, and then into cars. The site of another quarry near Frijoles was first picked out by the writer from a railroad train because of its steep ridge-like topographic expression. On further exploration the ridge was found to be composed of basalt, but the rock was a little too much jointed for the desired use, the facing of a breakwater. The hills and higher areas, because of their steep slopes, contain no stagnant water in which mosquitoes and other pests can breed, and have good drainage, hence, where convenient, they were utilized as camp sites. By clearing the jungle cover off the tops of the higher hills they became convenient triangulation stations for the surveys of the Canal Zone. Plate IV indicates the location and area of the hilly region, which is practically coextensive with the areas of igneous rocks, and Plates I and II convey some idea of its character.

COASTAL-PLAIN TYPE.

Coastal plains locally border both the Atlantic and the Pacific coasts of Central America. In the Canal Zone this type of land form is principally on the Atlantic side and comprises five kinds of subtypes. They are: (1) Coastal swamps, of considerable depth, composed of soft dark mud and organic matter, and inland swamps and alluvial basins; (2) coral reefs, awash at half tide, and coral flats, composed largely of coral débris, slightly above high-tide level; (3) river flats of alluvium in the lower valleys; (4) bars at and near the mouths of rivers, beaches, sand spits, etc.; (5) seaward-tilted relatively smooth plains, low hills, and higher dissected remnants of former plains. A discussion of the origin of the land forms of the isthmian region is not within the scope of this report.

The first practical relation between the engineering work and this type of topography was the great difficulty of making surveys through the unhealthful black-mud and green-water swamps. Next came the construction of the Panama Railroad lines across the swamps. A feature of this construction was the necessity of vast fills with wide bases. The extra width was to obviate the sinking of the fill, with accompanying bulging of the swamp material on either side of the sunken part. Another most important matter was the building up of a great coral flat from its original high-tide level to 3 or 4 feet above, in order to furnish a site for the town of Colon and the Atlantic terminal plant of the canal and the Panama Railroad. On the other hand, the flats formed mostly of coral débris, unlike the swamps, presented a solid foundation on which to dump spoil, and thus to build up and prepare a solid base for the town. The low mud swamps and river alluvium greatly facilitated the dredging of the approach channels to both the Atlantic and the Pacific locks.

CLIMATE.

In computing the cost of any engineering project due weight should be given to the effect of climatic conditions on labor efficiency and also on construction plans.

RANGE OF TEMPERATURE.

The Canal Zone has a moist tropical climate, modified by oceanic conditions, and therefore showing little yearly and daily variation of temperature. The nightly average is 71° and the daily about 85° F. The daily range of temperature is greatest in the dry season, for then there are few clouds to obscure the sun's heat in the day or to absorb and reflect the radiation from the earth at night. The highest recorded temperature over a number of years is 97° F., observed at Ancon February 13, 1906, and the lowest 59° F., observed at Bas Obispo February 9, 1907.

RAINFALL.

On the Canal Zone there are two seasons—one short and relatively dry, the other long and wet. The dry months are January to April, inclusive; the other months are wet. The wettest period is from September to December. Most of the rain falls during the day, between 10 a. m. and 4 p. m., but may fall at any time. The average yearly precipitation over a number of years for Panama is as follows: Ancon or Panama City (Pacific side), 71 inches; Culebra, 90 inches; Monte Lirio, 135 inches; Colon, 129 inches; Porto Bello, 170 inches.

Tropical conditions generally add to the cost of engineering work, from causes as follows: (1) Inefficiency of native labor; (2) inefficiency of imported labor working under tropical conditions; (3) cost of sanitation; (4) loss of time during heavy rains; and (5) hindrance to engineering works through floods, etc. These factors added many million dollars to the cost of the canal. They are all discussed at some length in the annual reports of the Isthmian Canal Commission.

AGRICULTURAL AND FOREST PRODUCTS.

The question of subsistence for the vast army of canal employees was, of course, a vital one. It was found that American vegetables could not be profitably grown in the Canal Zone. In fact, most of them, as well as oats, wheat, barley, and most kinds of American grasses, could scarcely be grown at all. Hence, these products were imported from the United States. A great cold-storage plant was established at Colon, with smaller distribution plants along the zone. In this way fresh meats, vegetables, and fruits were brought in and distributed in prime condition. The inventors of canned goods and cold storage were certainly heavy contributors to the campaign of canal construction.

The hill land of the Canal Zone will never have much agricultural value, even for tropical products, because of the steepness of the slopes and the shallowness of the soil. The steep land if cultivated would wash rapidly in the heavy tropical rains of that region. A subsequent discussion of the effect of streams brings out the fact that erosion took place in the upper parts of stream valleys, whereas deposition of gravel, sand, and silt was the dominant process in the lower ends. Now, in nearly all countries the broad flood plains that have been built by deposition of rich silty soil are especially adapted for agriculture. The flood plains of the Chagres, Trinidad, and Gatun Rivers contained much rich land suitable for growing bananas, tobacco, and sugar cane. However, they also contained pools of stagnant water that afforded breeding places for mosquitoes and other pests, and hence were considerably more unhealthful than the higher ground. Much of this alluvial land has now been covered by Gatun Lake, and what remains will probably never have value commensurate with its richness of soil because of the depressing climate and the malarial conditions that prevail.

Most of the timber within the Canal Zone was cut years ago, so that nearly all the lumber needed for building the "construction-camp" towns had to be imported. Some of the ties for the Panama Railroad were made by natives from extremely resistant native woods, such as guyacan (or false lignum-vitæ) and níspero, or balata tree (*Mimusops darinensis* Pittier). The name, however, is also applied to other trees of the same family. For making dugout canoes the natives use chiefly espavé (*Anacardium rhinocarpus* D. C.), cedro, or Spanish cedar (*Cedrela* sp. pl.), and pochote (*Bombax Barrigon* Seem.). In general, the forests were more of a hindrance to the work than otherwise, for they formed a thick jungle through which many survey lines had to be cut and clearings made for roads, trails, reservoirs, and town sites.

STREAMS.

The heavy precipitation is evidence that the country must be well watered. The Chagres, which flows for a part of its course through the Canal Zone, is the chief stream (see Pl. IV). It has a relatively large catchment basin, the slopes of which are locally steep; hence during heavy rains the volume of its flow is vastly increased, only to shrink greatly during the dry season. Trickling streams swell into destructive torrents after long and heavy rains. The percentage of absorption during the wet season is relatively small because the covering of porous soil, especially on the higher land and steep slopes, is very thin, and most of the rocks, except the river alluvium and gravels, the conglomerates, and sandstones, are exceedingly fine grained and have relatively little pore space or

COASTAL-PLAIN TYPE OF TOPOGRAPHY.

Flat lands near Mount Hope, with some small knolls, or rounded hills, rising 10 to 50 feet above the general low flat surface of the plain. This group of small knolls has been called the Monkey Hills. The hills are remnants of a higher land surface; the interhill areas having been washed away by stream erosion when the isthmian land stood some hundreds of feet higher than now. *a*, A knoll rising about 30 feet above the surrounding plain.

water-containing capacity. Then, too, the rainfall is so heavy that the ground is soon saturated and the percentage of run-off becomes high.

The streams on their way to the ocean pass through three general stages of size and activity. In the first stage the stream is small and flows swiftly down the gradient of a steep valley, carrying along bits of soil and rock from the surface over which it passes. In the second stage the stream is less swift but much larger, having been reinforced by many smaller influents. Because of its greater volume its carrying capacity is greater, and bowlders, cobbles, and gravel are rolled along its bottom, grinding each other as they go, and thus manufacturing finer and more easily transported material. The third and last stage is that in which the stream flows through a wider, flat, and almost base-leveled valley, on the last course of its oceanward journey. The water now moves with comparative slowness, hence can carry only the finest sediments and is forced to deposit part of its load.

The wide lower part of a stream valley is generally called a flood plain, for it is flooded during very high water and receives a small building-up increment of sediment from each flood. The heaviest deposition of sediment is along the line where the river overflows its banks, because the current is materially slowed down along this line. The ground is therefore built up higher near the river bank, and in this way what are termed "natural levees" are formed. The finer sediment is often held in suspension until the river current is checked when it reaches the sea. The sediment then, if not carried away by tidal currents, falls to the bottom at or near the river mouth to form bars or shoals.

RELATION OF STREAMS TO CANAL CONSTRUCTION.

The streams of the Canal Zone have a most vital relation to the canal, because they furnish the water to fill the dammed-in part of the waterway between the locks (see Pl. IV). Without the streams only a sea-level canal would be possible. The damming of the streams originated Gatun Lake and caused the backwater to fill Culebra Cut to navigable depth. The waste water from Gatun Lake will operate an immense hydroelectric plant which is to supply the whole Canal Zone with light and electric power, and is to be used to electrify the Panama Railroad. In the building of the railroad, roads, etc., and in some of the concrete construction and other work incidental to canal construction, it was found that gravel served the purpose almost as well as crushed rock and was much cheaper.

The general method by which streams roll rock fragments and bowlders along their bottom in their swifter upper courses and gradually grind them into gravel, and how they deposit the gravel where their velocity slows down somewhat and carry the finer silts down to their flood plains and to their mouths has already been discussed. The streams may, therefore, be looked upon as great gravel and silt manufacturing and sorting plants. They, of course, turn out and classify material vastly cheaper than the rock-crushing plants can. The Gamboa Bridge of the Panama Railroad crosses the Chagres River well above the normal flood plain, in the gravel-dumping zone of the river, and hence is convenient to bars of good ballast and gravel for construction work. In this important item the streams contributed directly to the construction of the canal. On the other hand, because of the liability to floods in the streams, it has been necessary to have the bridges spanning them extra strong, and therefore more expensive.

VALLEYS.

When water flows over a land surface it follows the line of least resistance, not only so far as the gradient is concerned, but also in respect to the ease with which it may erode a channel for itself in the material over which it passes. It thus tends to do minimum erosive work on the resistant areas of rock, so that in time these are left as remnants or hills between the streams. The softer rocks, being more susceptible to stream erosion, become the sites of valleys. In this way, through geologic time, the Obispo River and its tributaries kept wearing down the drainage outlet from the Culebra Basin, thus lessening the excavation work necessary for the future Culebra Cut. The Chagres, the Gatun, the Trinidad, and their tributaries kept excavating valleys that are now the site of the great Gatun Lake. The Rio Grande River Valley and its tributary valleys were utilized, for these streams had centuries ago begun the excavation of what man has now elaborated into the south end of Culebra Cut, the Miraflores Lake, the terminal port, and the dredged out approaches to the locks on the Pacific side. Not only in the making of the canal, but in the building of the railway lines, the water and sewage systems, etc., the valleys were utilized as far as possible.

BAYS, HARBORS, AND SHORE CONDITIONS.

Several features of the bays and harbors of the Canal Zone have involved engineering problems. Of these the relative shallowness of Colon and Panama Bays and the natural exposure of Colon Bay to northerly winds, and the Pacific entrance of the canal to southerly winds are the most important. In order to make these bays more sheltered and safe for ships, great breakwaters have had to be built.

Now come such practical questions as: Are Colon and Panama Bays likely to grow more shallow so that much dredging will be necessary to keep them navigable, or will the ship channels in them gradually become deeper from tides and shore currents? It might also be asked why Panama and Colon Bays are relatively shallow, whereas Porto Bello Harbor, not far distant, is about 60 feet deeper.

The shallow bays are the slightly submerged margins of the adjoining flat shore area. In relatively recent geologic time (Pleistocene) these bay areas were dry land. Approximately, the same gentle seaward slope of the land is maintained below the water level of Colon Bay out to the edge of the continental shelf as occurs above it, except for the marginal coral reefs and flats that form a wedge-shaped fringe around the present shore line. This fringe of limy rock and coral is about 35 feet thick at its outer edge and rests on the normal rock bottom. It locally presents an abrupt face to the sea and its outward margin is a few feet below high-tide level. There are also a few low headlands with steep slopes formed by the erosive action of sea waves on some of the points of higher land. Toro Point, 50 to 60 feet above sea level, is one of these.

The land surrounding Porto Bello harbor is steep and mountainous, and its slopes continue relatively steep down to the flat bottom of the harbor about 90 feet below the surface of the water. In the upper end of this harbor, however, considerable silt has been dumped by streams. Two other factors have made both Panama and Colon Bays more shallow, while they have had relatively little effect on the deep harbor of Porto Bello. They are: (1) During the rainy season all streams are swollen to great size, and they come down red and mud-colored from their load of sediment. When they reach the ocean the sediment is deposited, as already explained, and each year a thin layer of silt is left on the bottom of the bay into which they empty. In the time since this process began enough silt has been deposited to make the water considerably shallower. (2) In very recent geologic time the whole land mass has risen, as proven by old beach marks of fresh appearance, which, near Colon, are about 6 feet above sea level, and on the Panama side are a little higher. This rising has left Colon Bay about 1 fathom and Panama Bay more than a fathom shallower than they were previously. This recent rise of isthmian land, which reached a maximum of about 30 feet in Los Santos Province, probably took place within the last 1,000 years, according to evidence in hand. In fact, it may have been partly accomplished since Columbus's ships first plowed the waters of Colon Bay in October, 1502.

The Rio Grande River and its tributaries do not now enter Panama Bay except as overflow from the new Miraflores Lake. This lake acts as a settling tank and thus considerably reduces the sedimentation in the bay. Sedimentation in the Colon Bay is not now nearly so active as formerly, because much of the land around the bay is nearly worn to base level. On the other hand, the land is, very probably, still slowly rising, and if so the harbors and the approaches to both locks are still being slightly decreased in depth. The rise, if any, must be so slow that a little dredging will easily take care of it. Then, again, the rise may stop or sinking may begin at any time. Tidal currents tend to fill in the channels at many places in the bays, but dredging easily keeps them clear, especially as the silt brought in is very fine.

GENERAL GEOLOGY OF THE CANAL ZONE IN ITS ENGINEERING RELATIONS.

GENERAL ENGINEERING RELATIONS.

Within the Canal Zone there are many different kinds of rock. Many of the rock units, however, occupy relatively small areas, so that any considerable excavation may pass through several of them. Most of the Canal Zone types can be included in 11 different groups of bedded and noncrystalline formations and 6 different groups of igneous or crystalline rocks. Each rock unit, however, may vary considerably from place to place, especially in such physical characteristics as shearing, jointing, texture, and hardness. They all differ somewhat in age, in character of material, and in origin. Figure 1 shows a general geologic section across the Isthmus, showing the different formations between the Atlantic and the Pacific shores. Figure 2 shows the rock succession in the Canal Zone.

All of the rock descriptions following have a bearing on the engineering work, especially on the excavation methods used in Culebra Cut. Each description is followed by a discussion of the practical considerations that particularly apply to that formation. However, the general phases of practical interest are summed up first.

The cost of breaking any rock for excavation depends on three chief factors, which are: (1) The hardness of the rock and its resistance to drilling; (2) the minimum size to which it must be broken to give greatest efficiency of results in handling; and (3) the tenacity or resistance to breakage of large masses of the material.

If the rock is hard the cost of drilling is increased not only by the longer time required but also by the larger cost of sharpening drills, greater drill breakage, more frequent delay from drill bits sticking and from broken drill points obstructing and sometimes necessitating the abandonment of the holes, and easier deflection of the drill hole if it crosses at an acute angle an open joint or fracture zone containing soft or crushed rock.

The degree of smallness to which the rock must be broken depends chiefly on the size that gives greatest economy in handling consistent with the capacity of the excavating machinery and the use, if any, to which the material is to be put. A steam shovel with a dipper of 4 or 5 cubic yards capacity can pick up, balance, and dump on a railway car a bowlder weighing several tons. The most economical

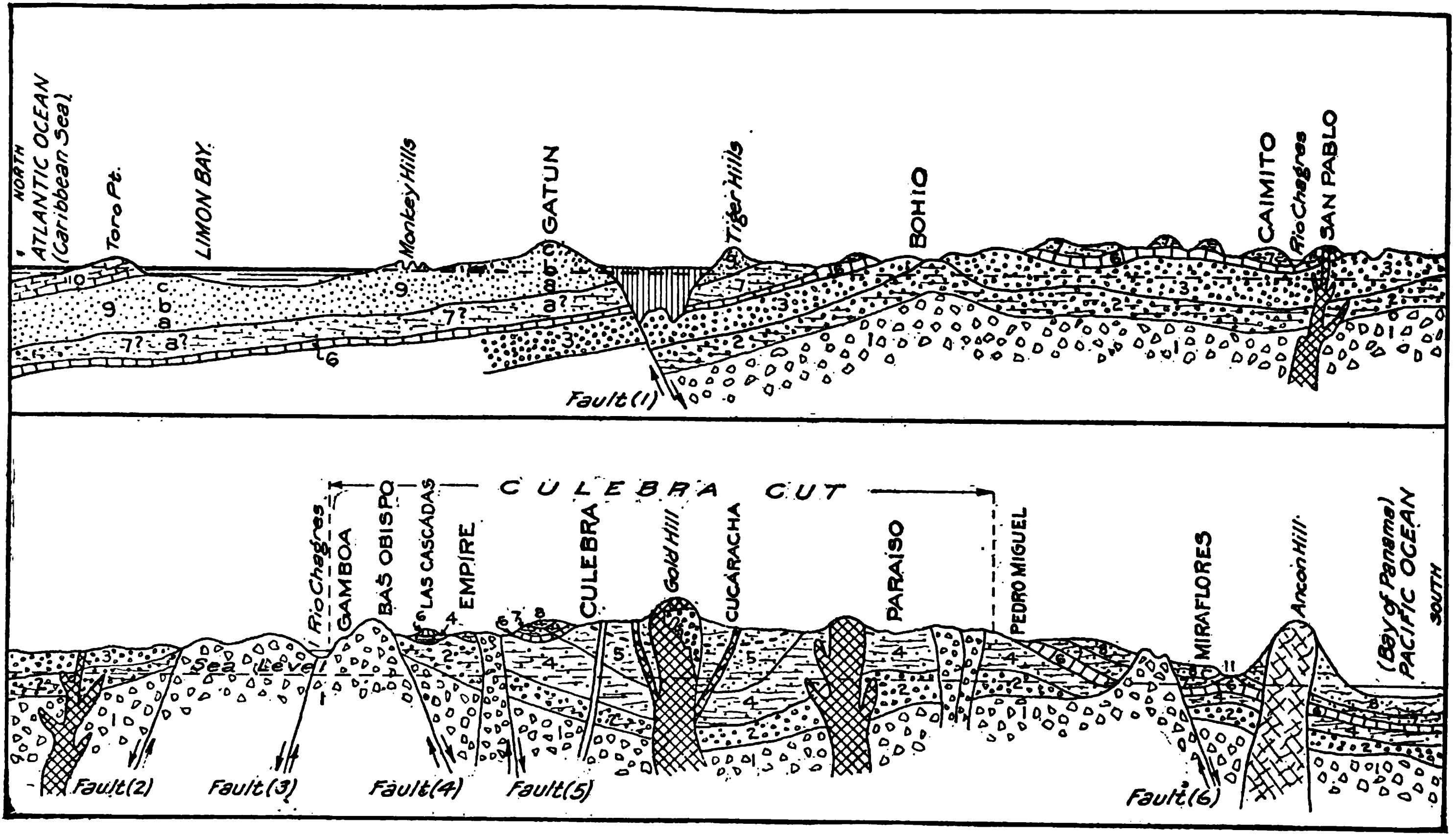


FIGURE 1.—Generalized geologic section across the Isthmus. Lower view shows continuation of upper view.

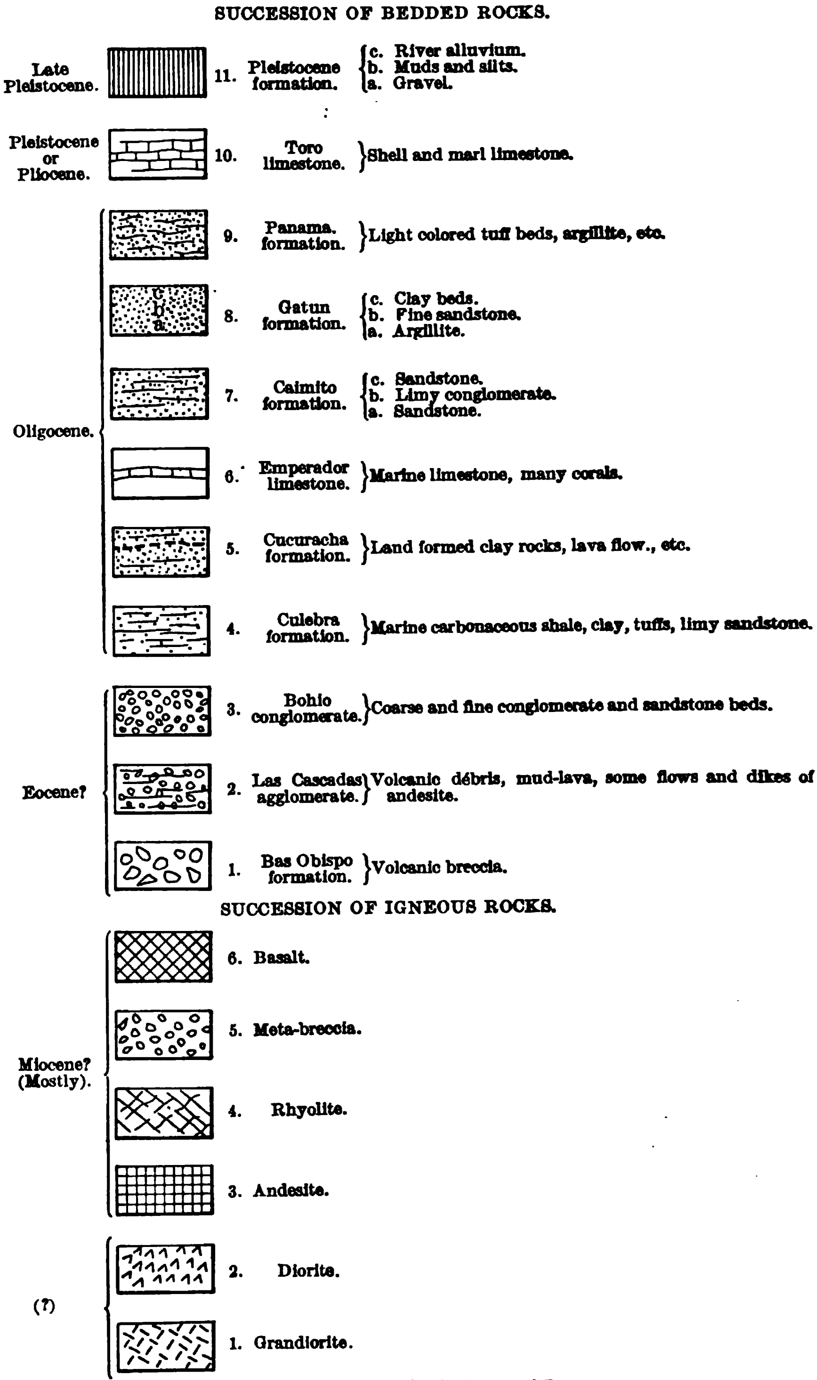


FIGURE 2.—Rock succession in the Canal Zone.

size to which the material should be broken ordinarily has to be determined by experiment.

The resistance of large masses of relatively hard rock depends not nearly so much on the tenacity of the solid rock as on the extent to which it is cut by natural joint, bedding, and shear planes. If it were desired to have the maximum size of the fragments not more, say, than 1 cubic yard, and if the joints happened to be well developed more or less at right angles to each other and so spaced that they divided the rock mass into fragments of not more than a cubic yard each, then the holes would not need to be so closely spaced and the charges could be much lighter to give the desired effect than if the joint planes were 2 or 3 yards apart or absent. If the material excavated is to be crushed and used for road making or concrete work, a high amount of jointing and fissuring lessens the amount of crushing necessary and greatly cheapens the cost of the product. On the other hand, if great coarse fragments are needed for armoring a breakwater against sea waves or an embankment against river currents, much jointing spoils any rock for such purposes, however suitable it may otherwise be. Soft argillaceous rocks with little tenacity are easily and cheaply drilled, and some such rocks on the Canal Zone were locally bored with augers. The holes in rock of this kind may be much farther apart and the explosive used may be a cheaper, lower grade and a slower acting one than would be required in hard rock.

ROCK FORMATIONS AND SPECIFIC ENGINEERING RELATIONS.

BAS OBISPO FORMATION.

The Bas Obispo formation is the oldest isthmian formation, so far as is known, and is probably of pre-Tertiary age. It was formed of rock fragments and ash blown from old volcanic vents. The débris settled over the surrounding region and was subsequently cemented into fairly hard rock by the slow-acting processes of rock cementation. Locally it shows some rough bedding and some rounded water-worn fragments. In composition it belongs to the group of rocks called andesites, and technically would be classed as an andesitic breccia. It outcrops extensively at Bas Obispo and near old Panama, and small outcrops rise above the alluvium near Miraflores and Diablo Ridge.

About 7,000 feet of the north end of Culebra Cut has been excavated in this formation. It is relatively hard and tenacious, except locally where it has been sheared by faulting. These locally faulted places have let down some masses and fragments of loose rock; but these were relatively unimportant as slides. On the whole, this material has stood fairly well at steep angles and will continue to do so. In blasting it broke out in relatively large fragments and did not readily weather or wash; so the excavated material from this section of the cut was in greater demand for fills and dams, for which maximum stability was necessary, than the softer and more friable rock.

LAS CASCADAS AGGLOMERATE.

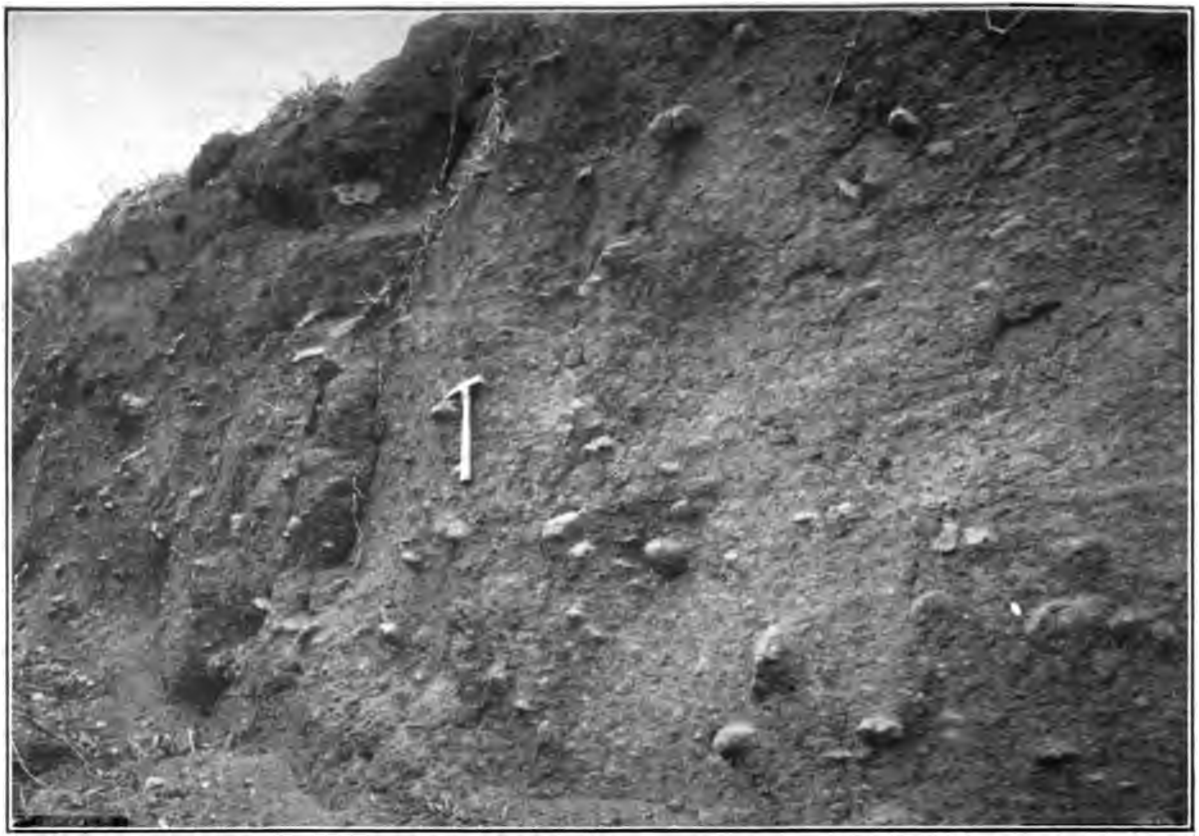
The Las Cascadas agglomerate also had its origin in fragmental rock material blown from volcanic vents and later washed down into different beds and masses. It rests uncomformably on the Bas Obispo formation, is much less consolidated and cemented than the latter, and is of much more recent origin. It is a greenish to gray, basic agglomerate, which contains large and small subangular fragments in a fine-grained groundmass of volcanic clay and tuff (Pl. V). The whole is arranged in massive to roughly bedded deposits, often unconformable with each other. Interbedded with these deposits are andesitic flow-breccias, some fine-grained grayish and some coarse-grained dark andesitic flows, and a few easily crumbled lava-mud flows which show columnar jointing where exposed in the canal (see Pl. XXVIII). The whole is cut by large and small basalt dikes. The formation outcrops extensively along the canal between Empire and Las Cascadas.

In degree of hardness and tenacity and in texture the rock of this formation presents great variety, but, on the whole, it is much softer and more friable and was cheaper to drill and blast than the Bas Obispo rock. Several considerable slides developed in it, due chiefly to local areas of weakness and faulting, and to its somewhat clayey character and ground-water content. These are described in detail under another heading. The extent and relative position of this formation is shown in figure 1.

BOHIO CONGLOMERATE.

The Bohio conglomerate consists of beds containing water-worn cobblestones and pebbles. These beds are separated from each other by layers of sandstone and clay rock. The lower part of the formation contains more cobbles and pebbles than the upper part and seems to be largely a product of running water. It is generally fairly well bedded, though locally massive, and contains hard bowlders, up to a few feet in diameter, of andesitic, dioritic, and other igneous rocks.

The upper part of this formation is of the same general composition as the lower part, but contains some beds of dark-gray clay marl rock which have the fossil remains of foraminifera—minute shell-bearing animals. The cobbles, bowlders, and gravel are from cherts, andesites, and diorites, and were derived from the older



LAS CASCADAS AGGLOMERATE. SHOWS CHARACTER OF VOLCANIC AGGLOMERATE MATERIAL EJECTED BY EXPLOSIVE OUTBURSTS FROM ANCIENT VOLCANOES THAT HAVE LONG SINCE CEASED TO EXIST.

intrusive masses of igneous rock now found at intervals along the central part of the Isthmus.

The formation must be at least 1,000 feet thick, and it outcrops extensively in the vicinity of Bohio and near Caimito Junction. Though not outcropping in the Culebra Cut, it was encountered in many of the cuts near Bohio on the new line of the Panama Railroad. It is somewhat difficult to drill, on account of the many hard cobbles and bowlders that it contains. Lack of jointing makes it rather more difficult to blast than might be expected from the fact that the matrix filling its interbowlder spaces is relatively soft and friable.

CULEBRA FORMATION.

The Culebra formation contains an upper and a lower member.

The lower member (fig. 1 and Pl. XV) consists of dark, well-laminated beds of soft shales, marls, and carbonaceous clays, with some pebbly, sandy, and tufaceous layers. There are a few thin beds of lignitic shale, but the whole upper part of the formation contains considerable organic matter. It outcrops in the Culebra Cut, near Culebra, and near Pedro Miguel.

The upper member consists of calcareous beds and lenses ranging in character from sandy limestone to calcareous sandstone, 3 to 10 feet thick, separated by partings of dark carbonaceous clays and fine-bedded tuffs (fig. 1 and Pls. VI and XV).

Locally this formation gives off a little natural gas and in some small areas it shows slightly bituminous shales.

The lower member of this formation, because of its soft, friable nature, could be drilled and blasted easily and was economically handled by the steam shovels. In spite of its weak and friable character it was relatively less given to sliding than, say, the Cucuracha formation, owing to the following facts:

(1) It is fairly well laminated or bedded, and the thin beds are relatively horizontal, except locally; hence there is practically no tendency toward sliding along the bedding planes.

(2) It is slightly sandy to granular and both its granularity and its bedding planes cause it to be better drained than the more clayey formation (the Cucuracha) in which maximum sliding developed.

(3) It seems to be relatively free from particles of chlorite and other greasy ferro-magnesian minerals which contribute greatly to the instability and mobility of the Cucuracha rocks.

The upper member of the formation contains many thin beds of sandy limestone and limy sandstone that are fairly resistant to drilling and blasting. They also strengthen the slopes and thus locally tend to prevent slides.

Certain local areas of the Culebra formation became heated, generally soon after having been exposed to the atmosphere by drilling or blasting. This heating was due to the oxidation of finely divided pyrite. It thus was necessary in certain areas to test the drill holes to determine whether they had become hot and therefore dangerous to load with dynamite. This matter of heating ground is discussed more fully in a subsequent section.

CUCURACHA FORMATION.

The Cucuracha formation (see figs. 1, 3, and 4 and Pl. XV) is here described in considerable detail, because in it not only the Cucuracha slide but also the big Culebra slides developed. It is so named because of being the site of the Cucuracha slide and because it is typically exposed near Cucuracha village.

The formation consists of a dark green, massive and locally bedded, slightly indurated, volcanic clay rock of andesitic composition. It is a land-deposited formation, overlying the marine Culebra beds, from which it is separated by 10 to 20 feet of slightly indurated gravel. It is the upper part of what Hill[a] and Howe[b] called the Culebra formation.

Locally it contains red beds and lenses, but these are of the same general character as the green clay rock in which they are interbedded, except that they contain slightly more iron and alumina and a little less silica. In certain beds there is a network of small irregular joints contiguous to which the greenish clay rock has turned red; such a change seems to be due to the oxidation of the greenish ferrous iron to the red ferric condition by surface waters. In some of these red beds, however, there has been some local concentration of iron and alumina products.

In addition to the red beds there are a few local beds and lenses of gravel and of sandy, dark gray, tufaceous material. This gravel, like the gravel at the base of the formation, is fairly fine, loosely cemented, and consists of the rounded fragments of indurated shales, cherts, and concretions from the lower part of this formation and from some of the older rocks.

There are also four distinct beds of lignitic shale, 1 to 5 feet thick. They are the fossilized remains of former swamps.

The formation is cut by some large and some small basaltic dikes (Pls. VII and XV), but these have caused scarcely any metamorphism. Faulting has considerably broken the beds and, owing to their soft and brittle character, relatively small faults, where the movement seems to have been less than 75 feet, have resulted in shear zones up to several feet wide. These rocks weather readily, and are

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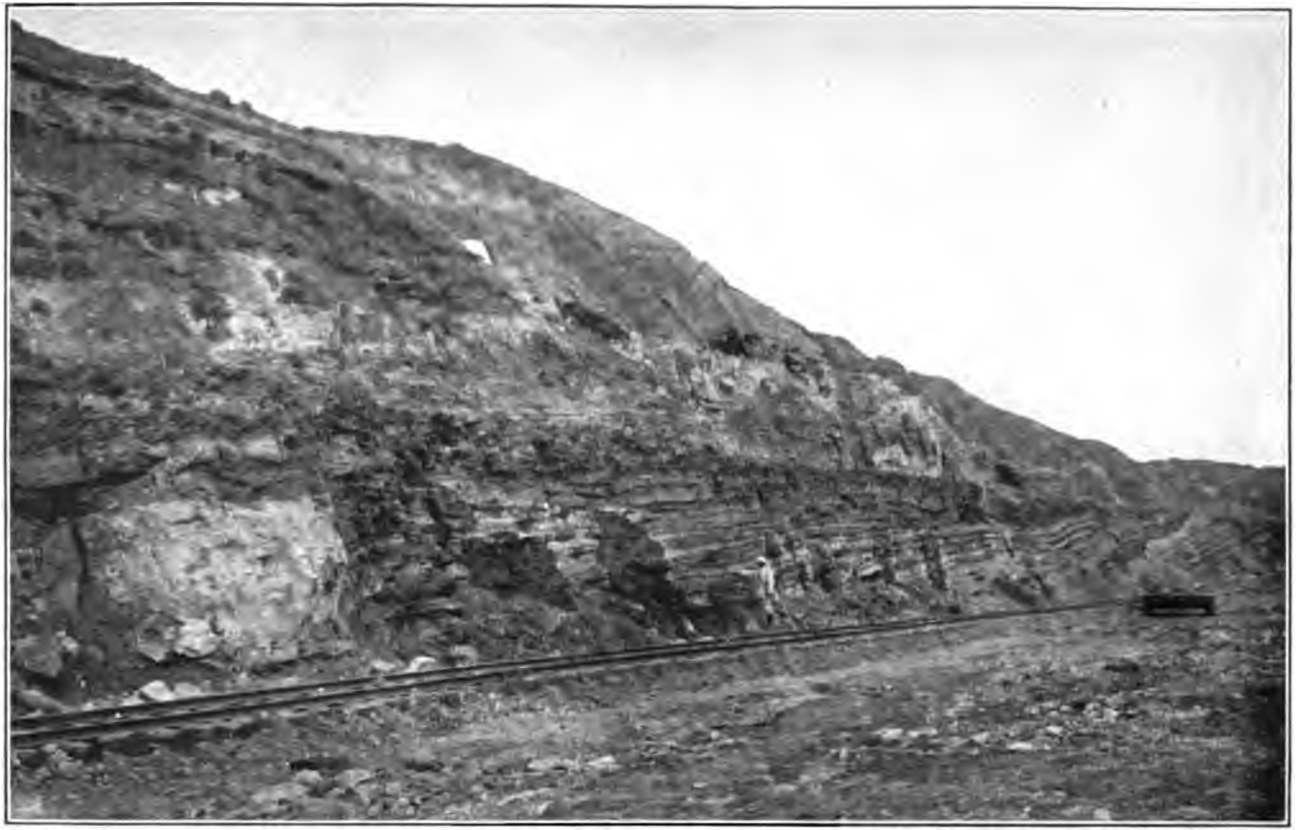
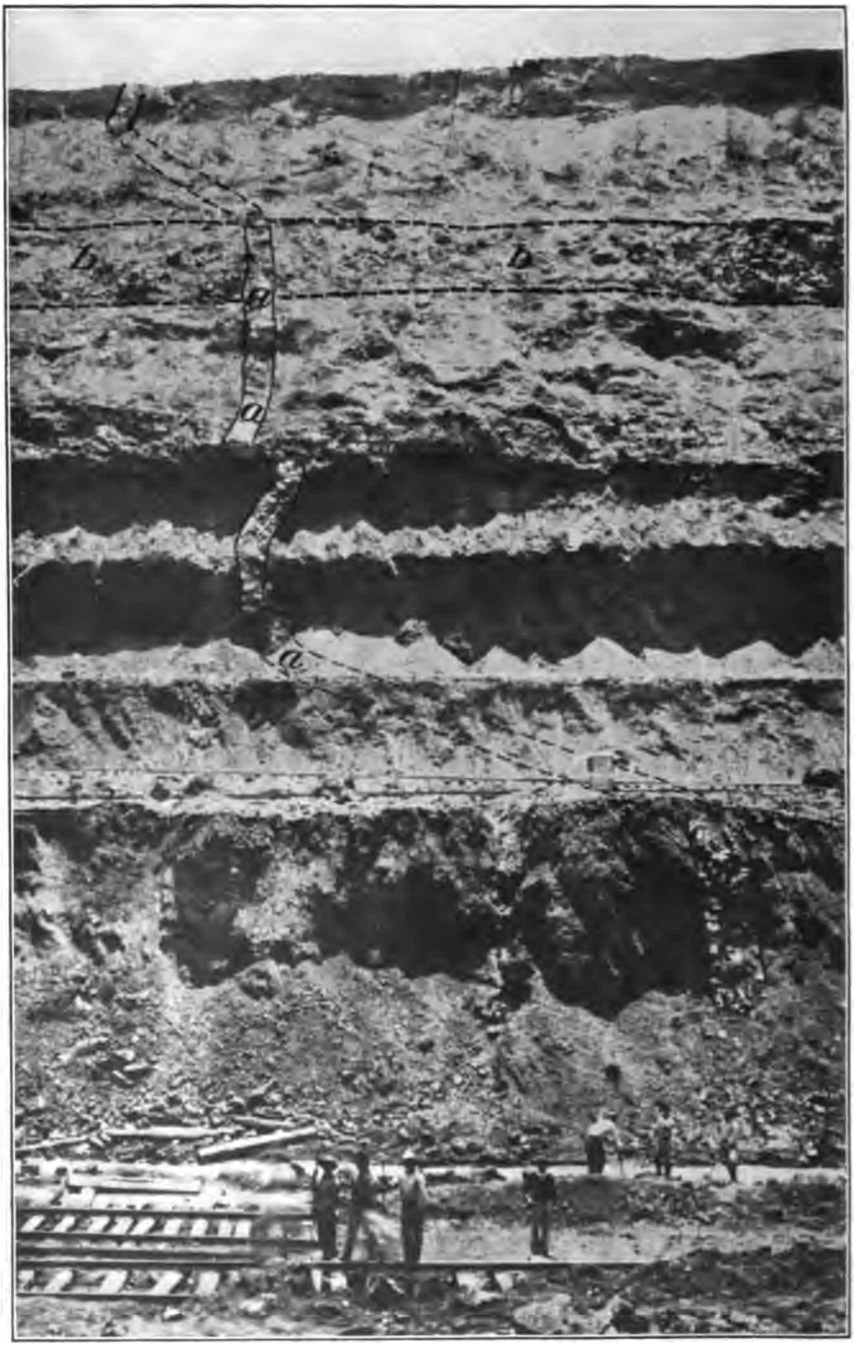
[a] Hill, R. T., Geological history of the Isthmus of Panama and portions of Costa Rica: Bull. Museum Comparative Zoology of Harvard College, vol. 28, 1898.

[b] Howe, Ernest, Canal Commission Ann. Report 1907, Appendix E, pp. 108-138; Isthmian geology of the Panama Canal: Econ. Geol., vol. 2, 1907, pp. 639-658; Geology of the Isthmus of Panama: Am. Jour. Sci., vol. 26, ser. 4, 1908, pp. 212-237.



SECTION IN CULEBRA CUT, NEAR CULEBRA.

Bedding of upper part of Culebra formation; thin limy sandstone beds separated by partings of dark carbonaceous clay and shale. These beds contain the remains of oysters, corals, and other marine animals. The dip is southerly, toward Contractors Hill.



DIKES IN CUCURACHA FORMATION.

a, Faulted and sheared dike of basaltic rock that cuts the cucuracha formation in Culebra Cut, near Culebra. *b*, Old lava flow in cucuracha formation.

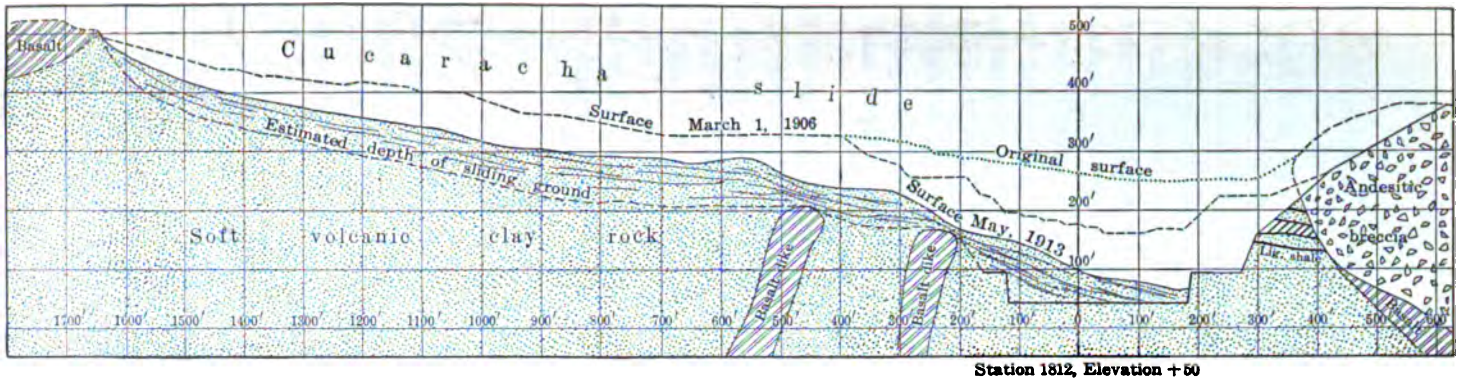


FIGURE 3.—Cross section of Culebra Cut through Cucuracha slide.

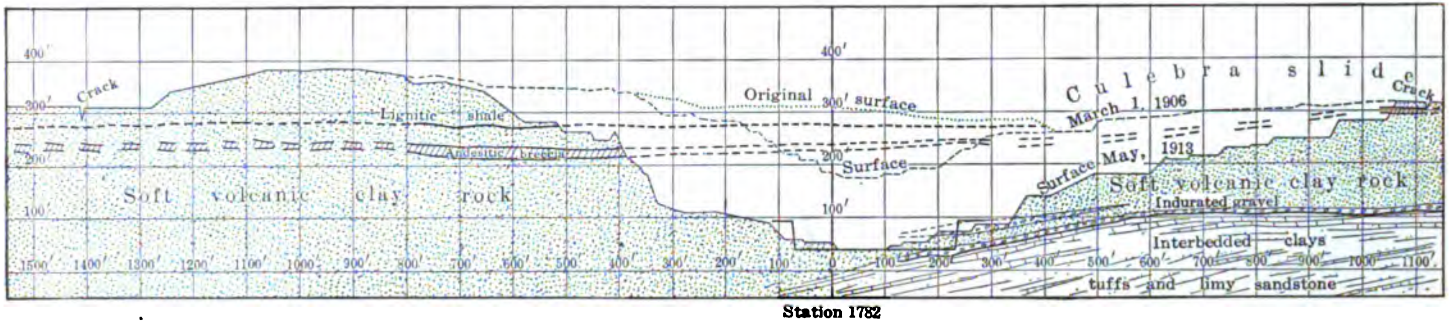


FIGURE 4.—Cross section of Culebra Cut at station 1782. As shown on left, a large crack developed at the surface, due to the fact that the rocks have deformed the excavation. This figure and figure 3 illustrate at what flat angles these rocks will slide.

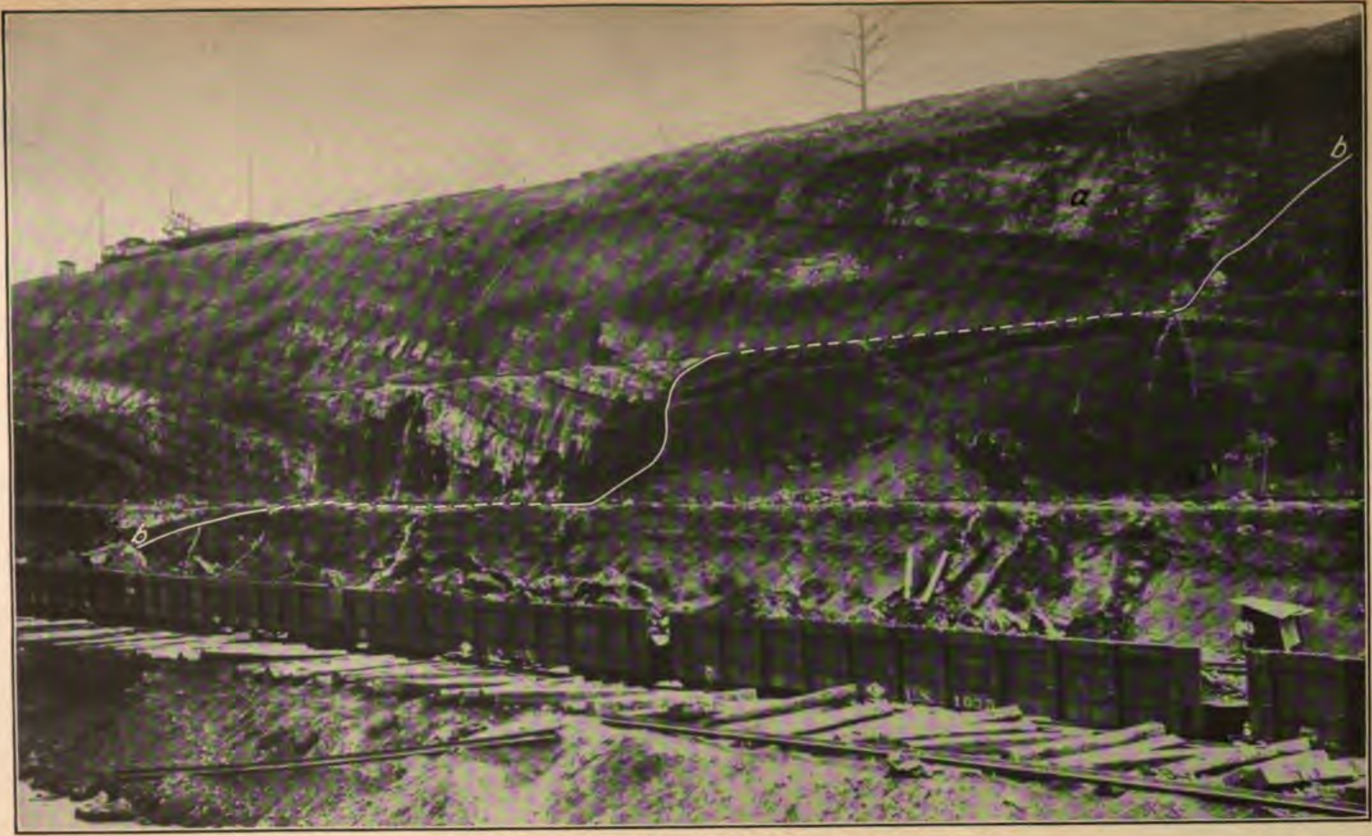
covered by 10 to 25 feet of red soil. They are easily eroded, so that the outcrops of this formation have mostly been worn into flats or valleys.

Extending for more than a mile over what must have been an old land surface, and now forming an interbedded unit of this formation, is a light to dark grayish, or, on fresh fracture, greenish, lava-breccia flow of andesitic composition. Hand specimens of it show a few little shiny faces of feldspar crystals up to 2 mm. in length, set in a groundmass that resembles indurated clay. The brecciated fragments are small, somewhat altered, and seem to have been picked up from the formation over which the flow moved. Under the microscope the rock is seen to consist of euhedral phenocrysts of andesine ranging in size up to 1 by 2 mm. and some crystals of potash feldspar set in a cloudy claylike groundmass, dark scaly areas resulting from the decomposition of some mineral, considerable chlorite, some calcite, and a little secondary quartz. The outlines of the brecciated fragments were recognized but their original composition was obscured by alteration. This altered andesite flow is somewhat jointed and weak, so that it adds but little strength to the slopes and is practically no protection against slides.

The prevailing grayish-green color of the formation is due to the fairly high percentage of very finely divided chloritic material that it contains. These greasy mineral particles are a marked source of weakness and mobility and are one of the factors that have caused maximum sliding in this formation. In contrast with the clays of the Culebra beds, these rocks are massive, largely of terrestrial origin, contain little organic matter outside of the few lignitic shale beds mentioned, have a greenish color from a high chlorite content, and are much more given to sliding than the other rocks.

EMPERADOR LIMESTONE.

The Emperador is a light-colored, fairly pure limestone. It lies unconformably on several of the older beds. Its outcrops are comparatively small and weather locally into a pitted and platy condition. Near Las Cascadas a section cut by the canal (Pl. VIII), shows the limestone, about 25 feet thick, overlying the upper part of the Culebra formation. It outcrops northwest of Empire, south of Las Cascadas, on the new line of the Panama Railroad, near San Pablo, near Frijoles, in the swamp southeast of Diablo Ridge, and extensively near Alhajuela. The formation also has prospective value as a local source of lime and possibly of cement. In places it is somewhat hard and tough, and is almost as resistant to drilling and blasting as is the Bas Obispo formation.



EMPERADOR LIMESTONE BEDS NEAR LAS CASCADAS, SHOWING FAULT.

a, Emperador limestone. It contains the remains of many sea animals, especially corals. *b*, Large fault which cuts off the beds here.

CAIMITO FORMATION.

The Caimito formation, which overlies the Emperador limestone, consists of three members, as follows: (*a*) A basal light-gray, soft, argillaceous, or claylike, sandstone, which grades upward into a yellowish argillitic sandstone that is bluish gray on fresh fracture; (*b*) a peculiarly calcareous conglomerate with some fragments of much decayed basic rock, which locally give a bright-green stain to small patches of the formation; (*c*) a light-colored to yellowish argillaceous sandstone fairly well bedded. The upper argillitic sandstone is the rock that outcrops in the Chagres River at Barbacoas, near San Pablo. Beds *a*, *b* and *c* may be seen in the section at Bald Hill, north of Miraflores. Bed *b* outcrops extensively at San Pablo and near the site of the proposed wireless telegraph station opposite San Pablo. The formation does not outcrop at all in the Culebra Cut. These rocks are relatively easy to drill and blast, and the weathered product from the more clayey beds of this formation might be suitable for the manufacture of common brick.

GATUN FORMATION.

The Gatun formation (figs. 1 and 2) consists of three members, as follows: (*a*) About 500+ feet of marls and argillites, or clay rocks, and some beds of soft sandstone and conglomerate; this member is bluish gray but locally contains many brown specks, indicating fragments of organic material; it is rich in the fossil shells of ancient marine life; (*b*) mostly fine, soft sandstone, about 100 feet thick, containing a few fossils; (*c*) light to creamy gray indurated clay beds.

The formation is extensive and constitutes the foundation on which the Gatun Locks are built. Fortunately, its fineness of grain renders it relatively impervious to ground water. The upper part of the formation weathers into red clay, and except where this is cut by streams, it covers the solid rock to a depth of 10 to 25 feet.

It is possible that the red clay, the weathered surface covering that obscures much of the Gatun formation, would be a good source of material for the manufacture of common brick. This rock has been used extensively for fills on the new line of the Panama Railroad, and in other places. It was convenient and cheap to excavate and load and answered well the uses to which it was put. However, it can not be used for fills that are subject to the scour of a river or of sea waves, because it abrades and weathers easily. This formation has much engineering significance because the Gatun Locks and Dam rest on it as a foundation. Its strength and adequacy to meet such foundational requirements are discussed under the heading "Foundations of the Gatun Dam and Locks" (p. 39).

PANAMA FORMATION.

The Panama formation is a light-colored well-bedded tuff somewhat acid in composition. Locally it contains some argillaceous beds. It outcrops extensively from Miraflores to Panama and in a few other places. The formation is at least 400 feet thick and overlies the Caimito beds. It seems to be relatively porous, fairly well bedded, somewhat jointed, and of a crumbly or friable nature. It was not difficult to drill or blast and is somewhat too soft for good road metal.

TORO LIMESTONE.

The Toro[a] limestone is sandy and fragmental, being locally a coquina or shell marl. Its type locality is at Toro Point, but similar appearing limestone fringes the Caribbean coast, except where large valleys have caused it to be eroded or covered with alluvium. In places it forms low bluffs or headlands, especially at Toro Point, west of the Gatun Dam, and at the mouth of the Chagres River. It is the rock from which Fort San Lorenzo was built. Rock from this formation was used as a hearting for the Toro Point Breakwater. It was suitable for such a purpose because of the ease and cheapness with which it was blasted and loaded onto railroad cars for transfer to the near-by breakwater. The soft and easily abraded character of this material prevented its use except as a core or heart. The breakwater had to be faced or armored with much harder and tougher rock, an abundance of which was available at Porto Bello.

PLEISTOCENE FORMATIONS.

The Pleistocene formations consist of (*a*) swamp deposits, black soil and silt, filling old channels to depths of 325 feet below the present sea level; (*b*) river gravels up to 10 feet above the present normal river levels; (*c*) old sea beaches 6 to 10 feet above the present beach level; and (*d*) bars, beaches, and the present river alluvium. All of these deposits have a distinct bearing on the engineering problems of the Isthmus. The deep swamp deposits not only made surveys difficult, but added much to the cost of building railroad fills across them. The river gravels were useful for ballasting the road, for local concrete work, and for other purposes. Some of the beaches yielded vast quantities of sand for making concrete. The present river alluvium locally has an agricultural value; and some of the river sediment, deposited as bars and shoals which partly obstructed the Atlantic and Pacific entrances to the canal, had to be dredged away.

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[a] Called Caribbean limestone by the author in reports of Isthmian Canal Commission for 1912 and 1913. See p. 63 and p. 570 of report for 1913.

IGNEOUS ROCKS.

The igneous rocks within the Canal Zone belong to six distinct groups or families as described below.

GRANODIORITE.

Granodiorite is a quartz-bearing, locally porphyritic rock, a member of the broad group of granitic rocks. It outcrops in a few places in the Chagres River Basin, for many pebbles of it may be found among the gravels of that river.

Another rock that may be classed as granodiorite but in reality is a dacite (or quartz-diorite-porphyry), is that forming Cocovi Island in Panama Bay. It was planned to use this rock in some of the smaller parts of the dry dock, hence it is here described in some detail. It is light colored, weathers almost white, and shows in the hand specimen only occasional shreds of dark ferromagnesian mineral. Under the microscope it is markedly porphyritic, with phenocrysts (large crystals in a finer groundmass) of andesine-labradorite greatly predominating over quartz, orthoclase, shredded and altered remnants of hornblende, and what probably was biotite. Many of the phenocrysts are cracked and broken and show zonal inclusions of glass. The largest are 6 mm. in diameter and from this size they range downward into fine groundmass crystals. The groundmass seems to be of about the same composition as the coarser crystals and shows a sort of pepper-and-salt effect. The accessory minerals are magnetite and apatite; also some secondary chlorite is present. The rock is strong, fine grained, and will make a good building stone.

DIORITE.

Diorite, a rock much like granite, but with little or no free quartz and with much dark hornblende, is found as rounded fragments among the gravel of the Chagres River, and in the Bohio conglomerate. Both diorite and granodiorite form some of the core masses of the mountains in the interior of Panama. The diorite is a tough well-crystallized rock and will serve well for building purposes.

ANDESITE.

Andesite, a rock of about the same chemical composition as diorite, is known at a few places in the Canal Zone. It differs from diorite in having reached the surface as a lava-magma, whereas diorite cooled below the surface and is therefore generally much more coarsely crystalline. Volcanic necks and dikes of andesitic rock cut the Las Cascadas agglomerate in various places. Similar dikes seem to cut the diorite at Point Parfan, opposite Balboa. Rocks of this general character form the steep headlands at Porto Bello and constitute the material quarried and crushed there and transferred to Gatun for use in the concrete work of the locks. Huge pieces of this rock, from the Porto Bello quarry, were transported on lighters and used to armor the breakwater at Toro Point. It is a typical andesite, hard, strong, and dark colored. Shiny little crystal faces appear scattered over the surface of hand specimens of it. Under the microscope it shows many large crystals of andesine and andesine labradorite, together with large crystals of augite and some of hypersthene, all set in a fine-grained dark to brown groundmass. This groundmass contains minute crystals or microlites of essentially the same composition as the large crystals.

Rock obtained at Point Farfan, opposite Balboa, where the French had an old quarry, shows a groundmass relatively coarse for andesite. In fact, the large crystals or phenocrysts of andesine and augite are not much bigger than groundmass or small crystals in which they are set. Quartz is also present in this rock in irregular masses and in micropegmatitic intergrowths with feldspar. The rock is a quartz andesite in composition.

The well-crystallized character of both the Porto Bello and the Point Farfan rock renders them tough and resistant to abrasion by waves; hence they are well suited for breakwater or sea-wall construction.

RHYOLITE.

Rhyolite has the same chemical composition as granite, but is much finer in grain. Crushed rhyolite was used for the concrete work of the Miraflores and Pedro Miguel locks; therefore it is here somewhat fully described. It is the rock that forms Ancon Hill, and Culebra and Naos Islands, in Panama Bay. Ancon Hill is a slightly curved, roughly tabular intrusion, about half a mile long and several hundred feet thick, cutting the softer Panama formation of light-colored bedded tuff. It stands as a much elongated cone or short curved ridge, 654 feet above sea level and nearly 600 feet above the lowland that surrounds it (Pl. IX).

The rock is much fissured and jointed, so that it is easily crushed. It weathers to a light-buff color, but is bluish gray on fresh fracture. In the hand specimen it shows a fine texture with some small lathlike phenocrysts, the larger ones having faces about 1 by 5 mm. in size. Microscopically the rock shows some flow structure, especially around the phenocrysts, which are largely oligoclase or oligoclase-andesine, with a few small and considerably altered needles of hornblende. The phenocrysts are much in the minority and in size grade into the coarser particles of the groundmass. The groundmass is fine grained and slightly cloudy and decomposed. It consists largely of perthitic

ANCON QUARRY, ON THE SOUTHWEST END OF ANCON HILL.

aggregates of orthoclase and plagioclase, with some quartz and feldspar intergrowths. A small amount of free quartz is present. The chief accessory minerals are grains of magnetite scattered through the rock, some ilmenite, a little apatite, and some light-yellowish grains of secondary mineral showing locally in the groundmass.

This rock is strong, fairly tough, and on the whole is well suited for concrete work.

BASALT.

Basalt, a dark, basic, fine-grained lava rock, is rather plentiful on the Isthmus. As dikes it shows in Culebra Cut near Empire, Culebra (fig. 1 and Pls. VIII and XV), and Pedro Miguel. As more massive intrusions it forms Office Hill at Empire, the top part of Gold Hill; also hills near Paraiso, Pedro Miguel, Rio Grande, near the Panama Railroad track 2 miles north of Monte Lirio, and in many other places. In fact, most of the steep hills and ridges within the Canal Zone (Pls. I and II), except Ancon Hill, are formed of hard basaltic rock that formerly came up through the softer rock as dikes and cores of lava. The basalt near Culebra Cut contains practically no olivine, whereas that found in many other places is rich in that mineral.

This rock is hard and tough and has locally been used in building roads. If crushed, it would serve well for concrete work. Its use to armor the projected new east breakwater at Colon was proposed, but the rock is so cut with joints that it would not break in large enough pieces to resist the abrasion of sea waves. However, it was to be used as the first coat of armor over the softer and more easily abraded heart or core material, the whole to be faced with large blocks of the Porto Bello rock. Basalt was also used to a considerable extent for facing the water-level zone of the Gatun Dam. Because of its engineering importance it is described in some detail.

Gold Hill is capped by about 250 feet or more of basalt, and a curved basalt dike 75 to 100 feet thick forms the westerly and northerly periphery of the hill mass. The dike and the capping are fairly typical of the variety of basaltic rock occurring within the Canal Zone that does not bear olivine. Locally the basalt shows some columnar structure (Pl. X) and is considerably jointed. It is tough and strong and resistant to weathering, so that it generally forms hills and ridges which have only shallow coverings of soil. It cuts sedimentary beds of Oligocene age and was perhaps erupted in Miocene (?) time. It shows relatively little contact metamorphism.

Hand specimens of this rock show it to be dark, compact, fine grained, and even textured, and it is comparatively little weathered. Microscopically it shows fluxional or flow structure, markedly in specimens from the capping on the hill and to a much less extent in specimens from the dike. In the flow capping the feldspars are abundant, labradorite lathlike crystals, ranging up to 1.2 by 2 mm. in size, predominating. In the dike specimens the crystals are a little stouter and larger, and many of them are cracked and bent. Augite is fairly abundant and occurs mostly as pale grains and granular aggregates. The accessory minerals are small black specks and irregular aggregates of magnetite plentifully peppered through the mass, a little ilmenite, and some apatite. Epidote in light and dark yellow irregular patches is present as an alteration product from some of the feldspar crystals, for it fills the little cracks of those that are crushed and broken and occurs as cloudy masses and aggregates in the interior parts of others. Three thin sections were examined, but no olivine was noted in them.

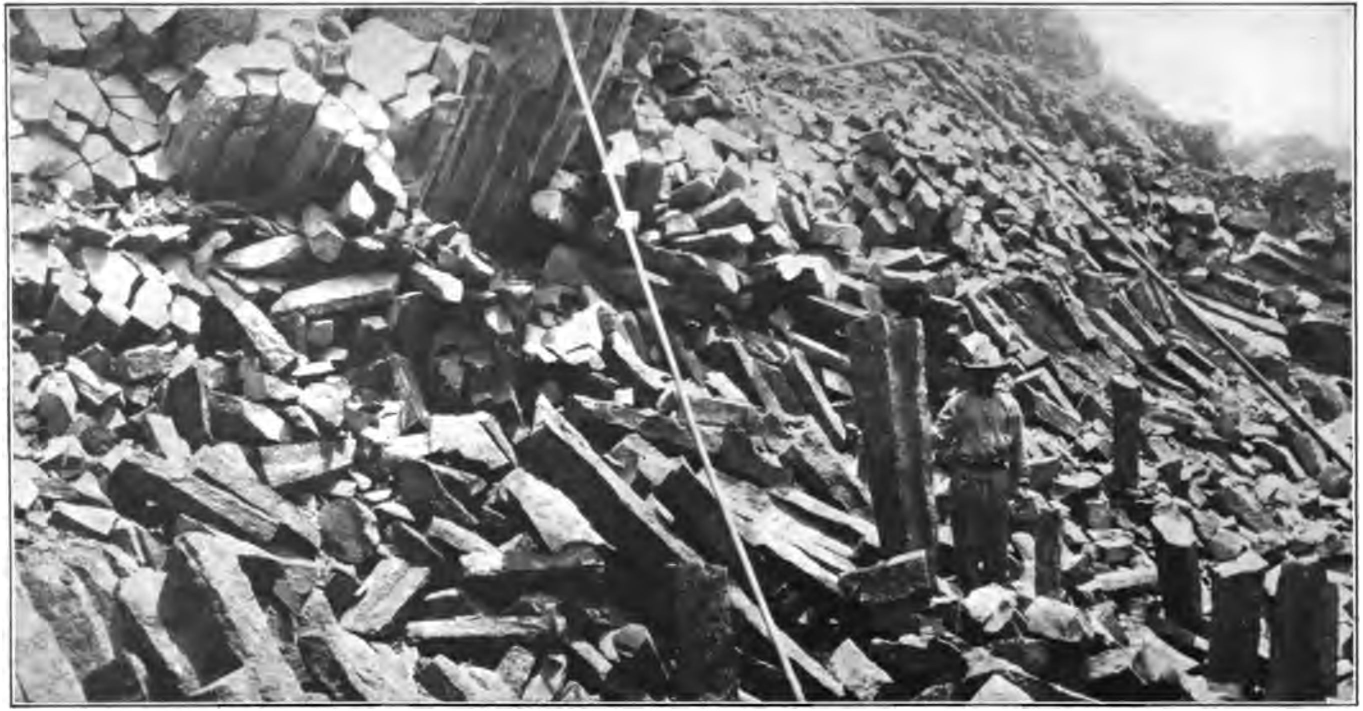
Basaltic rock from a little hill near the railroad track and 1½ miles north of Monte Lirio was used to face the water-level zone of the Gatun Dam. This rock is somewhat coarser in texture than that just described, and its feldspars are more calcareous. Otherwise the rock is much like that of Gold Hill.

META-BRECCIA.

Under meta-breccia are classed the metamorphosed tuff, sandy and argillaceous beds, agglomerate, and breccia masses that form Gold Hill, Contractors Hill, Office Hill (Culebra), and the breccias at Pairaiso, at Empire, and at other places (see Pl. I, II, and XV). Around at least a part of the periphery of several of these masses, and projecting through each of them in one or in several places, are basalt dikes. All of these masses, so far as is known, are separated from the rocks surrounding them by a contact along which faulting has occurred. Both the Gold Hill and the Contractors Hill masses have been faulted downward some hundreds of feet (see Pls. XV, XIX, and XX), These meta-breccias are similar to certain phases of the Bas Obispo breccia. Disturbed bedding is found in their upper parts, so that a vertical section through them would look somewhat like a vertical section through the upper part of the Bas Obispo, and a part of the overlying Las Cascadas agglomerate, with some higher-bedded tuffs included. The evidence indicates that these meta-breccia masses were pushed upward as somewhat metamorphosed and toughened caps on top of basalt plugs or cores. On cooling, shrinkage and outlets for various dikes and apophyses caused a gradual settling back of these plugs to about their present condition. These breccia masses have acted as strengthening pillars to buttress some of the sliding areas in Culebra Cut. This function is discussed in a subsequent consideration of slides.

LAVA-MUD FLOWS.

In addition to the igneous rocks enumerated, there are several cooled and hardened mud-lava flows which are found in the Las Cascadas agglomerate. Many of these local masses show columnar structure (see Pl. XXVIII) and are rather hard, but on exposure to the air for a few years they crumble considerably.



BASALTIC COLUMNS EXPOSED IN CULEBRA CUT, NEAR CUCURACHA SLIDE, IN MARCH, 1913. CLOSE VIEW.

STABILITY OF THE ISTHMUS.

A brief reconnaissance of the geology of western Panama was made possible by a cooperative arrangement between the Isthmian Canal Commission, the Smithsonian Institution, and the United States Geological Survey. The information thus gathered, coupled with that obtained from a study of the Canal Zone geology, leads to the conclusion that the isthmian land first appeared as an archipelago of islands in a shallow sea. During the Oligocene period, when the Cucuracha formation was laid down, there seems to have been established, possibly for the first time, a land connection between the two continents. At least there is, as yet, no positive geologic evidence of an earlier connection. Before the close of the Oligocene, the sea again swept over the submerged land, leaving only a few islands above the surface of the water. It appears that the Miocene began with an emergence, and that until its close the isthmian barrier once more effectively separated the oceans. With the beginning of Pliocene time, the land again sank until only the highest points were left above the surface of the sea. Another uprise ushered in the Pleistocene, and before this period was well advanced the land stood several hundred feet higher than now. Again sinking began, and continued until the land was 6 to 30 feet below present sea level. The last uprise, according to the fresh appearance of the old raised sea beaches and other evidence, must have taken place within very recent time, probably within the last thousand years.

In all, there is a clear record of four oscillations and the beginning of another elevation. Hence the question: Is the canal in danger from this uprise? Should the emergence be rapid, of course it would be in some danger; but the fact is, that the average rate of uplift for say, the last 1,000 years has been something less than 0.03 foot a year, or less than 3 feet in 100 years. Dredging could, of course, take care of this rise with little additional expense above the ordinary dredging necessary for the annual upkeep of the canal. Then, too, there is always the chance that this motion will stop entirely, and a sinking begin. Arguing then from what has happened in recent geologic time, it may be concluded that the canal is not in any appreciable danger from the geologic instability of the isthmian land.

It is believed that the sinking of the ocean bottom outside the relatively shallow depths of the isthmian shore waters has been the chief cause of the earthquake periods that have, so far as the record go, visited the isthmus every 30 to 35 years, and that each of these seismic disturbances has resulted in some increased uprise of the land mass. This matter is more fully discussed under the subject of "Earthquakes."

STRUCTURAL GEOLOGY IN RELATION TO ENGINEERING.

Under the subject of structural geology the folding, faulting, shearing, jointing, and some of the other changes that the rocks have undergone since they were formed is briefly discussed. All of these changes have some bearing on the engineering problems of the canal, but it is necessary to discuss many of them in connection with other topics treated under different headings, so only the main features and relations are here summarized.

FOLDING.

The bedded rocks were, of course, deposited in a relatively horizontal position. Since their deposition they have been folded and warped to various degrees.

From Gatun to the Caribbean the rocks of the Gatun formation dip gently northward and pass under the ocean at a slight inclination. The upper part of this formation is relatively fine grained and the lower part averages coarser. From such an arrangement of material, one would expect artesian conditions at Colon, especially now that Gatun Lake is almost 82 feet higher than that city. However, the rocks may be too fine grained and dense to permit flow of water through even their coarser lower beds. It is said that a well, some 600 feet or more deep, sunk by the Panama Government in Colon, furnishes considerable water, but develops no pressure at the surface.

A noticeable downfold or syncline in the bedded rocks crosses Culebra Cut in the vicinity of Gold and Contractors Hills. The axis of this trough trends almost northeast and southwest. Its greatest engineering significance is that it causes the stronger limy sandstones of the upper part of the Culebra formation to be bowed downward far below the bottom of the canal, so that they do not strengthen the slopes in the deepest part of Culebra Cut. The slopes where the worst slides have developed are composed only of the rocks of the Cucuracha formation, already described. The stronger limy beds lie below this formation, and outcrop as strengthening factors in the slopes of Culebra Cut on both sides of this downwarped trough or syncline, which is nearly 2 miles across.

There are many other minor flexures and small folds which have tilted the beds at angles up to 45° or more, but in general the bedded rocks are relatively flat and horizontal.

FAULTING AND SHEARING.

From the various oscillations of the land above and below sea level, as already noted, one would expect that the rocks constituting the isthmian land mass would have become broken and dislocated in these upward and downward movements, and this has happened. Great fracture planes cut the rocks and trend mostly northeast-southwest, or approximately parallel with the axis of the land mass, with some minor fractures leading in other directions. Along these fracture planes differential movements, some of which measure hundreds of feet, have taken place. The frictional drag of these movements has crushed and broken the softer rocks for several feet on each side of the plane of motion.

This faulting has had a most important bearing on engineering, because the faulted zones, especially where the rocks are extensively crushed and broken, have tended to promote slides. The present faults date from Miocene time or later, and most of them are probably of Pleistocene age. At any rate, so far as the digging of Culebra Cut is concerned, these faults have increased the cost of the canal several millions of dollars. The relation of the faults and sheared zones to the slides is fully discussed elsewhere in this report.

The chief faults trending across the canal are shown in figure 1 and Plates XV and XVI, but there are many more than those indicated there.

SUBMARINE ESCARPMENT AT PANAMA BAY.

Panama Bay is shallow out to its mouth. Beyond that the depth increases from 100 to over 1,000 fathoms in a short distance. This deepening is indicative of a great submarine fault escarpment passing just in front of Panama Bay. That there is an important fault escarpment there is established by the following facts:

On October 1, 1913, a considerable earthquake shocked Panama. It was especially severe in the southern part of Las Santos Province, where it seriously damaged several buildings. With the beginning of this earthquake the submarine cable from Panama City up the west coast broke. It parted just where it passes from the relatively shallow water of Panama Bay to the deep water beyond. When the break was located by the repair ship it was found that half a mile of one of the broken ends, which lay along near the foot of this submarine escarpment, was completely covered by débris and had to be abandoned and a new piece spliced in to take its place. It is, therefore, evident that the cable broke as the result of differential movement along this old fault plane, and that this movement caused the earthquake, and jarred down the submarine material that buried half a mile of the broken end.

About 1882 an earthquake shock, accompanied by the breaking of the cable, occurred in about the same way and at approximately the same place.

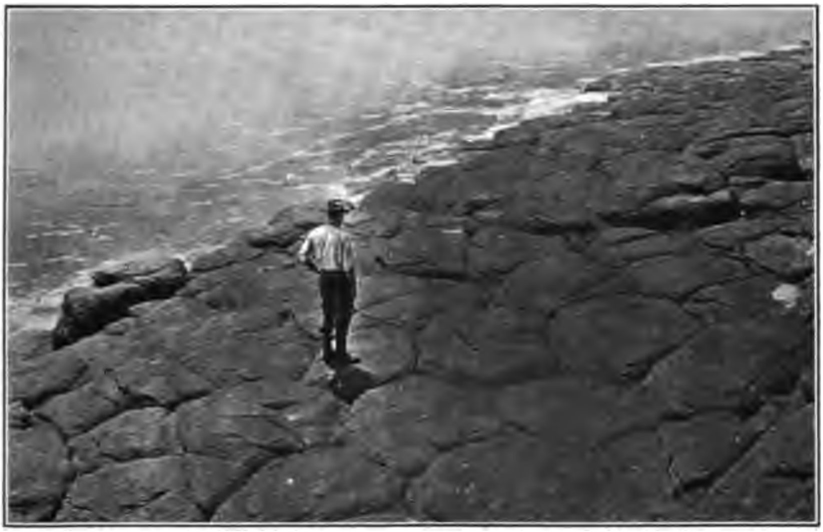
FISSURING AND JOINTING.

As mentioned previously, within the Canal Zone there are many dikes, cores, flows, and small masses of igneous rocks. These, while still in a molten condition, were projected upward through the bedded rocks. On cooling they were much cut by the developing of joints and shrinkage cracks, and later many of them were somewhat sheared by faulting.

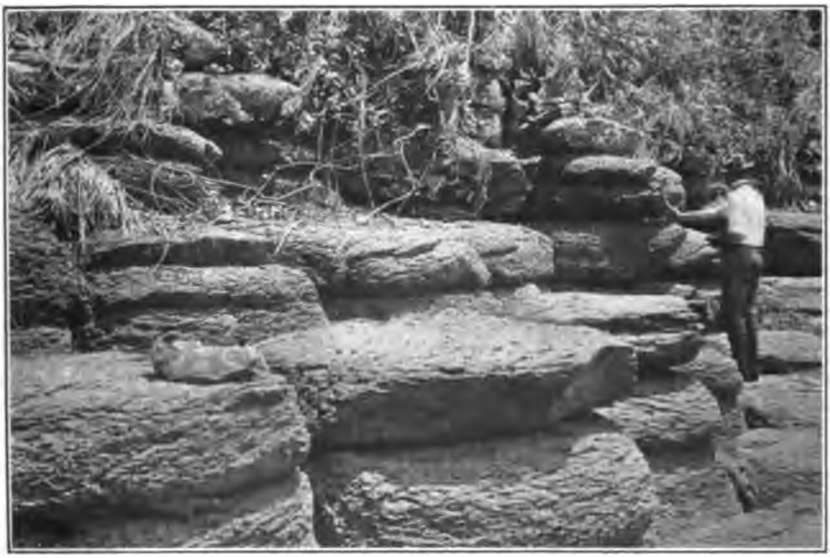
The jointing and fissuring (Pl. XI) of these rock masses has had a direct bearing on engineering results. The great amount of jointing and fissuring in the Ancon Hill rhyolite cheapened considerably the cost of quarrying and crushing that rock for use in the concrete of the Pacific locks. On the other hand, one of the reasons why the Porto Bello rock was more expensive to use was because it had few joints and fissures, and hence broke in large masses that were expensive to load and to break into smaller pieces. The difference in jointing between these two rocks was due to the fact that the Ancon Hill rock is a relatively small, tabular-shaped mass which cooled quickly and developed many joints and shrinkage cracks. It was then not strong enough to resist the crushing effects of faulting, and subsequent movements of this kind still further sheared and crushed it. On the other hand, the Porto Bello rock mass was much larger and cooled much more slowly; consequently it developed relatively few shrinkage cracks and was little affected by faulting.

Certain dikes and lava flows that for a time held back the Cucuracha slide would probably have been strong enough to have prevented further sliding there but for the fact that they were cut by many cooling and shrinkage joints and fissures. These dikes were large and for a year or more they held back the sliding material. However, when the cut had been brought to final depth in front of them, the accumulated stresses of the material behind caused them to shear off at points where they had been greatly weakened by joint planes (Pl. X). This gave a renewed activity to the Cucuracha slide that brought down into the canal over 1,000,000 cubic yards of material.

It has already been pointed out how the excessive jointing in some large outcrops of basalt a few miles south of New Frijoles prevented the selection of this material as armor for the east breakwater. Several other basalt outcrops were, for a similar reason, also found unsuitable for such use. Because of minimum jointing, Porto Bello rock when blasted yielded a large proportion of fragments that weighed several tons; hence coarse "armor" rock for the west breakwater was taken from Porto Bello. It was thought that the coarser product of the jointed basalt south of New Frijoles would be used for the first covering over the softer heart material in the east breakwater. The coarse "armor" rock for this will, however, probably have to be taken from Porto Bello, or from Sosa Hill, Balboa.



A. COLUMNAR JOINTING OR SHRINKAGE CRACKS IN BASALT, MALONIS ISLAND, PANAMA BAY. NOTE LARGE SIZE OF THE COLUMNS.



B. HORIZONTAL JOINTING OR SHRINKAGE CRACKS IN BAŞALT, MALONIS ISLAND, PANAMA BAY. THE JOINTS GIVE THE APPEARANCE OF BEDDING.

(Photos by E. D. Christopherson.)

In the latter rock the cooling joints seem to be far enough apart to give massive fragments large and heavy enough for "armoring" a breakwater exposed to sea waves.

INTRUSIONS.

As already explained, many small and large masses of lava were forced into or through the bedded rocks. Many of these intrusions locally disturbed the attitude of the invaded rock beds, tilting them up at various angles. Occasionally contact with the hot lava baked and metamorphosed the disturbed beds, but most of the intrusions caused only slight change in the beds.

These intruded dikes and masses have locally acted as piers and buttresses to strengthen weak slopes against sliding. The most noteworthy of these strengthening intruded masses are Gold and Contractors Hills.

ROCK MATERIAL USED IN CANAL CONSTRUCTION.

In the constructional work of the canal much nonmetallic material of local origin was used. The relative cost of excavating, loading, and transporting such material to the places where it was to be used, as well as its physical properties and general suitability for different uses, were features to be considered in planning the location of the necessary rock quarries and gravel pits.

DIFFERENT TYPES OF ROCK MATERIAL USED.

In all, seven different types of rock material were used, as follows:

1. Hard, tough, crystalline igneous rock, broken so as to pass through a 3-inch or 4-inch round-hole screen. Vast quantities of this were used to mix with cement for the concrete work of the locks.
2. A considerable amount of fine screenings (inch mesh) were used for light concrete work, road work, locally for railway ballast, etc.
3. Great pieces of hard, tough, igneous rock, some weighing several tons, were used to armor the west breakwater, Colon Harbor, against the destructive effects of ocean waves.
4. Filling material of all kinds, mostly soft rock dug out of Culebra Cut, was used to fill in swamps, build up "made land" where necessary, to form a large part of Gatun Dam and for the heart or core of breakwaters.
5. Soft mud dredged out of canal channels or from borrow pits was pumped into certain swamp areas to fill them and to fill the spaces between the coarse fragments of Gatun Dam, thus creating a seal to insure water-tightness.
6. Clean sand, free from organic matter, was used for concrete and other construction.
7. Gravel was used for railway ballast and locally for concrete and for other purposes.

SOURCES OF MATERIAL.

ANCON HILL.

Nature supplied the Canal Zone with some excellent sources from which to obtain rock for concrete work. The best of these is Ancon Hill, which furnished the crushed rock for the Miraflores and Pedro Miguel Locks. It consists of a tabular mass of rhyolite nearly half a mile long and some hundreds of feet thick (fig. 1, and Pl. IX). The rock is rather hard, so that it resisted erosion or wearing down by the streams that cut away the softer rocks around it. Because of this hardness Ancon Hill stands over 600 feet above the low lands that nearly surround it. The rock in this hill is much jointed and broken and is easily blasted; relatively little further crushing by machinery is necessary to prepare it for use in concrete or road construction.

In this instance the faulting or breaking of earth blocks in past ages has directly aided construction by cheapening the cost of crushed rock for the concrete work of the Pacific Locks. The same period of faulting, however, has been a hindrance in the excavation of Culebra Cut, for it locally weakened the rocks there and gave them a greatly increased tendency to slide. A detailed description of this rock is given on page 28.

PORTO BELLO QUARRY.

The Porto Bello quarry and crushing plant, located in Porto Bello Harbor, about 20 miles northeast of Colon, furnished the crushed rock for the Gatun Locks. This quarry is on a large area of andesitic rock which had few shrinkage joints and was too large and solid to be much broken by faulting. The joints are fairly well developed, but far apart, so that the rock breaks into coarse pieces when blasted. Each blast loosened a great deal of rock, but a large percentage of it broke along the joint planes into big blocks weighing several tons each. Many of these had to be broken by "adobe" blasting or "bulldozing" before they could be loaded and crushed, thus adding much to the blasting and loading costs. Here nature rendered little assistance in crushing the rock and rendering it more cheaply available for lock building, as it had done in the case of the Ancon Hill rock. Had the Porto Bello rock been well jointed, the United States would have profited to the extent of many thousands or perhaps millions of dollars. This quarry, however, did good service in furnishing large bowlders, which were used to face the west breakwater at Colon Harbor.

UNSATISFACTORY HARD-ROCK DEPOSITS.

In June, 1913, the Isthmian Canal Commission wished to find a hard-rock quarry that should be as convenient as possible to Colon and to the new line of the Panama Railroad. Most of the rock areas on the Canal Zone stand up as ridges or hills. One of these ridges, 2 miles south of New Frijoles, was "spotted" from a railroad train. Then trails were cut to it and a few small strippings made. The rock proved to be basalt, but subsequent work showed that, like nearly all the basalt masses of the Canal Zone, it was much jointed and broke into fairly small fragments on blasting. This quality though very desirable in rock to be used for concrete or road work rendered the basalt at this place unsuitable for facing a breakwater—the chief purpose for which it was required—unless reinforced by large heavy bowlders. Several other hard-rock areas were examined, but were all found to be more or less sheared and jointed, so that they would break into pieces too small to be really suitable for warding off the attacks of sea waves.

SOSA HILL.

Sosa Hill, close to the Pacific entrance, was found to be a fairly solid mass of andesitic rock, which had few shear joints and in which the shrinkage joints were far enough apart so that it would largely blast out in pieces weighing some tons with a relatively small percentage of fine material. This then is the best source known within the Canal Zone of material for armoring against heavy sea waves. However, the andesitic rock from Porto Bello may break out larger and be more economical for this purpose than the product from Sosa Hill.

CHAGRES RIVER GRAVEL.

The gravel for railway ballast and other purposes was obtained from the Chagres River near Gamboa Bridge. This is practically the only source of gravel within the Canal Zone. The lower Chagres contains too much alluvium, and in the upper part most of the rocks are too hard and tough and the streams too weak, except in flood time, to furnish much gravel. The available deposits lie between the rocky upper and the alluvial lower parts of the river near the railroad bridge, and thus convenient to transportation. The Chagres deposit furnished most of the ballast for the new line of the Panama Railroad.

UNSATISFACTORY GRAVEL IN CANAL ZONE STREAMS.

The Canal Zone streams as a rule do not yield much good gravel for the following reasons: (1) They are too short and most of them rise in ridges of hard rock, which affords relatively little material that they can easily wear into rounded gravel; (2) except in the flood season most of the streams are too feeble to roll stones along and grind them into rounded gravel; (3) in the wet season, after heavy rains, the streams become raging torrents, which tend to sweep away into the Chagres River or into the ocean material that might otherwise be a source of gravel;

(4) much of the rock traversed by isthmian streams is soft volcanic tuff and agglomerate that is easily converted into fine sediment. This forms a large proportion of the material carried by the streams during flood time, and in the lower rivers it covers and obscures much of the gravel that might otherwise be available.

STABILITY OF CANAL FOUNDATIONS.

GEOLOGIC STRUCTURE OF FOUNDATION ROCKS.

GENERAL CONSIDERATIONS.

Geologic conditions surrounding the foundations of heavy structures largely determine the plan, cost, and efficiency of such foundations. For instance, if a dam is built on a foundation rock of limestone leakage may occur, and that leakage if not remedied is likely to increase, because limestone is slightly soluble in ground water. If the water comes from a surface where there is much decaying vegetation, its solvent power is greatly increased by the increased content of organic acids and carbon dioxide. Wherever such water can penetrate a crevice in limestone, it will slowly enlarge that crevice by solution. Consequently, most large caves occur in limestone. The limestone formations on the Canal Zone locally contain many caves, openings, and irregular pitted surfaces, because the decay of the luxuriant vegetation there gives the ground water a relatively high content of organic acids and greatly increases its power as a solvent. Hence considerable care is necessary in locating a dam on a limestone foundation rock in order to be sure that no leaks, which may later enlarge, will occur.

If a masonry dam is to be secured to the foundation rock along its bottom and at both ends, certain geological factors must be considered. If the rock is shaly, thin bedded, and cut by many joints or fault planes, there is danger that certain blocks or sections of rock, or the upper beds to which the dam is secured, may under great pressure slip on the rocks below them and cause the dam to give away. Along many fault planes and along some bedding planes there are smoothened partings that readily lend themselves to slipping. Furthermore there is not only the pressure of the water against the wall of the dam, tending to push it over, and thus straining to widen the fissures in the foundation rock to which the dam is secured, but there is also the hydrostatic pressure of the water tending to force itself into those fissures and bedding planes and thus to exert a lifting or floating strain on the dam and on the slabs of rock to which the dam is attached.

Another important item relating to dams is to see that excessive leakage does not occur either through the dam, around the foundatons, or through the rim of the dammed-in basin. In order to ascertain what the percentage of leakage will be, individual tests must be made. The procedure should be about as follows: (1) An accurate topographic map of the region to be examined should be made; (2) the geology of the region should be carefully plotted on the topographic map and on sections, not only from surface indications, but from borings as well, the number of the borings depending on the requirements, the surface conditions, etc.; (3) the bore holes and any test pits that may be sunk should be plotted on the geologic map and all data as to elevation of the ground water in the different holes, flow of water from holes, depth at which water is encountered, character of material passed through, character of jointing and fissuring, slope and thickness of all the beds, hardness of material, etc., should be carefully recorded; (4) the fineness or coarseness of grain of the material and the proportion of each for all the geologic units should be carefully determined by experiment; (5) the permeability and absorptiveness of the different kinds of material should be carefully tested.

For very heavy structures it may be necessary to take care that none of the foundation piers or sills are set on a faulted or crushed zone in the rocks or on any locally soft area that would settle faster than would the other parts of the foundation. It is also necessary to be sure that the nature of the foundation rock is such that it will not be subject to slow swellings, deformations, or sliding movements toward contiguous excavations or valleys or toward caves or underground openings.

It might be well to mention here another rather remote contingency. In planning permanent structures of great cost, such as great churches or other delicately adjusted buildings or monuments, effort should be made to avoid building them across any major fault plane showing evidence of considerable differential movement in recent geologic time. A differential movement of only a few inches along a fault plane might almost wreck a building, even though the motion were slow enough not to produce an earthquake.

FOUNDATIONS OF THE GATUN DAM AND LOCKS.

POSITION OF GATUN LAKE.

The northern rim of the Gatun Lake Basin is formed by the Quebrancha Hills, which are a few hundred feet above sea level and trend northeast-southwest in a well-defined ridge. The seaward or northern slope of these hills is relatively gentle. Their southern or Gatun Lake Basin slope is much steeper and is a relatively straight line, except where modified by stream erosion. They seem to be monoclinal and give strong physiographic evidence that their southern escarpment is the locus of a large fault. Unfortunately, because of lack of outcrops, no definite stratigraphic evidence could be obtained on this point. At any rate no part of Gatun Dam and no part of the locks is built across this probable fault zone, so that the locks would not be harmed in any way should fresh differential movement or faulting occur along it. A glance at the map comprising Plate IV will show how the whole drainage of the Gatun Basin, including the Chagres, the Trinidad, and the Gatun Rivers, joined forces and emptied out into the Caribbean through a gap in the Quebrancha Hills little more than a mile wide. In order to form Gatun Lake, therefore, it was necessary only to close this gap; but to do this was a huge task; it meant the building of the Gatun Dam.

THREE GREAT QUESTIONS INVOLVED IN DAM.

With the project of this great dam, three questions in geologic engineering presented themselves. They were: (1) As Gatun Lake would have an area of about 165 square miles and would be 85 feet above sea level and separated from the Caribbean by only a few miles, would the seaward rim of this lake be found to be porous, or composed of limestone and thus result in leakage? (2) Would the bedrock be adequate in strength and in water-tightness to meet the foundational requirements of the locks and dams? (3) What would be the best type of dam to build under the engineering and geologic conditions involved?

Regarding the first of these questions the following was found to be true. The oceanward rim of Gatun Basin was found to be in general a fine-grained argillaceous rock. It has already been described as the Gatun formation (p. 25). This rock is so fine grained as to be relatively impervious. It contains a few local lenses of gravel and conglomerate, but these seem to be discontinuous and therefore relatively harmless in the matter of allowing leakage. Fortunately, the limestone locally forming the upper part of the rim is not known to extend below the 85-foot, or lake, level in any important place. If it did, a troublesome leakage that might rob the lake of considerable water might develop in it.

The second question was also answered in the affirmative, for it was found that the Gatun rocks are relatively solid and compact and suitable for foundations of heavy structures. However, some writers, even recently, have affirmed that the Gatun rocks would soften into mud when Gatun Lake filled, and thus the locks and dam would sink their foundations and be damaged or destroyed. Opinions of this sort spring from a vivid imagination and not from facts. In the first place, the Canal Zone rocks for centuries have been saturated with all the water they can hold; nine months in the year the rains are so heavy that the percentage of absorption as compared with the run-off is small; the rocks during that time carry all the water they can possibly contain. Secondly, just how the water is going to separate and break apart the minute particles of a fine-grained rock which are cemented together, without deforming movements to grind and disintegrate the rock, is difficult to understand. The Gatun rock might possibly become soft through deforming movement and mixing with water, but such movements are impossible where there is no lateral force, but only a downward or compressional load from the weight of the dam. This weight will tend to keep the rock in the same solid condition in which it originally was and is now. Solution had a chance to complete its effects centuries before the dam was built. If the weight of Gatun Lake is going to force water into the Gatun formation and cause it to soften into a semimud that will allow the dam to sink, then all the islands in Gatun Lake and all the hills around it, which are of the same formation, should also have their bases softened in the same way, and sink into the depths. Reports that the Gatun dam and locks are in any danger from sinking have not the slightest foundation on geologic data.

The third question, as to the best type of dam to adopt, was answered by the application of a few simple engineering and geologic principles. It was recognized that the cheapest dam would be one of earth and rock. Therefore, instead of dumping the trainloads of material brought from Culebra Cut and nearer excavations on some spoil bank, they were dumped on the Gatun Dam. In other words, this dam site was made a great dumping ground where many trains could run out and have their loads plowed off by machine unloaders. It was evident that if the base was made broad enough the dam would be amply strong to resist any water pressure that might develop, and would be nearly as secure as a ridge of hills. Also, an adequately wide base would provide a satisfactory foundation for the dam. In the matter of water-tightness, two main facts were evident: If the dam was high enough it would sink into any local areas of soft and spongy rock or soil that might be under it, and thus prevent leakage around its base; and if very fine material could be calked in so as to fill the interstices between the coarser fragments, the dam would be practically water-tight. It was to be nearly half a mile wide at its base and about 105 feet high, so it had ample base area for its elevation to make the foundation secure and to resist the head of water that Gatun Lake was expected to develop. In order to get the interstices between the coarser fragments filled with fine clay so as to make the dam water-tight, the outside rims of it were built of coarse material (fig. 5). Then suction dredges, operating in the river channels and flats on either side of the dam, pumped mud into the basin thus formed. This drained into the openings between the larger fragments and formed a thoroughly compact center mass or core which is practically water-tight. Later this puddled core was covered with coarse material as a sort of facing and the water-line area of the dam was faced with hard basalt fragments. In this general way a spoil dam with a water-tight clay-puddled heart was built.

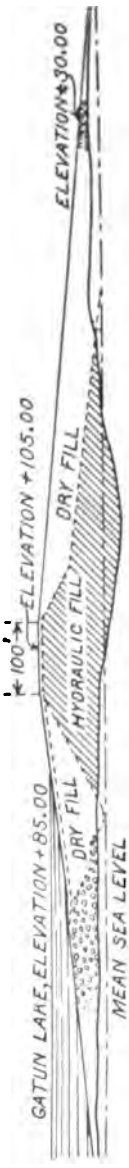


FIGURE 5.—Cross section of Gatun Dam showing dry fill and hydraulic fill.

DIFFICULTIES WITH HYDRAULIC FILLING MATERIAL.

The great magnitude of this dam may be realized by remembering that it is over 7,900 feet long, about 2,000 feet wide at the base, and 105 feet high. Figure 5 shows a cross section of it and gives some idea of its composition and plan. One of the construction difficulties was the excessive quantity of clay in the hydraulic filling material. This retarded drainage, and at times gave some difficulty in covering the puddled-in core with the dry-fill facing. The drainage of the core material was assisted by 20-inch drain pipes to remove the surface water and the very thin mud from the hydraulic-fill "pond" or sump. By varying the depth of this "pond" a greater or lesser amount of clayey material could be drained off. As the work progressed, the great mass of the dam slowly settled and became consolidated, and now seems to be almost as stable as a natural ridge or hill.

SOME DETAILS OF LOCK CONSTRUCTION.

The locks proper are founded on the bedrock of the Gatun formation, but the guide wall that extends out into Gatun Lake had to be built on an artificial foundation, because the bedrock is here about 150 feet below sea level. This wall is cellular and is built of reinforced concrete. The natural ground underlying it was about 8 feet above sea level. On this a wide fill with flat slope was made up to an elevation of 35 feet. Through this fill piles about 60 feet long and spaced 4 feet apart from center to center were driven and their heads were inclosed in a heavy reinforced concrete slab. On this the wall was constructed. As the work progressed a slow settling took place; however, the completed wall is now practically stable.

For the construction of the north guide and flare walls, foundational excavations to depths of about 70 feet below sea level had to be made before bedrock was reached. The material removed was too soft to permit loading with steam shovels and so was excavated by suction dredges, which cut their way in from the old French canal. When the excavation had been completed the dredges were taken out, the channel they had made was dammed, the water pumped out of the excavation, and the foundations laid.

Although the foundation rock of the Gatun formation is relatively impervious to the movements of ground water, still it does not absolutely prevent seepage. Therefore, in order to prevent excessive hydrostatic pressure from Gatun Lake on the bottom floor of the lock chambers, the flooring of the upper lock was made strong enough to resist full lake pressure. Behind the remaining lengths of lock walls drainage was provided.

The details of how experimental dams built of the material to be used in the completed dam were made and tested, also of the manner of building the Gatun Dam, may be found in the annual reports of the Isthmian Canal Commission, particularly the report for 1908.

DESCRIPTION OF GATUN-LOCK FOUNDATIONS.

Concerning the character of the lock foundations at Gatun, Howe[a] writes as follows:

> Excavation for the locks will be almost entirely in the fine-grained argillaceous sandstone except at the lowest point to be reached at the extreme southern end, where conglomerate occurs beneath the sandstones. The rocks as a whole are well consolidated and make excellent cores with the diamond drill. When taken out and exposed on the surface they remain firm and hard; in rare instances they have been found to crumble and break down into a sandy clay. The rocks are well compacted and capable of supporting heavy loads when confined, but being poorly cemented are unable to withstand erosion and should be fully protected where such action is anticipated.
>
> On account of the very considerable amount of clay present in all of the rocks it is believed that they will prove to be almost entirely impervious to water. An exception will be found in the conglomerate referred to. Although this rock contains considerable clay, it has been found where exposed at about sea level in test pit No. 1 to be water bearing. On April 1, 1907, a pump discharging about 12 gallons per minute was just able to hold the water at an approximately constant level in the pit. The surface of the conglomerate exposed in the sides and bottom of the pit was about 240 square feet. Wherever this conglomerate is encountered, it may be expected to be water bearing at all times. In the test pit this rock stands in vertical walls without timbering and as long as confined will support heavy loads.
>
> In order to obtain some idea of the compression that might take place under loads equal to or greater than those due to the weight of the lock walls, a freshly exposed surface of the argillaceous sandstone near test pit No. 1 was leveled and smoothed by means of a file. Upon a square foot of this surface 100 steel rails, representing a weight of 77,350 pounds, were balanced upon the side of a short I beam, to the underside of which a casting, having a smooth surface 1 square foot in area, was firmly bolted. After a load of 37,128 pounds has been applied a settling of 0.011 foot was recorded, measurement being made by means of a Y level with reference to a previously determined bench. From this point on, until 48 hours after the full load had been applied, no further compression was noted. It is probable that the set-

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[a] Howe, Ernest, Ann. Rept. Isthmian Canal Commission, 1908, pp. 124-129.

tling at 37,128 pounds was due to the pressing out of small inequalities on the prepared surface and that no measurable compression of the rock mass took place. The load applied to the square foot was equal to 537 pounds to the square inch, more than double the greatest pressure likely to be brought to bear upon the sandstones by the lock walls.

Further tests were made upon small pieces of rock taken from different test pits. None of these pieces was larger than one-eighth of a cubic foot. They were embedded in sand and the pressure applied to smooth surfaces one-half square inch in area. Seven tests were made upon pieces of conglomerate taken from the bottom of test pit No. 1. Some of this material had been exposed on the dump for 20 days. The samples failed at pressures of from 250 to 1,625 pounds per square inch; the mean of all the tests was 867 pounds to the square inch. Nine tests were made of the argillaceous sandstone taken from different test pits at the lock site and the site for the temporary spillway. Some of this material was taken from the walls of the test pits; other pieces had been lying on the dumps for a number of days. The samples failed at from 830 to 2,800 pounds per square inch. Two samples, both of which had been exposed on the dump for several days, showed no signs of failing when under pressures of 6,000 pounds to the square inch. Considering these as having failed at 6,000 pounds, the mean crushing load for the nine tests was 2,572 pounds per square inch.

INFLUENCE OF THE OLD CHAGRES RIVER CHANNEL.

As already explained, the Isthmian land during, say, middle Pleistocene time emerged until it reached an elevation of at least 375 feet above present sea level. As the land went up the stream currents were increased so that the streams cut deep narrow gorges for themselves. In this way the Chagres River intrenched itself some 325 feet below present sea level. After the emergence reached completion a slow sinking began and the streams, because of diminished velocity, began to fill their channels with silt. In this way the old Pleistocene channel of the Chagres was filled so that its presence was revealed only by borings made across the Chagres Valley at Gatun and at Bohio, where dam and lock sites were being explored. As to the character of the material which now fills the old channel or Pleistocene valley of the Chagres, Howe [a] says:

> When the Chagres had succeeded in cutting something more than 300 feet below this level, the land began to sink slowly and the gorges were gradually filled with gravels, sand, silt, and clay. At different horizons in these alluvial deposits are shells and trunks and branches of trees that indicate a gradual rather than a sudden subsidence. * * *
>
> * * * If subsidence takes place before the period of canyon cutting has ceased, as in the case of the Chagres, the velocity of streams is checked and they are capable of carrying fine material only, which is deposited far back from the mouths. In other words, one should expect to find at the bottom of such a gorge as the Chagres at Gatun a comparatively thin deposit of coarse gravel, sand, and a few bowlders, while the greater part of the alluvium should be of the finest clay and silt with a certain amount of fine gravel and sand mixed with it. Such, in fact, is the character of the alluvium filling the gorge of the Chagres, as shown by numerous borings made at Gatun. The boring records indicate considerable sand and gravel at points comparatively near the present surface; it is unlikely, however, that any sand or gravel,

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[a] Howe, Ernest, op. cit., p. 127.

unmixed with clay, occurs in the deposits except at or very near the bottom. The reason for the boring records specifying "sand and gravel" as they do, is that when samples are taken from time to time during the process of sinking the hole, only the coarsest material is collected, the finer clay being held in suspension and carried off with the water flowing from the hole. In certain of the holes temporary flows of water were encountered, but after a few hours the flow invariably ceased. Such flows are believed to occur when, in the course of making a boring, the casing is introduced into a lenticular deposit of sand or water-bearing gravel under pressure from the overlying beds. The sands, being surrounded by much less pervious material, are in the nature of reservoirs in which water is stored under pressure, and when this pressure is released at the point where the casing enters the sands water may rise to the surface, if under sufficient pressure, and flow from the top of the casing until the pressures are readjusted. Far from indicating porous materials underlying the site for the dam, the occurrence of such flows of water only proves the extremely impervious character of the materials lying between the surface and the water-bearing beds.

As a foundation for an earth dam, the geological facts show that the alluvium filling the Pleistocene valley of the Chagres at Gatun will be entirely satisfactory.

The rock in which excavation must be made for the locks is firm and hard, and only slightly permeable to water; it will stand in vertical walls without timbering and will support loads many times greater than those to which it will be subjected.

Spillway sites 1 and 2 in the Trinidad drainage are at low divides between the Trinidad and streams flowing into the Carribbean. The rocks at these two points are essentially the same as those occurring at Gatun. They are found in a fresh condition a short distance below the surface and have about the same hardness as the Gatun rock.

At spillway No. 3 the borings failed to discover fresh rock at the depth required for foundation, the material being a clay of decomposition derived from rocks probably the same as those at the other two spillways.

FOUNDATIONS OF THE PEDRO MIGUEL LOCKS.

The Pedro Miguel Locks are built on material that seems slightly less resistant locally than the foundation rock of the Gatun Locks, and may show local differences in settling. It consists of the upper member of the Culebra beds described on pages 21 and 22 and the andesitic dikes that cut them. These dikes have two chief influences as follows: (1) They are harder than the rocks that they cut, so, locally, they will constitute areas of minimum settling, and if proper precautions had not been taken they might give rise to cracks where the concrete structures cross them. (2) The dikes are somewhat in the nature of walls that will tend to lessen the seepage through the slightly porous sediments that they cut. They would greatly lessen seepage were it not for the fact that they are considerably jointed.

DESCRIPTION OF LOCK SITES.

Concerning these lock sites Howe[a] says:

Although the Culebra beds are variable in texture, those in which excavations for the Pedro Miguel Locks will be made are remarkably uniform. They consist of well-bedded sandy shales containing plant remains in a few places, and at some horizons lenticular masses of concretionary limestone are common. The physical properties of the rocks differ but little from those of the rocks at Gatun. Being more argillaceous the shales are naturally less permeable to water, and experience in sinking test pits showed that they were very tough and required more blasting than did the Gatun rocks under similar conditions. This greater toughness is to be attributed to the argillaceous nature of the rock; for the same reason their compressive strength is probably slightly less than that of the argillaceous sandstone of Gatun.

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[a] Howe, Ernest, op. cit., p. 128.

In the vicinity of Pedro Miguel the Culebra beds have a prevailing dip of 5° to 10° to the southeast, varied by sharp local folding; moderate fracturing has accompanied the folding, and slickensides are shown at many points where exposures have been made by recent excavation.

At the point where it is proposed to construct the lower gates a dike of augite-andesite or "trap" crosses the lock site in a direction slightly east of north. It has a width of about 30 feet where exposed on the southwest side of the cut, and its contacts with the shales are clean and sharp. On the northeast side of the cut the dike turns to the north and the contact is more irregular, small apophyses having been thrown off into the shales. The Culebra beds, which are here sharply flexed and considerably fractured, owe their deformation to some extent to the intrusion of the dike; the dike itself, however, is also fractured, and slickensided surfaces within its mass as well as in the shales indicate that movement has taken place subsequent to intrusion, accompanied, it is believed, by minor faulting along the west side of the dike.

These structural features have a direct bearing on the character of the foundations. The dike, on account of its fractured condition, which borings have shown to exist below the lowest points to be reached in excavating for foundations, will be permeable to water. The shales also have been made more or less pervious as a result of their deformation and consequent fracturing. On account of this fracturing local seepage of surface water is to be anticipated, but the character of the Culebra beds northwestward is such that no flow of water is likely to occur through them even when the canal in the summit region is filled with water. Although the general dip of the beds is to the south, borings have shown the rocks to be the clay shales in large part, and as such not water bearing. Furthermore, several intrusive bodies of andesite or basalt cross the canal line between Culebra summit and Pedro Miguel, which will effectually check any flow of underground waters.

The difference in hardness between the trap rock of the dike and the much softer shales of the Culebra beds must be taken into account in planning the lock walls. The possibility of even very slight differential movement in the zone of fracturing and dislocation in the dike and shales adjacent to it should also be considered. By constructing the lock walls in sections, damage which might be done to monolithic walls as a result of such movements or from the unequal settling of foundations of different hardness, will doubtless be obviated.

FOUNDATION OF THE COROZAL LOCKS AND DAM.

The Corozal locks and dam are built in part on the Caimito formation, described on page 25, and in part on what appears to be the upper layers of the Culebra formation. It is somewhat permeable but gives a relatively firm foundation. Of this lock and dam site, Howe[a] says:

> The rocks that underlie the alluvium at depths of from 30 to 60 feet below mean tide are sandy shales * * * Rhyolite tuffs are exposed in the hills at the northern end of the dam site.

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[a] Howe, Ernest, op. cit., p. 129.

Mechanical analyses of samples of alluvial materials collected at the site of the Corozal Dam and from the sand bars in the bay beyond La Boca are given below. They were made at the request of the consulting engineers, Messrs. Noble, Stearns, and Freeman, in order to determine the permeability of these materials and their availability for use in constructing the dams.

Mechanical analyses of sand and mud from La Boca, Canal Zone.

[Made at the laboratory of the United States Bureau of Soils.]

Constituent.	No. of sample.					
	2	3	4	5	6	7
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Organic matter	0.8	1.8	0.8	7.5	5.8	2.1
Fine gravel, 2 to 1 mm	7.1	1.1	7.8	3.1	2.8	18.9
Coarse sand, 1 to 0.5 mm	14.9	.8	39.9	.7	3.3	21.2
Medium sand, 0.5 to 0.25 mm	13.3	.6	26.8	.9	2.5	19.7
Fine sand, 0.25 to 0.1 mm	42.2	2.7	11.0	6.2	23.6	15.7
Very fine sand, 0.1 to 0.05 mm	.8	2.6	.4	3.4	7.1	2.2
Silt, 0.05 to 0.005 mm	1.4	43.1	4.7	31.4	25.9	9.6
Clay, 0.005 to 0 mm	3.0	49.2	9.3	54.4	34.6	12.6

No. 2. Coarse sand, 18 inches below surface, sand bar, La Boca.
No. 3. Alluvial clay from dredge, one-fourth mile south of dock, La Boca.
No. 4. Sand from river bank, near north end of Sosa-Corozal Dam site.
No. 5. Mud from surface of mud flat opposite ship's ways, La Boca near oil station.
No. 6. Mud from dredge, one-half mile south of dock, La Boca.
No. 7. Dry sample, hole No. 14, La Boca Dam, elevation—18.5, 20 feet below surface.

FOUNDATION OF NEW ADMINISTRATION BUILDING, BALBOA.

The new administration building at Balboa, a very heavy structure, was built on a little hill some 50 feet or so above the flat ground in front of it. One corner of the building is rather close to a steep slope, so it was thought that the weight of the structure might cause this slope to slide, and thus wreck the building. It was said that this corner of the building rested on bowlders and clay which might not support the added weight. The writer was detailed to investigate this matter. He found that the supposed bowlders and clay constituted only the weathered surface part of a jointed rhyolite mass. Weathering had progressed from the surface downward along the joint planes, thus leaving semirounded unweathered core masses, which looked somewhat like bowlders, between the weathered zones of the joints. With depth the weathering grew less and the rock became much more solid, so that the building was in no danger whatever. The following report, covering the matter, was then made by the author to the chairman and chief engineer of the Isthmian Canal Commission:

The rock under the surface soil on which the new administration building at Balboa stands is the same rhyolite rock that forms Ancon Hill, and it has about the same degree of jointing. The fragments of this rock, between the joints, are strong and tough. Any slides that might occur in it would be due to steepness of slope and would consist in a mere sloughing off of loose material. There would be no deformation or flowagelike movement such as that which gave rise to some of the large slides in the cut near Culebra.

Toward the surface the jointing in this rock becomes more pronounced, so that the individual pieces between the joints are separated by partings of earthy material. They thus have some resemblance to a mass of bowlders mixed with clay. In reality, however, this is not a bowlder-clay formation.

From the surface downward the rock becomes much less jointed and much more solid. In a cut of any depth or in the side of a mountain, this rock, under its present conditions of jointing, would maintain itself at a slope of 2 on 1, except, of course, the top covering of earth and broken rock, which would stand only at a much flatter angle.

Cross sections show that the maximum slope from the basement floor to the foot of the slope at the railroad is about 7 on 11. This then is well within the limit of safety so far as any slides are concerned.

Now, what effect will the weight of the building, 97 tons per column, have on this mass of rock? If the pier foundations have been put down below the earth and loose rock to the relatively solid, though much jointed, rock below, then there will be no danger of the building causing any slide movement here, for the following reasons:

1. This rock will not deform or flow under any pressure that this building can possibly exert on it, in spite of the fact that in large masses it has not much tensile strength because of jointing. No amount of weight that need be considered would cause a sinking in one part of this material, with a corresponding bulging in another part.

2. At any slope less steep than 2 on 1 this rock (apart from its soil and loose rock covering near the surface) would stand without any danger whatever from slides. The steepest slope from the basement floor to the railway level below is only about 7 on 11, therefore, the margin of safety is ample.

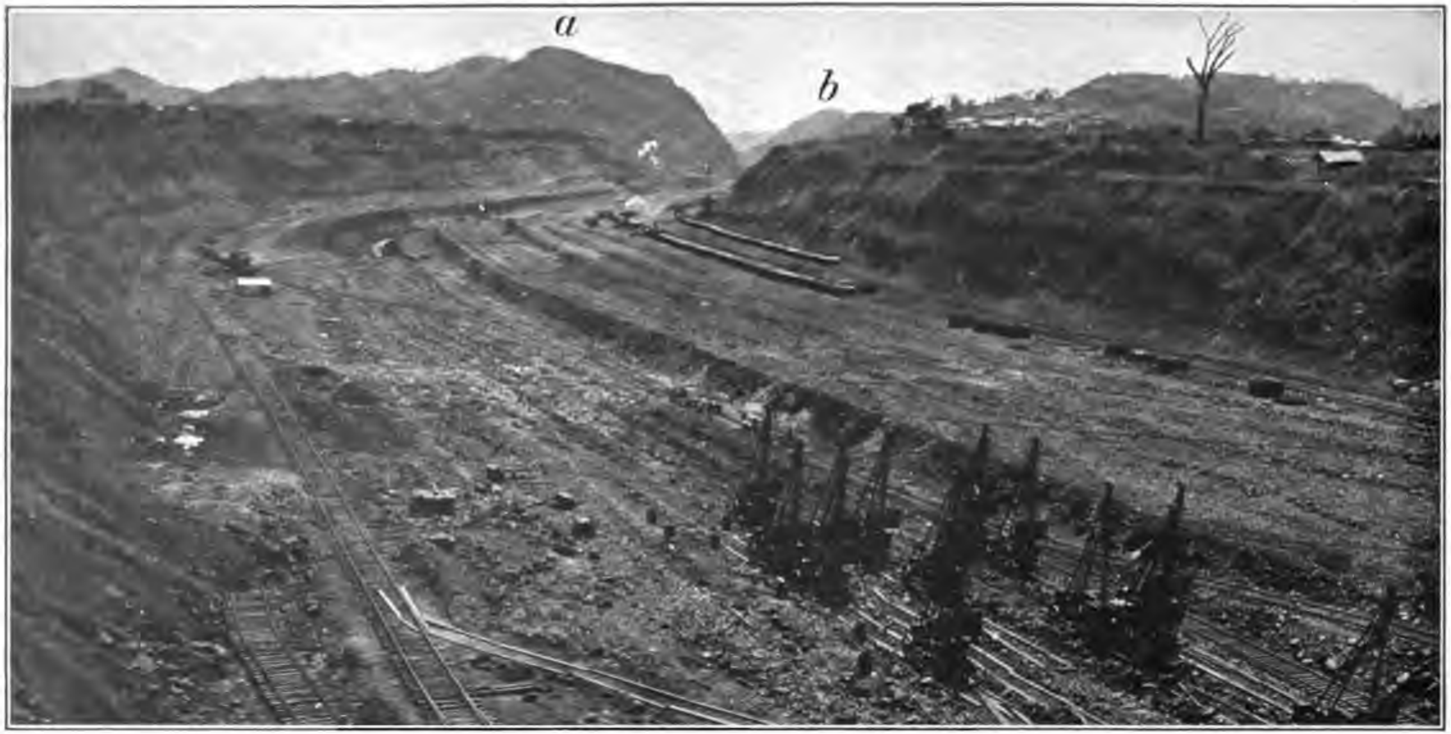
3. Where gravity is able to wedge off masses of rock from any steep slope, then any weight added to such rock masses will increase their tendency to slide. Where, however, the slope is flat enough for any rock mass to be at repose on a solid foundation, then weight (normal to the horizontal) added to it will decrease its tendency to slide. This is merely a special application of the principles illustrated by the parallelogram of forces. The weight of the building, therefore, will in this case not increase the tendency of this slope to slide because the slope is already well within the angle of repose.

4. The weight of the building will, of course, increase the tendency of this rock to deform and bulge up at the base, but, as already explained, it is strong enough to resist any compressional pressure that could possibly be brought to bear on it by any building.

In conclusion, the building is quite safe from the danger of sliding. Of course, this memorandum does not consider the question of the possibility of uneven settling of any part of the building, except to state that if the foundation piers have been put down to the relatively solid (though jointed) rock beneath the soil and loose-rock covering there will be no noticeable settling.

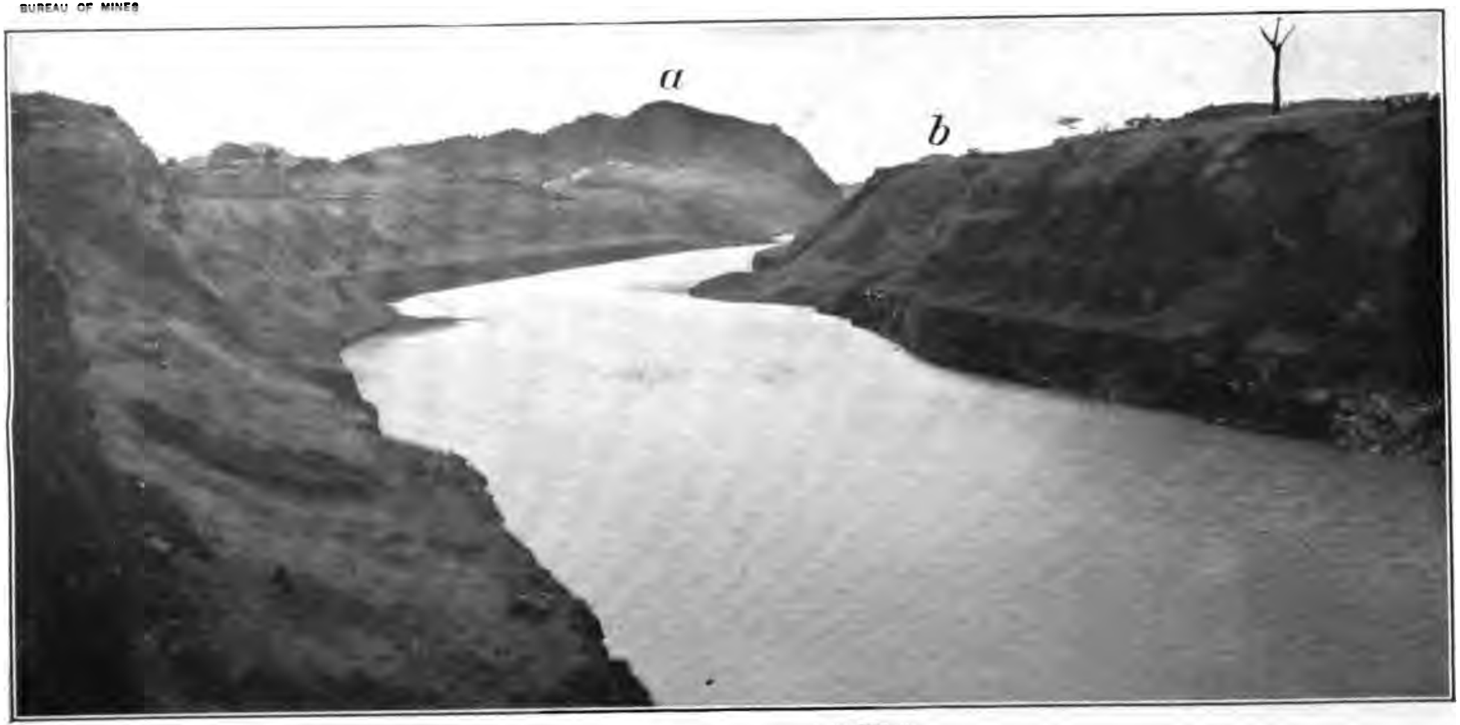
SLIDES.

Culebra Cut is a vast ditch (Pls. XII and XIII) that passes through many varieties of rock in the 9 miles of its length. Some of these are weak and unstable, and where the slopes of the cut were steep and 100 to 300 feet high the weaker rocks locally crushed down to flatter slopes. The flatness depended on the material involved, but, whatever the material, whenever the slope got flat enough the sliding stopped. At no time did the engineering staff constructing the canal believe that the slides were a menace to the ultimate completion and successful operation of the canal, in spite of the fact that at times



CULEBRA CUT IN THE MAKING. LOOKING SOUTH FROM EMPIRE BRIDGE, MARCH, 1911.

a, Gold Hill. *b*, Contractors Hill.



CULEBRA CUT COMPLETED AND WATER IN.

a, Gold Hill. *b*, Contractors Hill.

they were somewhat troublesome. They have made necessary the excavation of about 30,000,000 cubic yards more than was included in the first estimates for Culebra Cut, but they have not in the past and will not in the future endanger the ultimate success of the canal. Because of their importance as a factor in canal construction they are discussed in considerable detail.

ANALYSIS OF CONDITIONS LEADING TO SLIDES.

The following information, with accompanying illustrations, has largely been given by the author in a paper[a] read before the Twelfth International Congress at Toronto, Canada.

Any excavation in the earth's crust sets up stresses in the contiguous rocks, because of the unbalanced pressures created by the substitution of atmospheric pressure for the greater pressure of the material excavated. These stresses are divisible into two distinct groups, as follows: (*a*) Crushing or direct gravity stresses, which have a maximum effect near the toe of the steep slope of the excavation; and (*b*) tensional or flowage stresses, also due, though less directly, to gravity, which exert a horizontal strain toward the excavation and give maximum deformation near the surface (see fig. 6). Now, what might be called the resultant (*c*, fig. 6) of these two groups of stresses may for convenience be called the straining or slide-producing factor.

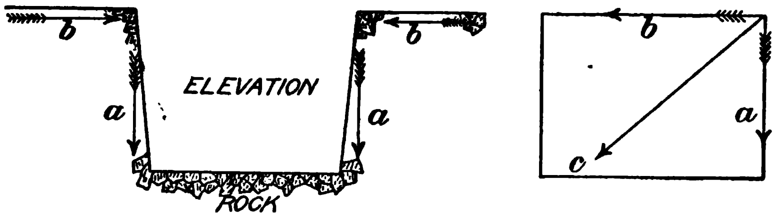


FIGURE 6.—Diagrams showing resultants of deformative pressures.

Two major classes of excavation set up strains in their slopes, or in the continguous rocks, which may result in slides. They are those due to processes of nature, as stream erosion, solution, and fault escarpments, and those due to works of man, chiefly open cuts, underground openings, and submarine excavations. The extent to which any excavation will cause sliding or deformation of the contiguous rock material depends on the following factors:

1. Crushing and tensile strength of large masses of the material involved. These factors vary according to (*a*) the strength of the small component masses, (*b*) the character of the jointing, (*c*) the character of the bedding, and (*d*) the fault conditions.
2. Physical and chemical character of the rock units.

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[a] MacDonald, D. F., Excavation deformations: Trans. 12th Int. Geol. Cong., 1913, Ottawa, Canada.

3. Amount and character of the ground water.
4. Earth tremors set up by earthquakes, blasts, passage of railroad trains, or other causes.
5. Other factors, as (*a*) heavy structures contiguous to excavations, (*b*) water freezing in rock openings and wedging off rock masses, (*c*) variation of barometric pressure, and (*d*) earth strains from the kneading of tidal pull.

INFLUENCE OF CRUSHING AND TENSILE STRENGTH OF THE ROCKS INVOLVED.

If a rock has high crushing strength, with few joint or other parting planes, the exposed face will stand almost perpendicular, without shearing or slipping, at any height with which we need to have concern. If it has high crushing strength in small solid fragments, but is much cut by joints, faults, or bedding planes it will not deform or flow but will slough off masses from steep places until a certain angle, which might be called the angle of repose, is attained. Fissures, faults, or bedding planes that incline toward the excavation and intersect its slopes are planes of weakness and promoters of slides, especially if the bedding planes have shale, lignite, or other greasy rock partings, or if the fault planes are filled with clay or slippery talcose material. Even if such partings are horizontal, relatively light back pressure may push the material that rests on them out into the excavation. Beds of dense, relatively impermeable, greasy clay which slope toward an excavation are especially likely to allow any material that may rest on them to slip off their moist inclined surface.

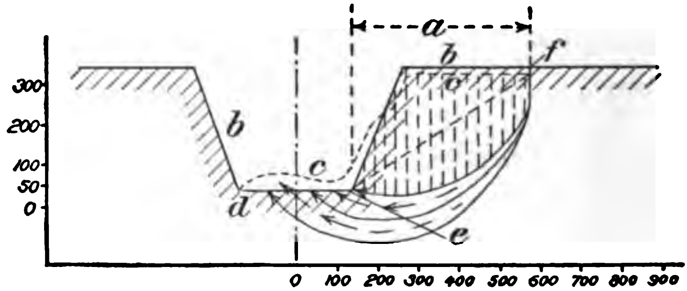


FIGURE 7.—Ideal cross section to illustrate excavation deformations. *a*, Width of zone of deformation, measured from toe of slope; *b*, original surface; *c*, surface after deformation; *d*, bottom of excavation before deformation caused it to bulge upward; *e*, toe of slope, intersection of slope and bottom planes; *f*, crack, or break, caused by deformation of basal part of the block shown with perpendicular broken lines. The curved lines from *f*, with arrow, show approximately the chief zone of deformation and the direction of the motion.

If a rock has low crushing strength, movement may manifest itself as a deformation or sinking near the excavation, with a slight advance of the lower slope toward the excavation, and a bulging of the bottom (see dotted line, fig. 7). Similar movements have occurred in the Culebra Cut. In most instances, however, such a movement is only one stage of disturbances that later end in slides, especially if the excavation toward which they are strained be deepened. If a plane such as *ef* (fig. 7) be imagined to extend from the toe of the slope, *e*, through the point of farthest deformation, this plane may not be the final slope at which the material will ultimately come to rest, for such a flat slope might never be attained by sliding. On the other hand, it is not the limiting plane below which deformation or movement of the rocks will not take place, for in certain of the Culebra Cut slides movement well below such a plane has disturbed and weakened the slopes (figs. 7 and 8 and Pl. XIV). The term "angle," as applied to the measure or extent of excavation deformations, is apt to be misleading, so the notation explained below is now much used by engineers.

If an excavation 200 feet deep causes a deformation in the contiguous rocks to a distance of 500 feet in a horizontal direction from a vertical line at the toe of the slope, *e*, shown in figure 7, the phenomenon may be designated as an excavation deformation of 200 on 500 feet. This designation indicates not only the depth of the excavation and the width of the zone of deformation, but also the angle of what may, in sand for instance, be the final slope or angle of repose. Engineers now often refer to slopes as being 1 on 1, 1 on 2, etc., meaning 1 unit up and 1 or 2 units over. The designation, however, is a measure only of the angle of slope unless the actual number of units of depth of excavation and width of deformation zone are given. If the rock has a low tensile and crushing strength, conditions would, in the absence of excessive mobility through ground water or other causes, approach those of a talus slope or an excavation in sand, where the slopes would be permanent when they had reached the "angle of repose" for the material involved.

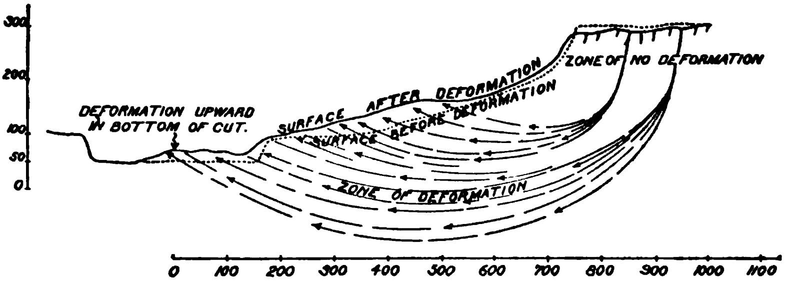


FIGURE 8.—Ideal cross section to illustrate canalward deformation movements.

INFLUENCE OF PHYSICAL AND CHEMICAL CHARACTER OF ROCK UNITS.

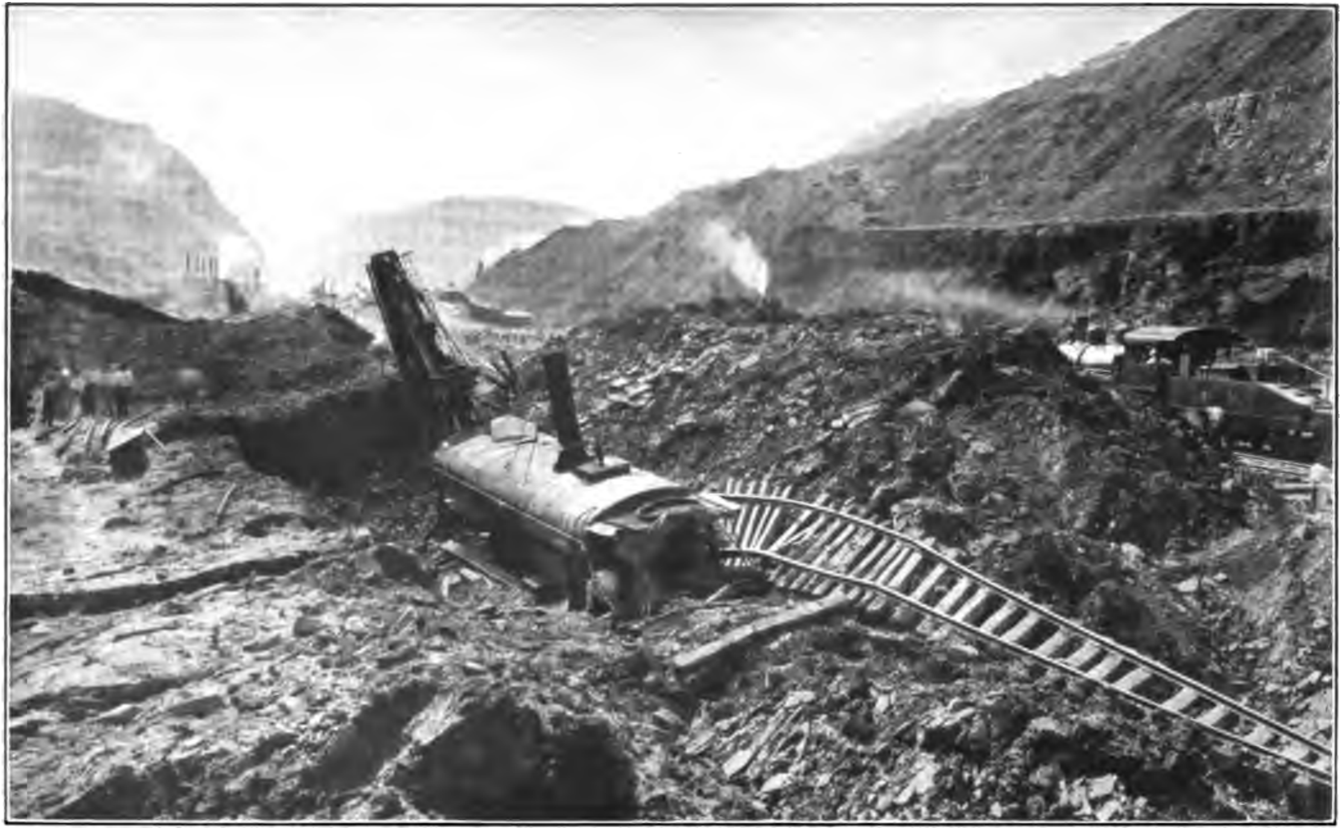
Excavation slopes in very soft rocks, especially very fine-grained and compact argillites and clays, may remain almost vertical until the excavation reaches a depth of 50 to 125 feet, or until the unbalanced pressure is great enough to cause them to deform. This deformation destroys the stability of their clayey cementing material, loosens them for the invasion of quantities of ground water, which their fineness of grain had excluded previous to deformation, and induces muddiness and mobility, so that the slope, originally almost perpendicular, may break back to 1 on 10. Deformations closely approaching this have occurred in Culebra Cut in some of the volcanic clay rocks.

Excavations that change the level of the water table may weaken the surrounding rocks by dissolving and loosening their more soluble parts, especially in a region of abundant vegetation, where the ground water contains much carbon dioxide and organic acids. On the Canal Zone the limestones contain many caves, small caverns, enlarged joints, and pitted surfaces, evidences of comparatively rapid solution. The volcanic clay rocks, particularly those of the Cucuracha formation, crumbled rapidly when they were dug from Culebra Cut and exposed to weathering action. Bowlders of this soft rock 2 or 3 feet in diameter have crumbled into clay in less than a year.

The crumbling seems to be due to two causes: First, the outer layers of these clay rocks may absorb rain water and heavy dews, and subsequently be dried out by the hot sun. It is known that after having been once dried out most clay rocks have their absorbing capacity increased much beyond what it was before drying took place. By this process of absorption and drying out, soluble salts are leached out and are sometimes deposited on the outside of the rock as barely noticeable white coatings or as scattered, minute, translucent crystals. These seem to be largely magnesium, iron, and calcium sulphates. This wetting, drying, and leaching loosens up the small individual grains and they drop off, thus removing layer after layer until the whole rock is disintegrated. Second, oxidation, which gives an increase of volume and increases the tendency to disintegrate, may occur. These rocks have a considerable percentage of ferrous iron (up to 3.7 per cent), which is somewhat oxidized to the ferric condition by contact with the atmosphere and with oxygen-bearing rain water.

EFFECT OF GROUND WATER.

Ground water in clay or in weak clayey rocks is an extremely important slide-producing factor. The chief ways in which it aids in the deformation of rock masses are as follows: (1) By greatly increasing the mobility and slippery character of the rock material; (2) by adding weight to a rock mass, which may already be strained toward an excavation; and (3) by weakening a rock through solution and softening. If a relatively porous material rests on top of relatively impervious rock, especially if the contact between the two slopes toward an excavation, ground water or rain will add weight



BULGING OF ROCKS IN BOTTOM OF CULEBRA CUT, CULEBRA, IN FRONT OF LARGE BREAK OR DEFORMATION SLIDE IN EAST BANK BETWEEN STATIONS 1746 TO 1758.

to the porous mass by filling its interstices. It will also soften and weaken the porous material and greatly decrease its adhesive grip on the sloping surface of the relatively impervious rock. The water will descend through the porous material, but will be deflected by the relatively impervious rock; and on this impervious surface will be deposited the mud particles and other lubricating material gathered by it in descent. The contact thus becomes a slippery zone that greatly promotes slides. Even capillary water in a weak rock is a source of danger, especially in fine-grained rocks, for with deformation much of the capillary water may be forced into shear planes, thus giving them increased lubrication. In estimating the sliding or deforming tendencies of a rock careful determinations of its water content should be made, both fresh and air-dried samples being used. The most troublesome slides of Culebra Cut were in fine-grained basic volcanic shales and argillites of fairly massive character. These, after having been air-dried, show 6 to 17 per cent of water and contain considerable finely divided chlorite, a hydrous mineral. Two samples from the Cucuracha formation—the formation in which maximum sliding has developed—were analyzed in the United States Geological Survey laboratory. They showed 12.26 and 9.47 per cent of water when heated below 100° C., and 5.41 and 6.71 per cent when heated above 100° C. Any considerable percentage of chlorite particles is an important weakening factor in rocks, and one that renders them slippery and unstable.

EFFECT OF EARTH TREMORS.

Earth tremors cause deforming movements in rock masses. Many earthquakes cause landslides and rock deformations. Heavy blasts also generate earth waves that loosen and weaken excavation walls and greatly increase their tendency to slide. A blast generates two sets of vibrations, one transmitted through the earth, the other through the atmosphere. The atmospheric vibrations travel seemingly with the same velocity as sound, the earth vibrations much faster. Surface blasts expend a greater proportion of their energy in atmospheric vibrations than do deep blasts. Hence, surface blasts make much more noise and jar houses more than deep blasts, but they have little influence in causing slides. On the other hand, the tremors created by a deep blast help to bring down, even at a considerable distance from the explosion, rock masses already dangerously strained. Two large blasts in Culebra Cut gave the following approximate vibration records: A blast of 2,250 pounds of dynamite, exploded in 14 holes 24 to 28 feet deep, gave a maximum amplitude of 20 millimeters. Another blast of 5,370 pounds of dynamite, exploded in 48 24-foot holes at about the same distance from the instrument, gave an amplitude of 28 millimeters. These records were made on an imperfect instrument improvised by C. M. Saville, one of the engineers. Its magnification was about 10, so that the amplitudes of the earth waves set up by the blasts were about 2 and 2.8 millimeters, respectively, quite enough to damage seriously a steep slope of brittle rocks already heavily strained. Vibrations from railway trains may also damage slopes that are already in danger of sliding.

EFFECT OF OTHER FACTORS.

Heavy structures contiguous to excavations greatly increase the tendency to slide or cave, as subway and foundation engineers well know. The wedging off of rock masses by the freezing of water in joints, fissures, and interstices is familiar to all geologists who have studied talus slopes in temperate climates. Earth strains, originating from the variation of barometric pressure and the kneading of tidal pull, should not be ignored in the study of excavation deformations. One is apt to forget that the maximum variation of atmospheric pressure near sea level may be more than 4,000,000 tons per square mile. So, if the adjustment of atmospheric pressure in a very large cave or mine lags behind any quick increase in atmospheric pressure at the surface, a considerable strain may be added to the roof of such an excavation.

TYPES OF SLIDES OR DEFORMATIONS IN CULEBRA CUT.

In Culebra Cut there were four distinct types of slides, or excavation deformations, as follows: (1) Structural breaks and deformations; (2) normal or gravity slides; (3) fault-zone slides; and (4) surface erosion.

STRUCTURAL BREAKS AND DEFORMATIONS.

GENERAL DESCRIPTION.

The largest and most important slides developed from structural breaks and deformations. Fortunately, they occurred only near Culebra in a section of the cut not much over a mile long. These deformations first manifested themselves by the appearance of one or of a set of cracks or fissures (fig. 4) parallel or somewhat oblique to the edge of the cut, and from a few yards to some hundreds of yards back from it and from each other. Some of them were traceable on the surface for several hundred yards and gradually developed into perpendicular crevices up to one-third of a yard wide and many yards deep. The second stage of this phenomenon consisted in the settling or tilting of these big block masses that had been divided from each other by the fissures. This movement was a slight and almost even settling (0.1 to 1 yard) of the block or blocks, or a tilting of them toward the excavation, or both. Some of the blocks sank a little in front and tilted up in the rear so that they were a yard above the front part of the block behind. The third and last stage consisted in the dropping downward of the front block, owing to the failure and squashing out of its base. The whole block then disintegrated and sloughed down into the excavation. This last stage ran its course in a few hours to a few days; the other stages required some months or more than a year to reach completion. The movements of the last and of the second stages were often accompanied by bulging of the bottom and of the lower slopes of the excavation (see figs. 7 and 8 and Pl. XIV).

CAUSES.

This type of deformation was due to a primal cause, the unstable geologic condition of the rocks, and an immediate cause, the oversteepness and height of the slopes, the blasting, and other work attributable only to man. The unstable condition of the rocks resulted from several geologic factors, chief among which were: (1) The formations involved consist of soft, weak rocks, comprising massive, partly indurated volcanic clays, friable, bedded tuffs, and soft, brittle, and slippery lignitic shales, all with a relatively high ground water content. These rocks have low cohesion and crushing strength. They are so weak and unstable that locally the unbalanced pressure at the foot of the high slopes was often greater than the resisting strength of the rock. This condition induced deformation, with accompanying slow flowagelike movement, and at times a bulging in the bottom of the cut in front of the slide (Pl. XIV). (2) The rocks have been further weakened by faulting and by joint and bedding planes. Shearing zones, due to faulting, trend across the canal, as shown in Plates XV,[a] XVI, XXV, and figure 1. The grinding movement of the rock masses against each other along the faults has been sufficient to shear and crush them for several feet on both sides of the planes of movement. These crushed-rock zones greatly weakened the slopes. (3) Deforming movements caused much rupturing of the weak rocks so that from a dense, fine-grained, relatively impervious condition they locally became much sheared and subject to invasion by large quantities of ground water which tended to leach, oxidize, and disintegrate them. (4) Lignitic-shale beds, especially those dipping toward the excavation, were planes of weakness along which there was a strong tendency for the overlying material to slip. (5) The presence, too, of a considerable proportion of chlorite particles in the volcanic clay rocks tended to lubricate the mass.

The second or immediate cause of this type of slide was chiefly the oversteepness of slopes where the banks were high and the rocks weak and more or less saturated with ground water. In an excavation through granite the slopes or walls might be made perpendicular for depths of several thousand feet without their crushing in, but if such excavation were carried to a depth of several miles, perpendicular walls could not be maintained even in granite, for the unbalanced pressure at the foot of such walls would exceed the crushing strength of the rock and collapse of the lower part of such excavation would result. The critical depth and steepness for the rocks involved have locally been exceeded in Culebra Cut. The earth vibrations set up by deep, heavy blasting, near slopes already under a severe strain and carrying much ground water, have had a strong tendency to develop slides. The geologic conditions were not sufficiently considered in the first plans for digging Culebra Cut.

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[a] Formerly published by the author in an article on "Slides in the Culebra Cut at Panama," Eng. Record, vol. 66, August, 1912, pp. 228-233.

REMEDIES.

For this type of slide there was only one remedy that had utilitarian value under the conditions involved, and that was applied. It consisted in making the slopes less steep by removing material from their upper parts until the unbalanced pressure at the foot of the slope became less than the crushing or deforming strength of the rock. To do this, steam shovels were put on the banks to terrace them back on either side of the cut and relieve the strained condition (see Pls. XVII and XVIII). At first sight it might seem preferable to have let the slides come into the excavation and to have shoveled them out from the bottom until permanent slopes were reached, thus saving the expense of much blasting. But deformations of this kind weakened the rocks far below the bottom of the excavation (see figs. 7 and 8), and this weakened material would stand only at a much flatter slope than was necessary before it became loosened and disintegrated by deformative movements. Further, as each block or mass crushed down it generally left behind not a gradual slope, but a steep face 12 to 25 yards or more high, which greatly assisted in the generation of other slides. The additional expense and difficulty of shifting and adjusting the railroad tracks, the air pipe lines, the drainage, etc., were also involved. Time figured as a causal factor in these large slides and deformations, for some of them ran their course in a few weeks or months, but others showed cracks for many months or even for years before sliding. Such fissures were sources of weakness which sooner or later, with the deepening of the cut, became troublesome, unless remedied by the reduction of the slopes.

SLIDES IN GOLD HILL AND CONTRACTORS HILL NOT LIKELY.

The highest and steepest parts of Culebra Cut, namely, Gold Hill and Contractors Hill, did not deform and crush, because they are composed of relatively strong rocks of volcanic origin. These more solid rocks extend down deep into the earth and thus constitute vast

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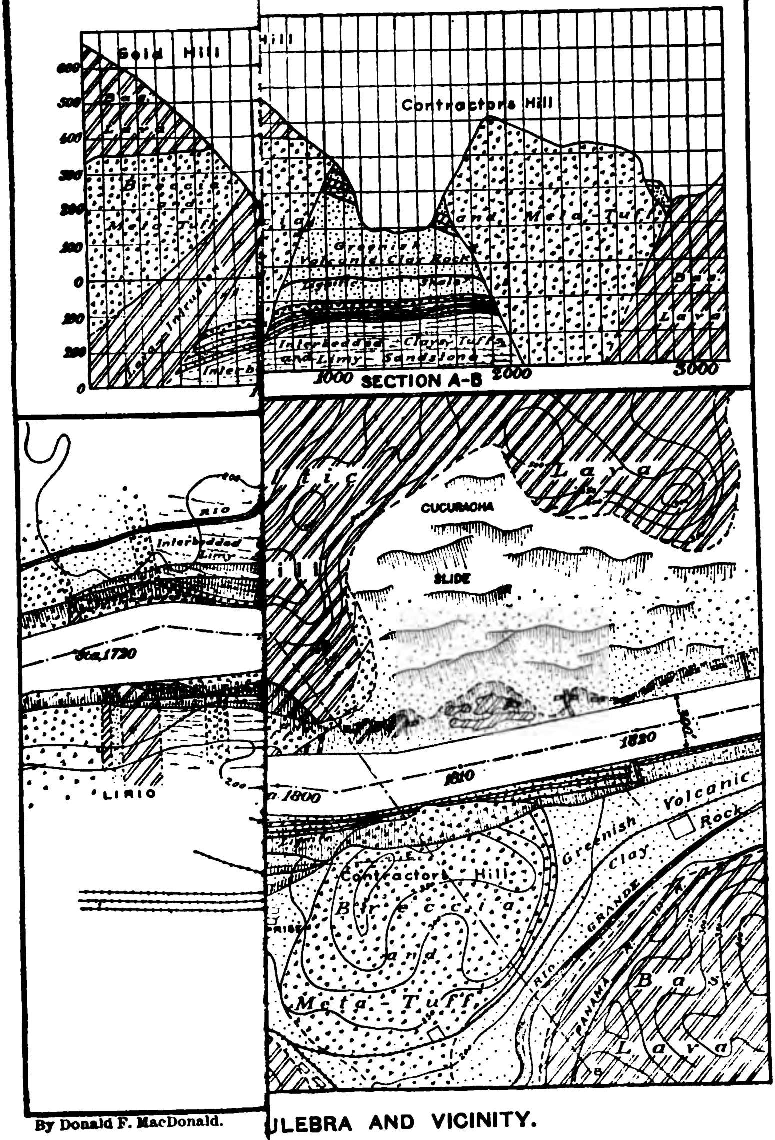


By Donald F. MacDonald.

ULEBRA AND VICINITY.

unyielding buttresses against slides in a section that would otherwise be most susceptible to them. Of course rock loosened by jointing, due in part to weathering and in part to former heavy blasting, from time to time sloughed off from these hard rock masses, but on the whole they have stood solid and immovable.

YARDAGE OF THE CHIEF SLIDES CAUSED BY STRUCTURAL BREAKS.

The chief slides due to structural breaks have taken place in that part of Culebra Cut lying between Gold Hill and Empire Bridge. On the west side in this section of the cut sliding began in October, 1907, and extended so that in all over 70 acres of material moved or was seriously cracked. This movement necessitated the removal of over 11,000,000 cubic yards more than was comprised in the first estimates. This amount, of course, included the material removed from the upper part of the slopes in order to make them less steep and more stable. Sliding began on the east side in this section of the cut about January, 1907. In all some 55 acres of land surface has been in motion here since then, adding close to 8,000,000 cubic yards to the first estimates. The relieving process on this part of Culebra Cut is at this date (January, 1914) not yet quite complete, and a considerable amount still remains to be moved before perfect stability of slopes will prevail.

The structural-break type of slide is therefore responsible for the movement of about 19,000,000 cubic yards of material over and above that included in the first estimates.

PREVIOUS STATEMENT REGARDING STABILITY OF HIGHEST PARTS OF CULEBRA CUT.

Before the subject of the structural-break type of slide is left, it might be well to set forth the salient features of a memorandum regarding the stability of the places of maximum height of slope in the Culebra Cut, especially at Gold Hill and Contractors Hill. The memorandum was prepared by the author in the fall of 1911 in response to a more or less current opinion to the effect that these hills would, because of their great height and steepness, deform their bases and crush down into Culebra Cut as great structural-break slides. An examination of the geologic conditions showed that such would not be the case, as mentioned in the excerpts that follow.

> Three sections are shown on Plate XV, namely: A–B, Gold Hill, through Contractors Hill, to the lava slopes beyond the Panama Railroad; A–C, Gold Hill to Mount Zion Reservoir; and D–E, from a point northwest of Col. Hodges's house to Contractors Hill. These sections are drawn to depths of about 200 feet below sea level. The rocks shown at that depth are known to occur there from the fact that they outcrop at certain angles on both sides of Gold and Contractors Hills, and a projection of both angles and of the probable curvature of the beds locates them at about the depth indicated. That stronger limy sandstone rocks occur below the green clays in the vicinity of Contractors Hill is certain; that they occur at the exact depth shown on the cross section is not certain, for they may be, say, 25 feet higher or lower than the

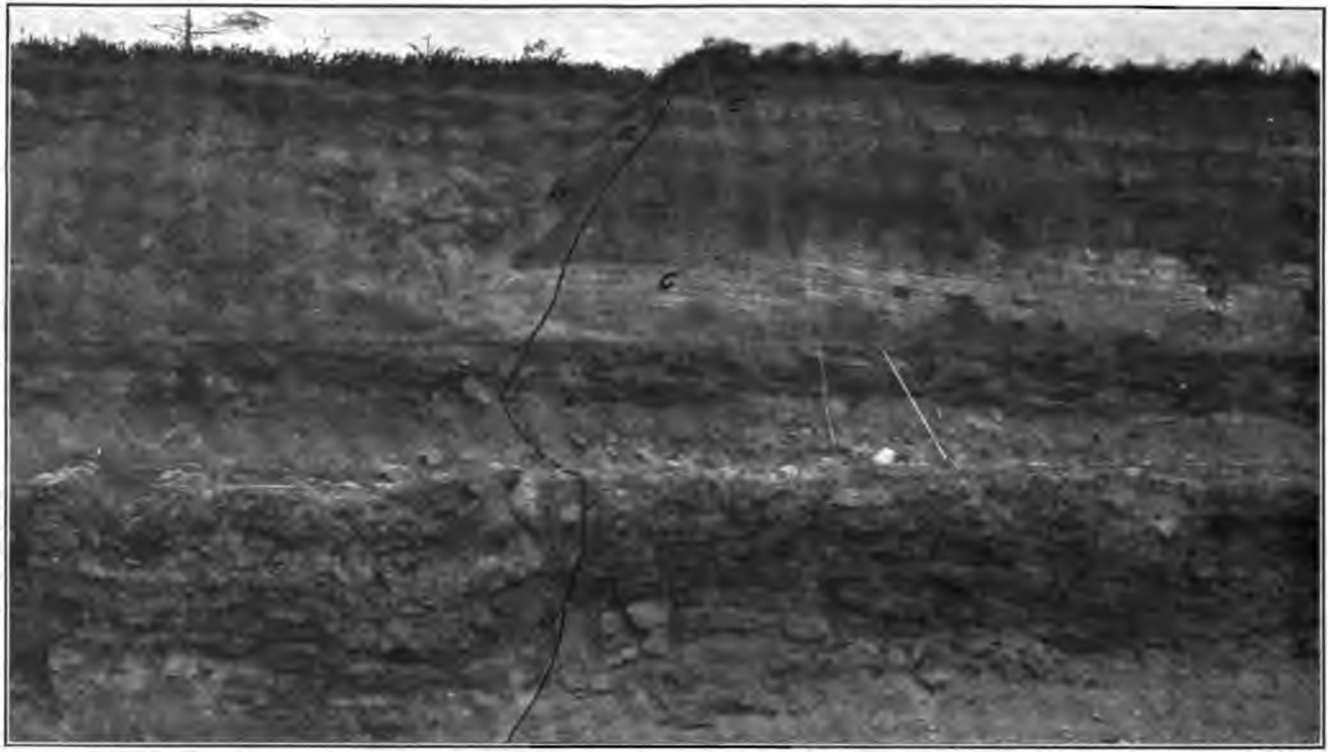
elevation there recorded. The presence of these rocks has a direct relation to the slide problem, for they are strong enough, except where faulted, to limit the downward depth of any deformative movement of the weak green clay rocks on top of them. Unfortunately, southward of station 1762 they are too deep to prevent present deformative movements of the green clay rocks above them. Northward of that station they come to the surface for a considerable distance, and their presence is marked by sustained slopes between stations 1762 and 1749.

Contractors Hill.—Contractors Hill is a cap on top of a basalt plug which came up from a depth of probably much more than a thousand feet. It contains fragments of basalt, and consolidated at an elevation above the stability of its foundation. Gravity, cooling, and time were factors that caused a settling back to equilibrium, as it has done. What is the evidence of this settling back? Plate XIX shows a part of a shear plane that extends around the canalward side of Contractors Hill, and must entirely surround it. On this plane is a hard fragment *c*; inclosed in a softer matrix and behind it (above) is a sloping little ridge of softer material. This hard fragment, held fast in a softer matrix, had a part of this matrix scrubbed away from all sides of it except that on the upper side, the side opposite to the direction of its motion. This and other similar data absolutely establish the fact that the last movement of this mass was downward. The amount of this downward motion is indicated as follows: The zone of crushed rock *d* is about 10 feet wide. This wide zone of rock has been crushed and highly sheared by the downward motion, and the experience of geologic observation of many different kinds of faults is that shear zones 10 feet wide, under conditions similar to these, result only where the differential movement of the rocks has been, say, more than 300 feet. Now, it is evident that if this great roughly circular mass of rock, after having come up, settled back through the weaker surrounding rocks a distance of 300 feet, its foundation must be at least a thousand feet below the surface. Therefore it seems certain that the unbalanced pressure caused by the excavation of a few hundred feet of the surface material will never result in any disturbance to such a deep foundation as that of Contractors Hill. Of course local masses of rock will crush down from its steep slopes occasionally, and some of these may be rather extensive, but the hill mass as a whole will remain perfectly solid.

Gold Hill.—Gold Hill, in its origin and composition, is somewhat similar to Contractors Hill. The top of it, however, is covered with a thick flow of hard basalt, and a massive basalt dike has come up around its periphery on at least two sides (see Pl. XV). The downward motion of Gold Hill is evidenced by the fact that the small lava breccia mass *d'*, Plate XX, has come down from, and is a fragment of, the lava breccia *d*. With other sheared material, it has been dragged in by the downward frictional pull of the hill mass. The sheared zone *c* around the visible portion of the margin of the hill is as great as that around the margin of Contractors Hill, showing that the downward movement of both masses has been almost equally great, and that their foundations are undoubtedly several hundred feet below sea level. Gold Hill is also greatly strengthened by the massive lava dike that comes up on at least two sides of it, as shown in figure 1. The origin of this lava rock (basalt) has certainly been more than a thousand feet below sea level, and its strengthening influence is very great.

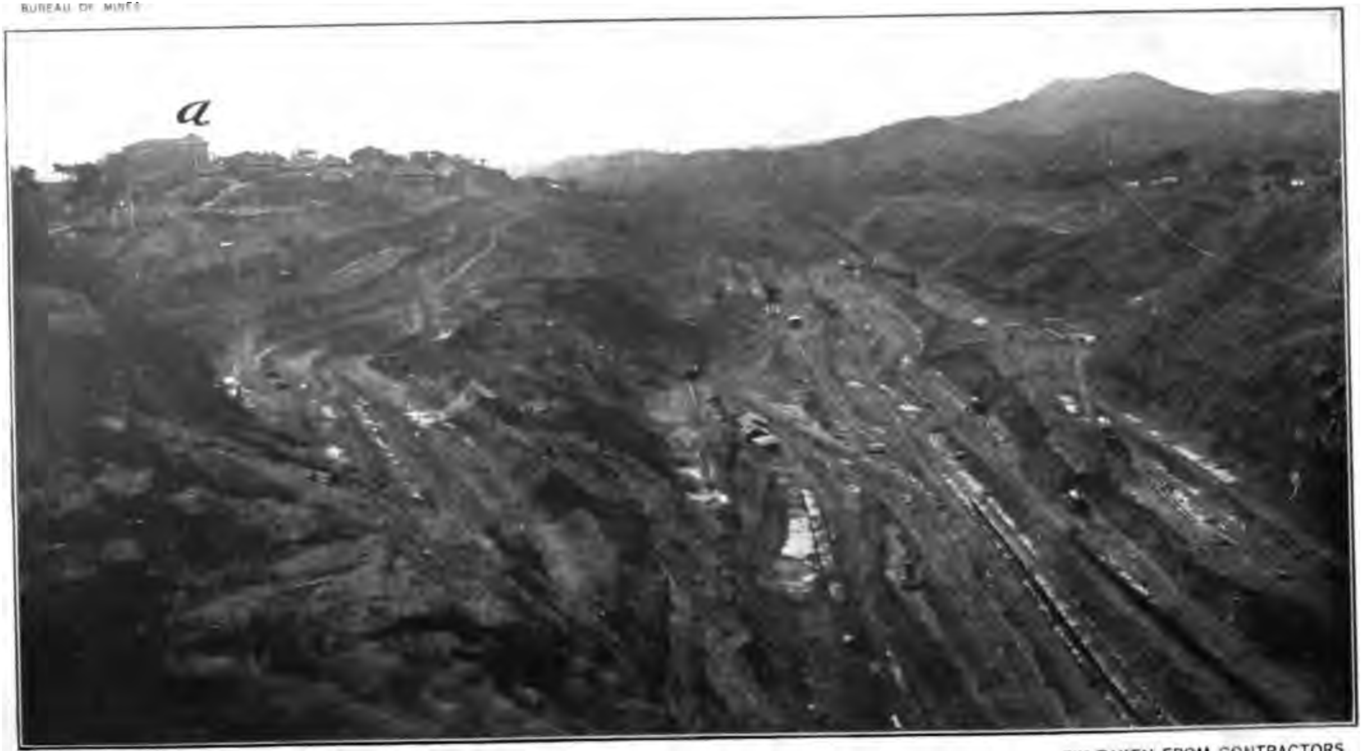
Faults.—The extent, direction, and dip of the chief faults in the vicinity of Culebra are shown on Plate XV. Most of these diverge outward from the contact zone between the two solid masses, Contractors Hill and Zion Hill. These fault planes have greatly weakened the already weak, friable, greenish clay rock, and for this reason the "storm center" of "breaks" and slides for Culebra Cut will continue to be here until permanent slopes are finally established.

Probable limits of sliding ground.—Lines representing the probable limits of sliding ground are shown on Plate XV. Such lines must necessarily be more or less approximate. In some places slides may reach beyond these approximate limits, and in other



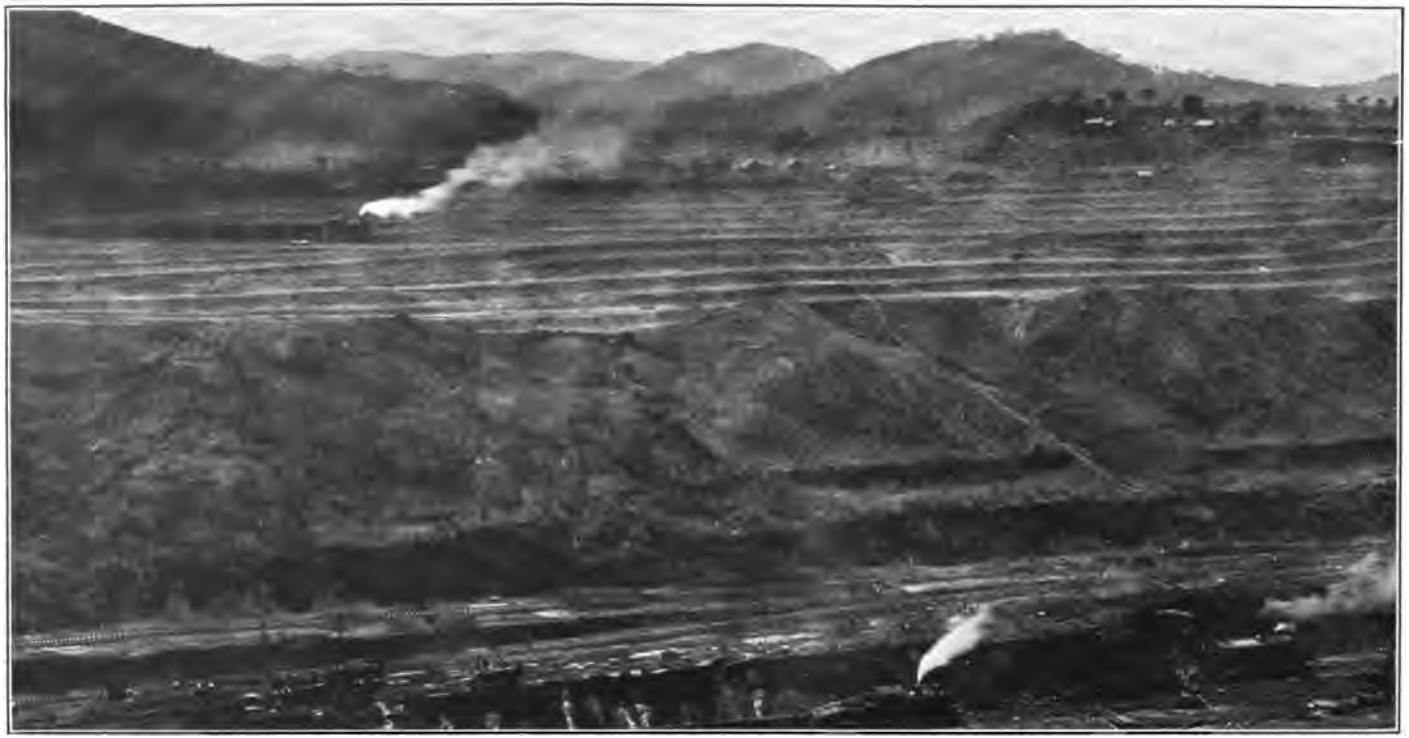
FAULT ZONE NEAR CULEBRA.

a, Fault plane cutting across canal. *b*, Zone of weakened sheared rocks. *c*, Limy sandstone beds which are stronger than the dark carbonaceous clay beds under them (see Plate XV, 1*b*). *d*, Dark carbonaceous clays and bedded tuff (Plate XV, 1*b*). Note how rocks on left of (*b*)—the overhang of the fault plane—have slid into the Cut.

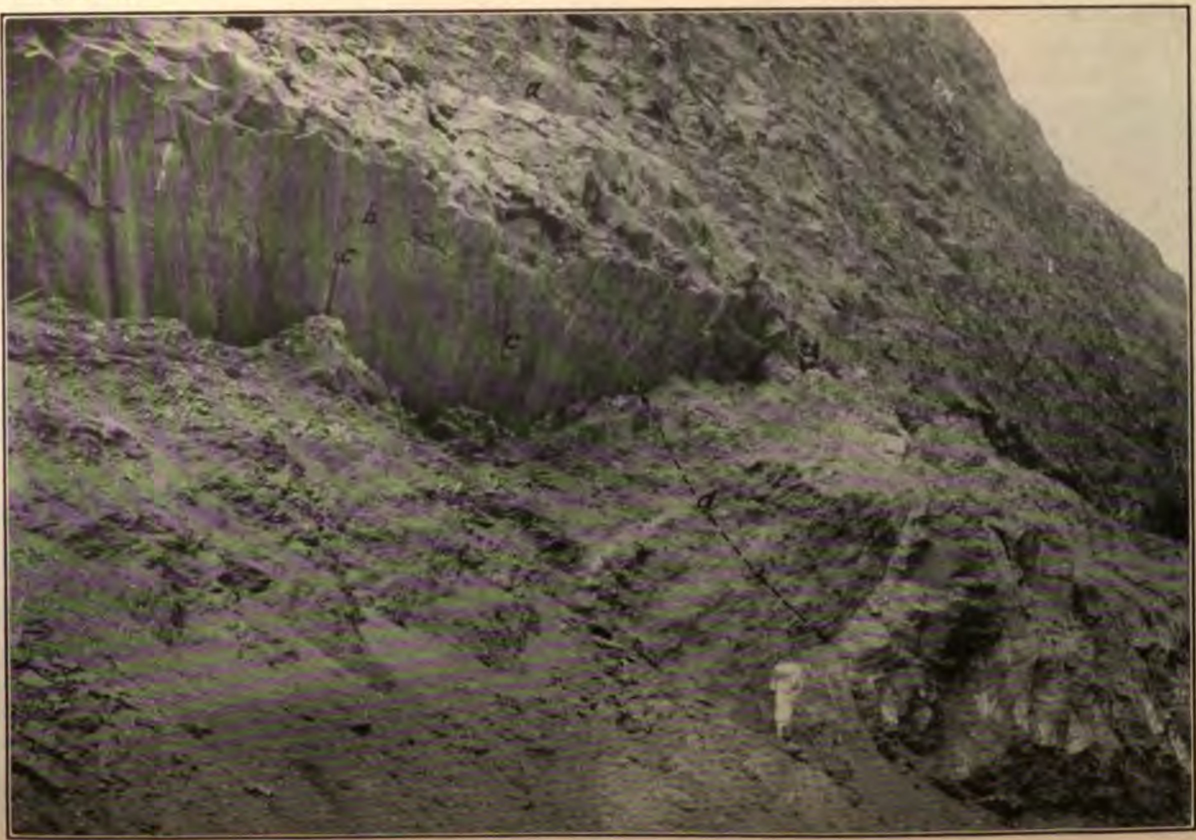


TERRACING ON BOTH SIDES OF CULEBRA CUT, NEAR CULEBRA, TO RELIEVE STRAINED CONDITION OF THE SLOPES. VIEW TAKEN FROM CONTRACTORS HILL, LOOKING NORTH.

a, General offices, Culebra.



TERRACING TO RELIEVE STRAINED CONDITIONS OF SLOPES, EAST SIDE OF CULEBRA CUT, STATIONS 1735 TO 1765.



FAULT PLANE AT CONTRACTORS HILL.

a, Contractors Hill mass, which seems to be surrounded by the fault plane *b*. The motion of the hill mass was downward, as shown by the protected little areas of softer matrix behind (above) the hard basaltic fragments in the hill mass, as shown at *c*. The sheared zone *d* is here over 10 feet wide.

places these limits may not be reached. On the whole, however, it is believed that these lines represent pretty closely the area of ground that is likely to be disturbed. This area may be narrowed down somewhat by the relieving excavation work which may be completed, say, before the close of the next dry season. It will be noted that these lines converge on the cut about station 1760, and are comparatively restricted for the next 1,000 feet northward. This is due to the strengthening effects of the sandy limestone and limy sandstone beds that occur toward the surface there. The western slope in the vicinity of 1740 to 1745 will probably be disturbed to a rather wide extent, but the sliding there will be somewhat superficial, and large movements of the break type should not occur. These superficial slides are due largely to the presence of the lens of greenish, sandy clay rock shown on Plate XV. The upper part of this becomes muddy, and any material that may overlie it easily slips into the cut.

On both sides of the canal in the vicinity of stations 1737 and 1747 faults have disturbed the strata. These faults, however, are mostly at right angles to the cut, and will therefore be much smaller factors in producing slides than would be the case if they cut across with diagonal trend. The slides, then, along near the above-named stations give no evidence of assuming very large proportions. Northward of these stations in the region mapped the sliding problem grows less until the Empire breccia is reached. This breccia as a whole will not slide, though local masses of rock may fall from its steep slopes. In conclusion, it may be said that the crack that developed under and disturbed the Club House, Culebra, was due to renewed movement along an old minor fault plane. All the ground under the Club House may slide, but the northwest corner of the building is not more than, say, 100 feet from ground that will not slide.

Summary.—In summary, maximum sliding in Culebra Cut will occur and is occurring on the west slope about opposite the Isthmian Canal Commission Chapel, because the already weak rocks there have been greatly weakened by faults which trend diagonally across the cut. Relief excavation work vigorously pushed ahead in this vicinity will greatly ameliorate conditions.

PROBABLE LIMITS OF SLIDING GROUND.

In spite of much subsequent deepening of Culebra Cut with some flattening of its slopes, the "probable limits of sliding ground" shown on Plate XV (made in 1911) hold good, with some exceptions, at the date of writing, August, 1915. In a memorandum to the Chairman and Chief Engineer of the Isthmian Canal Commission, dated May 10, 1913, the writer predicted a serious slide just north of Gold Hill, the outer limits of which would be at some cracks 1,500 feet back from the center of the canal. This slide will be far outside the limiting line mentioned above. A somewhat unlooked-for slide on the east side of the cut, between stations 1746 and 1758 in February, 1913, was caused by two large fault zones which were masked by sloughing material so that their dangerous character was hidden until the slide came down.

SUMMARY.

In summary, then, the structural-break type of slide was due to oversteep slopes in places where the banks were high and the rocks weak. The remedy was to unload the unbalanced pressure and to reduce the slopes. The ends sought to be accomplished by the remedy were a reduction of the amount of excavation that otherwise would be necessary and noninterference with tracks, drainage, etc.

The slopes stood at a much steeper angle before being disturbed than after they had been weakened by deformation.

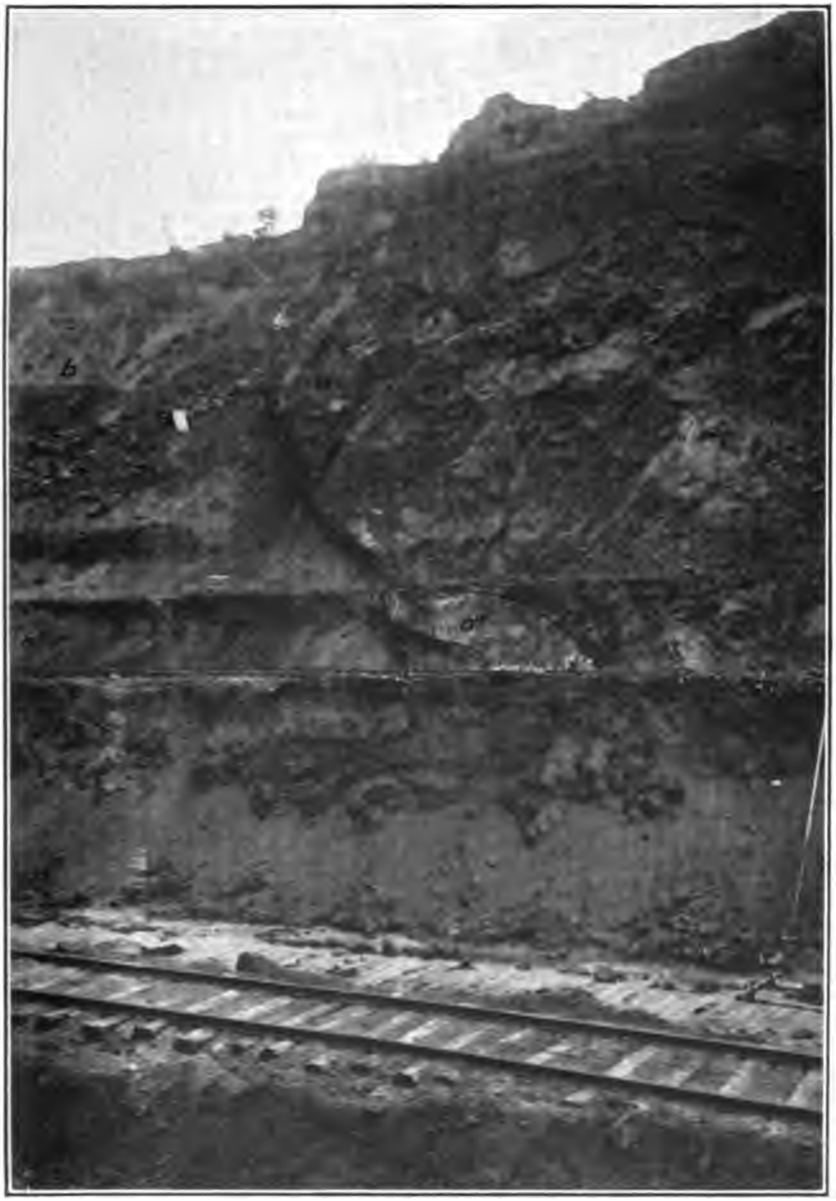
NORMAL OR GRAVITY SLIDES.

The normal or gravity type of slide was due to several factors. Locally along Culebra Cut porous material lies on top of relatively impervious clay, shale, or igneous rock. Rain and ground water saturated this porous mass, but were impeded in their downward course by the relatively impervious rock. Thus a muddy, slippery zone was formed along the plane of contact between the pervious upper and impervious lower materials. When this plane sloped toward the excavation, or where there was thrust or head of pressure toward the excavation from higher ground in the rear, a slide of the normal or gravity type often resulted (Pl. XXI and fig. 3). Where bedding and joint planes dip toward the excavation and intersect the slopes they greatly assisted gravity to wedge off rock masses. This type of slide had certain distinguishing features. The rocks were not deformed or weakened below the plane of actual sliding. The sliding material moved off a relatively solid base and this base was not pulled down or squeezed out by the frictional drag. Hence these slides were not as destructive as the break-deformation slides were, for they did not weaken the slopes below the plane of actual sliding. No saving of excavation could ordinarily be accomplished by removing material from the upper parts of such slides. It was better and cheaper to let them run their course and remove them from the bottom of the cut. Drainage was almost the only remedial factor that could be applied to them. The Cucuracha slide (Pls. XXIII and XXIV), active at intervals since the French company began operations, was the worst slide of this type.

Slides of the normal or gravity type have been relatively numerous, but, with the exception of Cucuracha slide, less important and much smaller than those of the structural-break type. The following list of these slides shows approximately how much material each involved:

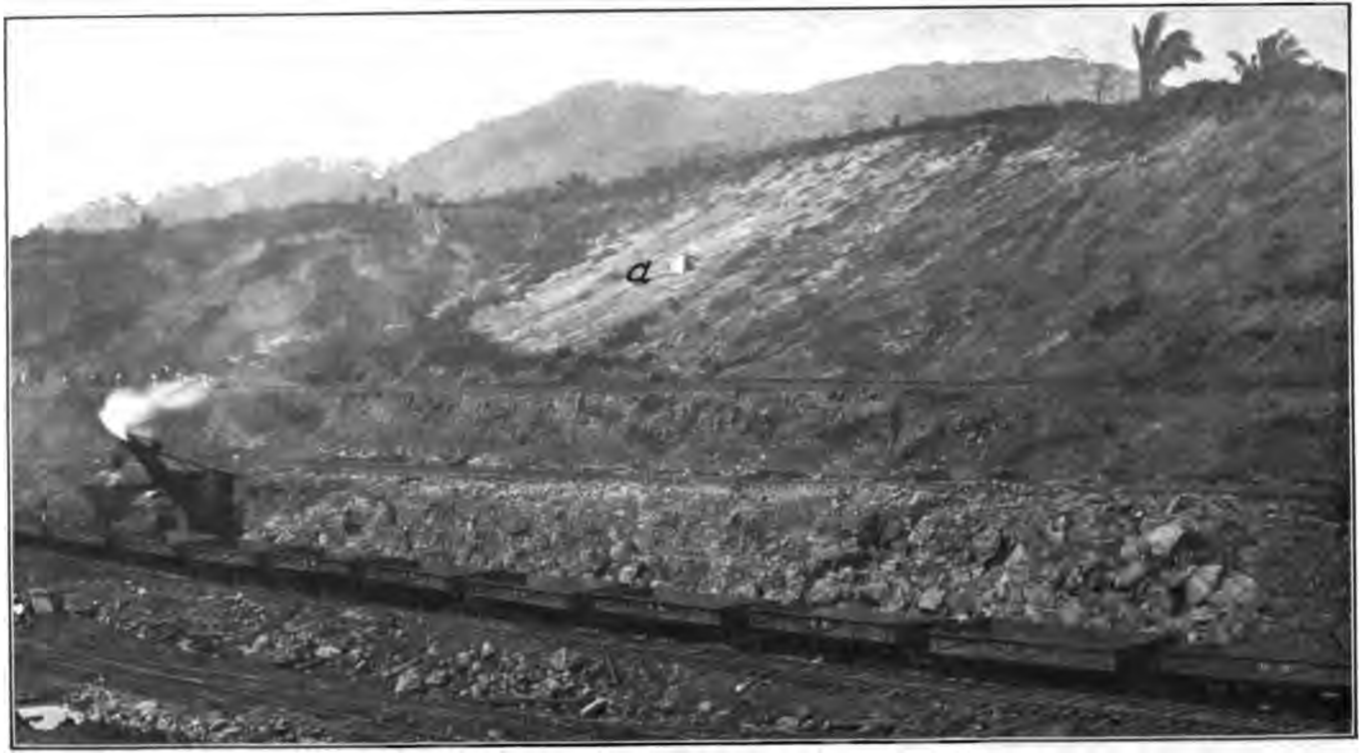
Extent and location of past slides in Culebra Cut.

Location.	Date when slide first developed.	Quantity of material excavated.	Area of slide.
East side:		*Cubic yards.*	*Acres.*
Buena Vista	May, 1912	68,000	1.2
Las Cascadas	February, 1908	583,000	11.5
Whitehouse	October, 1908	609,000	6.0
Powder house	October, 1909	613,000	5.8
Empire	May, 1912	1,213,700	20.0
Cucuracha	July, 1905	5,359,500	50.0
Cucuracha village	September, 1911	231,000	4.0
Paraiso	March, 1907	385,000	5.7
Pedro Miguel	January, 1913	13,800	.2
West side:			
Whitehouse yard	June, 1912	75,000	1.0
Cunette	September, 1910	67,000	.9
Total		9,218,000	106.3



FAULT PLANE AT GOLD HILL.

a, Northerly edge of Gold Hill mass, part of the basalt dike which forms the northern and western periphery of this hill; *b*, Cucuracha formation; *c*, fault zone, which extends around hill mass. *d*, old lava flow; *d′*, mass of old lava pulled down from *d* (over 100 feet) by frictional drag of hill mass, as it was faulted downward in late geologic times.



SLIDE NEARLY OPPOSITE LAS CASCADAS, ACTIVE IN 1908 AND 1909.

Relatively porous rock became saturated and slid off the comparatively impervious underlying rocks (a) toward the canal.

For nearly a year, in 1912, the Cucuracha slide showed little movement, owing to the fact that it was held back by large dikes of basalt (see Pl. XV and fig. 3). These seemed quite large enough to limit further sliding to a few hundred thousand cubic yards. However, when the cut in front of these sustaining dikes had been deepened to the final level the earth pressures behind them became so enormous as to shear and crush the dikes and start almost 2,000,000 cubic yards of material moving. The pressure developed by this slide and the flatness of the slope on which the material moved (see figs. 3 and 9 and Pl. XXII) were most astonishing. However, in spite of this the dikes would almost certainly have held but for the excessive jointing that had rendered them much weaker than their size indicated. Now that the cut is down to final depth in front of this slide the latter has entered its last stage of activity, and a year or two of dredging should entirely eliminate the menace of this slide.

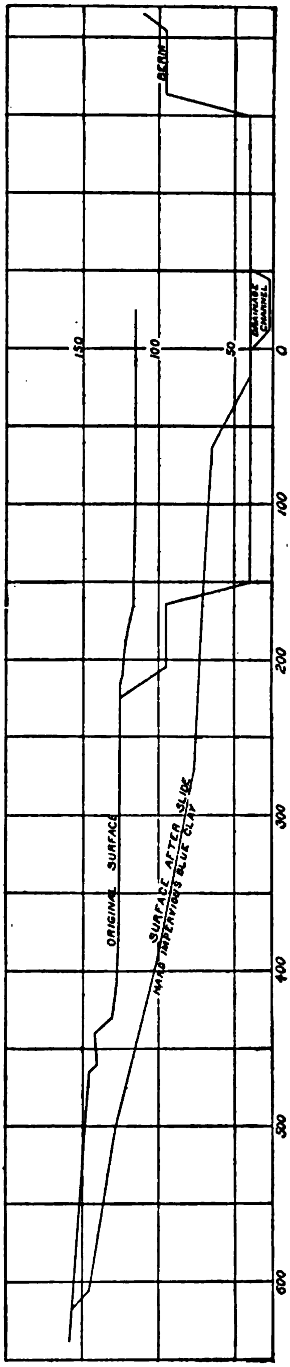


FIGURE 9.—Cross section of slide shown in Plate XXII.

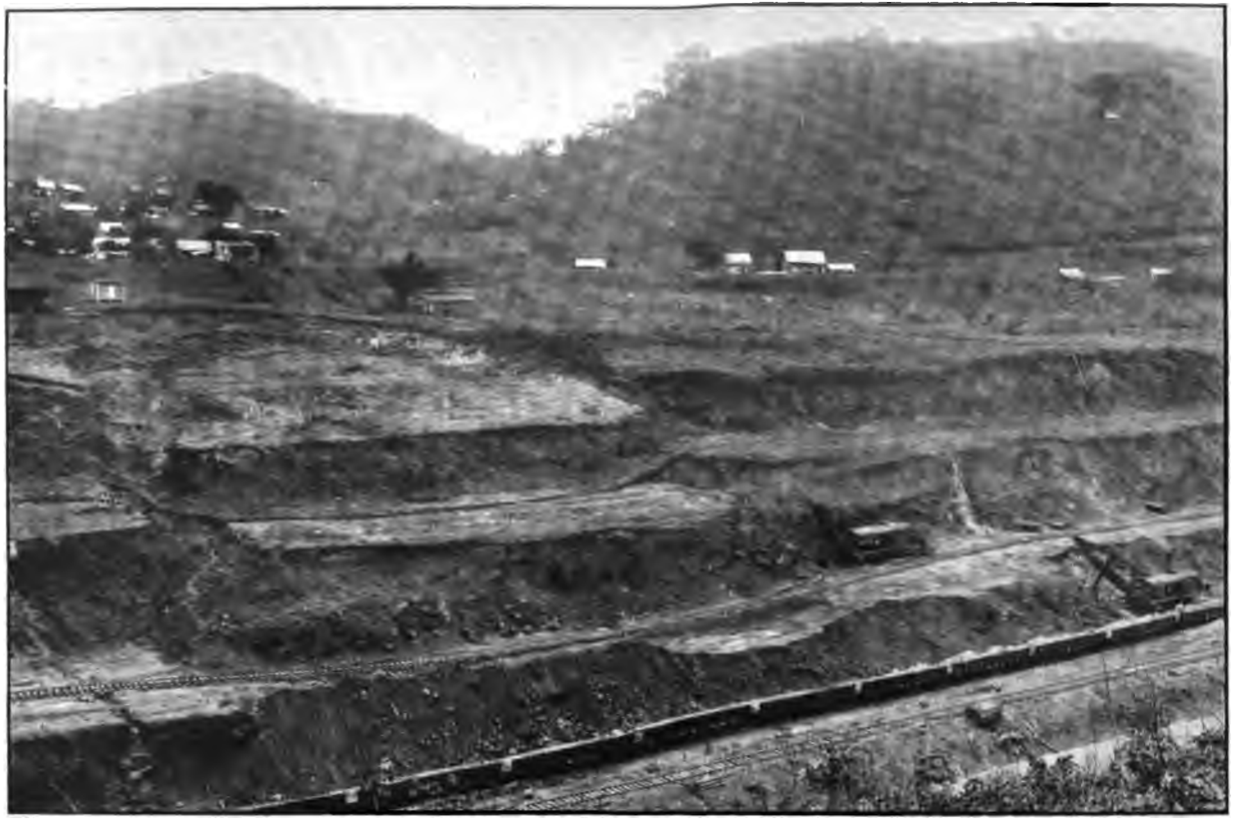
FAULT-ZONE SLIDES.

The third type of slide was that occasioned primarily by sheared and weakened zones in the rocks, due to fault displacements. As the rock masses moved past each other, in the adjustment of earth blocks in late geologic time, the frictional pressure or drag was often great enough to crush and shear the material for several feet on each side of the plane of movement (Pls. XV, XIX, XX, and XXV). Where faults of this kind cut diagonally across the canal large masses of rock in the acute angle between the fault plane and the plane of the canal slope generally fell out into the excavation. Where the fault plane had a considerable dip it left a mass of rock, with a narrow base and wide top, resting insecurely against the slippery crushed rock of the fault zone and quite separated by this crushed material from the solid rock on the other side of the break. This development, of course, threw an extra strain on the base of the faulted-off mass, and if that failed, the whole block crushed down.

The most notable slides of the fault-zone type were the La Pita slide (Pl. XXVII) and the large slide (Pl. XXVI) that occurred about 600 feet north of La Pita between Empire and Las Cascadas on the east side of the canal. In both cases a brownish breccia locally covers a dark-gray volcanic agglomerate to a depth of 60 feet, the whole forming a steep bank of 90 feet at La Pita and of over 125 feet in the case of the other slide. In each a fault (Pl. XXV) crosses the cut, making a diagonal angle with the trend of the canal. The differential movement of the rocks along this fault plane had sheared and crushed them. This plane of shearing dips northerly at an angle of about 65°, thus leaving a weakened overhanging mass which rested insecurely against the rocks from which it had been faulted off. A diversion ditch on top of the bank came within 100 feet of the slope here, and the water from this ditch leaked out in places along the breccia-agglomerate contact, but chiefly along the sheared fault zone. In the case of the northmost slide the basal agglomerate contained volcanic mud-lava flows which on cooling assumed columnar jointing (Pl. XXVIII), and these joints afforded channels for the invasion of seepage and ground water. The water softened the agglomerate, especially down near the foot of the slope, where it was already weakened by the fault zone and under great strain from the overhanging mass. The result was that the basal agglomerate failed and crushed down under the weight of a huge block of the breccia that contained about 20,200 cubic yards in the case of the La Pita slide, and 300,000 cubic yards in the case of the slide 600 feet north of La Pita.

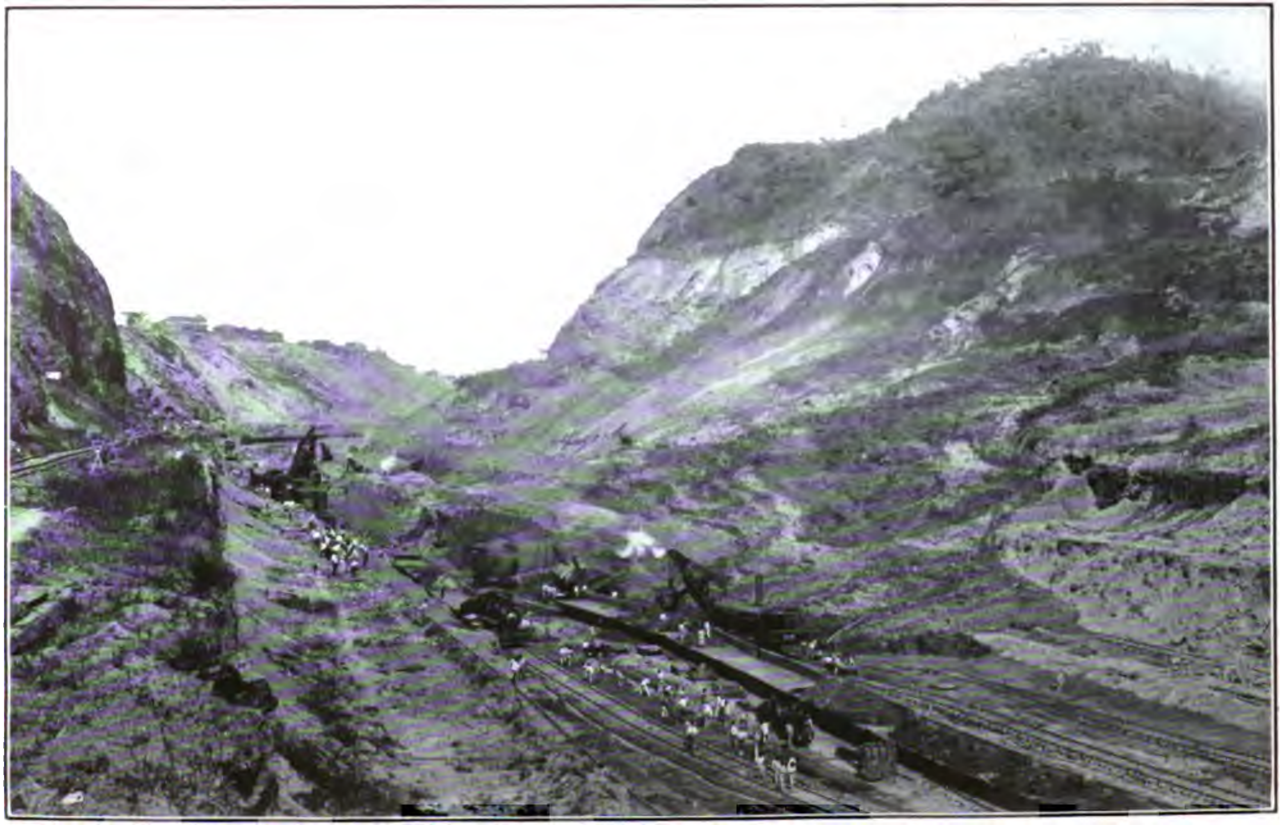
Among slides of this type may be classed that which occurred near the division office, Empire. It was due to a fairly large fault which trends across the canal and has considerably sheared the rocks on both sides of the cut. Because of the higher slope more sliding occurred on the west than on the other side (Pl. XXIX). Here the crushed basaltic rock which forms the ridge on which the office stands (building near left side of Pl. XXIX) gradually slid on a steep, slickensided slope of clay and sheared rock, letting about 278,000 cubic yards of material into the cut.

The fault-zone type of slide, unlike the others, occurs in rocks strong enough to stand at a steep slope except that large blocks of them are weakened by diagonal canalward sloping faults, which leave overhanging parts of large rock masses resting insecurely against slippery fault-zone material. Slides of this character were not common. The only remedy for them was to lessen the slopes in the vicinity of the

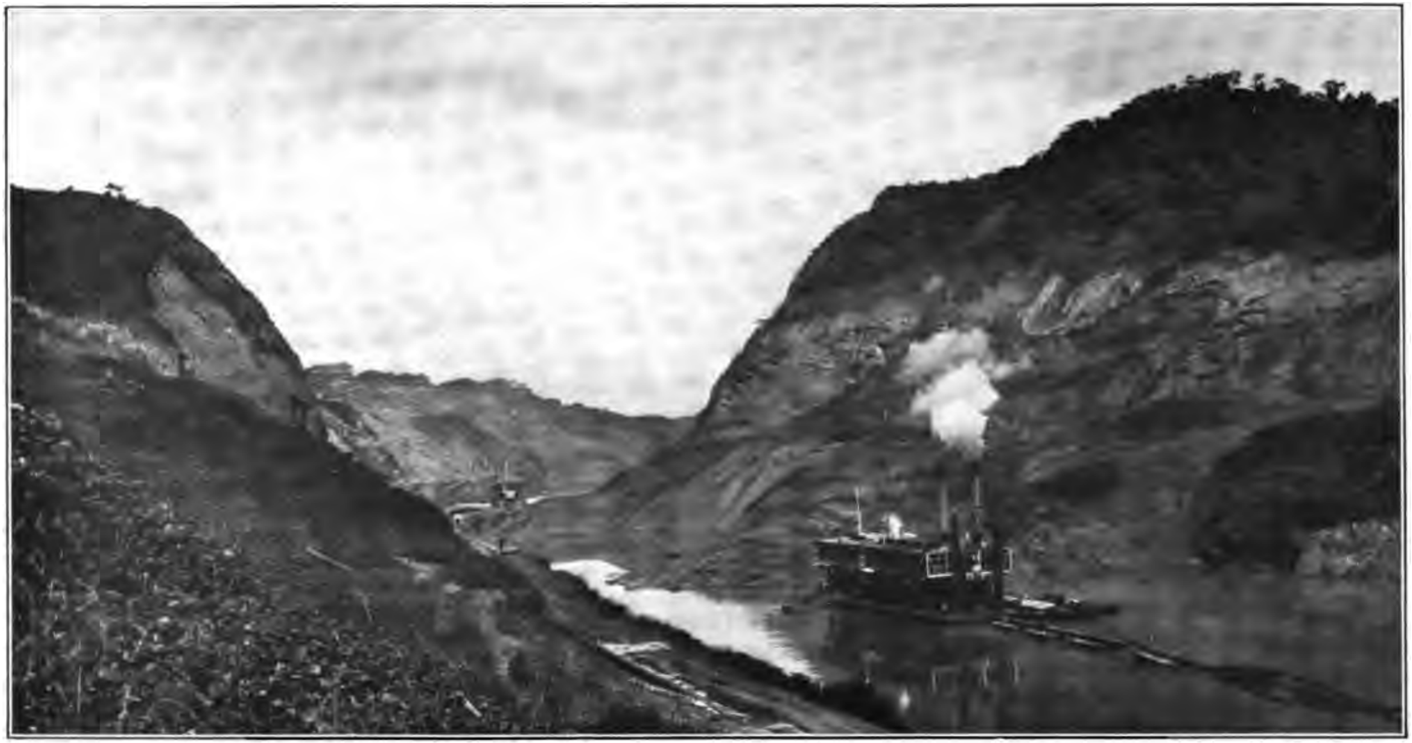


SLIDE NEAR CUCURACHA VILLAGE, EAST SIDE OF CULEBRA CUT.

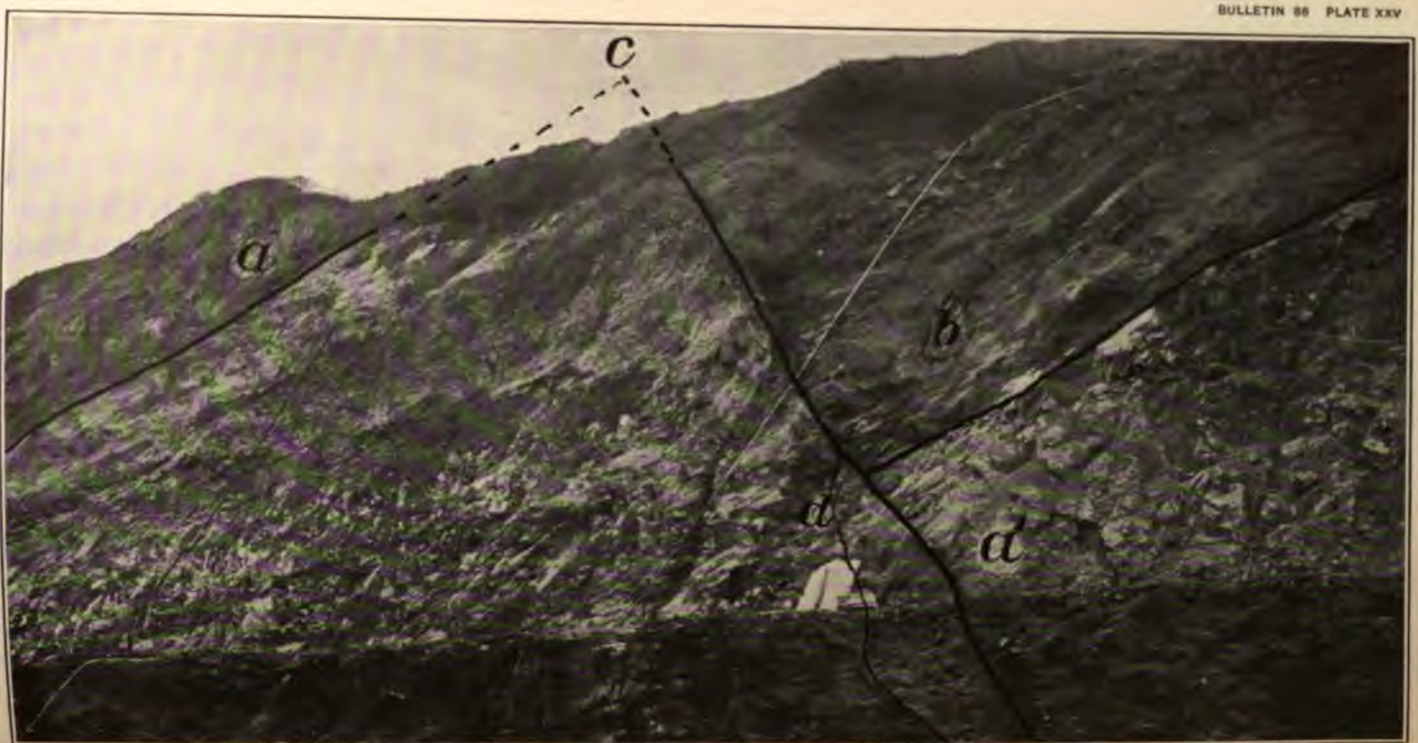
Slide included 231,000 cubic yards.



CUCURACHA SLIDE, LOOKING NORTH FROM WEST BANK, SOUTH OF CONTRACTORS HILL, FEBRUARY 7, 1913. NOTE STEAM SHOVEL TILTED OVER BY SLIDE.



DREDGES WORKING ON CUCURACHA SLIDE AFTER THE WATERS HAD BEEN TURNED INTO CULEBRA CUT.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